Bulletin 556
BUREAU OF MINES

MINERAL FACTS AND PROBLEMS

By the Staff of the Bureau of Mines

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DEPARTMENT OF THE INTERIOR

March 3, 1849

UNITED STATES GOVERNMENT PRINTING OFFICE, WASHINGTON : 1956

UNITED STATES DEPARTMENT OF THE INTERIOR
Douglas McKay, Secretary

BUREAU OF MINES
J. J. Forbes, Director

For sale by the Superintendent of Documents, U. S. Government Printing Office
Washington 25, D. C. - Price ($5.75 cloth)

FOREWORD

Analysis of known facts is a prerequisite to successful research. Once all available information relative to a given research project is obtained and carefully analyzed and evaluated, we can delimit our objectives, define our problems, and direct our investigations into the most productive channels.

The Bureau of Mines attempts to delimit the objectives of its research on metals, minerals, and mineral fuels in essentially the same way that a commercial enterprise does. A commercial firm, seeking to improve its competitive position in a single commodity, tries to gain a comprehensive understanding of the technologic and economic features of related commodities and their allied industries. The Bureau of Mines, charged with the development of ways to improve supplies of mineral raw materials to serve the broad public interest, does the same. However, because the ultimate aim of a Federal agency is promotion of the national welfare, the Bureau of Mines must weigh the research needs and potentialities of each mineral commodity against those of all others and against the requirements of national security and our dynamic economy.

In 1952 the Bureau of Mines began programing research under a formal system that has two basic requirements: (1) Current work is reviewed and evaluated frequently; and (2) new projects are undertaken only after careful appraisal to make sure that they will contribute as much as possible to the solution of specific problems.

These problems are defined in program statements that have been drawn up for each mineral commodity. The statements, revised periodically to reflect the rapidly changing positions of individual commodities in relation to national demands, provide broad basic and background information from which ideas for research can be generated and evaluated.

As originally conceived, the commodity statements were intended for use only by Bureau of Mines personnel in discharging responsibilities delegated to the Bureau by the Congress. Soon, however, many requests were received for the unpublished statements, and it became clear that a volume containing abridged versions of all of them would be valuable.

Such a volume should be useful to scientists, engineers, and executives who frequently must obtain information on mineral commodities but who lack time for tedious searching through scattered and unfamiliar references. This bulletin is intended to meet such a need. However, no attempt has been made to compile a comprehensive volume, as the resulting publication would have been unmanageably bulky.

In preparing the bulletin, new drafts of the commodity statements were sent to selected critics in industry and in Federal and private research institutions for review and comment. Over 500 such critics have contributed their ideas. The Bureau of Mines wishes to extend its gratitude to these eminently qualified specialists, who, by the devotion of much time and effort, have helped to make this volume possible.

PAUL ZINNER,
Assistant Director for Programs.

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Mineral Facts and Problems

INTRODUCTION

By

Charles W. Merrill,[1] Louis C. McCabe,[2] and R. A. Cattell[3]

AN ADEQUATE, dependable, and continuing supply of raw materials is indispensable to the United States and its industries in meeting the needs of an expanding population, a rising standard of living, and the national defense. Within the Federal Government the primary responsibility for appraising the adequacy of the country's mineral supply and of making appropriate recommendations within the executive branch and to the Congress has been placed in the United States Department of the Interior.

In fulfillment of its role in governmental responsibility for raw-material supplies, the Bureau of Mines has analyzed the factors in current and future mineral supplies and has developed 85 statements for specific commodities that assess the national position and problems for these metals, minerals, and mineral fuels. These statements are based on the assumption that production of materials in the United States is to be performed by industry in accordance with the Nation's devotion to a private-enterprise economy.

The role of Government is to be helpful to industry in furthering long-range research projects where the hope of immediate commercial operation has not been attractive to industry. This lays the groundwork for later development by private industry. In addition, Government has the responsibility of providing for the national security and of assisting industry, within the scope of its authority, in solving short-term problems. This entails a continuing review of the national position in every material of strategic or domestic importance and dissemination of comprehensive statistical data to other Government agencies and to private industry.

The availability of governmental data encourages private undertakings in areas that might not otherwise be developed. It is clear that industry tends to devote a major share of its efforts to the pressing problems of the moment; therefore Government action can render its greatest aid in looking ahead, anticipating the needs of the future, and planning its activities to allow for the maximum contingency in raw-materials requirements of the Nation as a whole. Many problems obviously call for collaboration and joint effort between private industry and Government.

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[1] Chief, Division of Minerals.

[2] Chief, Division of Solid Fuels.

[3] Chief, Division of Petroleum.

OBJECTIVES

Each of the individual commodity statements making up the bulk of this volume culminates in a section entitled "Problems." These are normal objectives of research and investigation, both technologic and economic, and fit into a pattern of broad major objectives in which both industry and Government are interested. These objectives for domestic mineral resources including fuels may be enumerated as: (1) Wise production and utilization of United States resources; (2) discovery and development of new sources of supply; (3) maintenance of reserves and stocks at adequate levels; (4) fostering of a productive capacity large and flexible enough to exploit effectively the available resources to the full extent of foreseen requirements; and (5) assurance of access to foreign supply to supplement domestic output where necessary. In addition, Government and domestic industry have an interest in foreign resources paralleling that in domestic resources, though it is usually less acute.

Further guidance as to broad objectives is to be found in the following seven points:

1. APPRAISAL OF MINERAL POSITION

Agencies of Government charged with the responsibility of conducting the country's foreign affairs, which include fostering its international trade, as well as those agencies charged with the procurement of materials for defense and internal development, require information on the Nation's mineral position to carry out their assigned functions. Most well-conceived Government action, when traced to its source, will be found to spring from the work of one or many research workers. They must supply many of the data used in formulating sound policies. The Congress likewise depends on objective, authentic information so that its legislative decisions may be based on knowledge and facts. It cannot exercise sound judgment or take proper action unless provided with comprehensive data.

2. RESEARCH

The ever-changing mineral needs of industry and defense, supplied by sources subject to depletion by exploitation and replenishment by discovery, require that the results of successful scientific and technologic research be applied aggressively and unremittingly to the mineral-supply situation. It has been recognized that wise and efficient use of natural resources depends to a large measure on advances in technology. It is also evident that the Government has a unique role in the coordination and integration of research in this field.

Research in metallic or nonmetallic minerals, coal, petroleum, or natural gas may be undertaken separately, but it must be recognized that all have problems in common and the scope and magnitude of effort in each field must be guided by their interrelationship. It is evident that industry will not or cannot use this approach. The requirement, then, is a unified program in order to have an effective one. Problems of interfuel competition and their effects on energy required to process underdeveloped minerals economically offer an example of the need for integrated, coordinated research efforts, guided by sound economic evaluation. Out of this recognition of common need will come an effective approach to minerals conservation.

As regards mineral resources, aggressive research—from basic science through applied technology—must characterize all stages, first in determining where and how best to explore and then on through exploration, development, production, refining, processing, utilization, markets, and reclamation.

As regards nonfuel minerals, there are need and opportunity for research on exploration devices and techniques, in development drilling, and for physical research on new materials. Equally urgent is the need for effective research into the extensive area of mining technology (mining methods, mechanization, transportation, rock breaking, loading, and integration of operations); and of milling, preparation, and refining technology (crushing and grinding, dry and wet concentration, jigs, dense-medium processes, pneumatic methods, flowsheets, and treatment of concentrates and tailings).

Metal extraction embraces smelting (furnace construction, temperature control, blast control, slag and melt composition); hydrometallurgical extraction or chemical processing (leaching, filtering, and electrolytic and chemical precipitation); nodulizing and briquetting; refining; and combinations of all these techniques under special circumstances. Similar processing is involved in many minerals, particularly those of the nonmetals group. In all these areas there is special need for research in making low-grade and refractory ore deposits available for use. Recovery of scrap and its preparation for reuse are highly important phases of conservation of resources.

In mineral-fuels research, improved utilization of coal may be brought about by developing more economical mining methods or by underground gasification. Whereas combustion and high-temperature coking now provide the main markets for coal, perfection of processes under study—some of which have been or are now being used on an industrial scale—may create considerable demand for this abundant mineral as a source of organic chemicals, fertilizer, high-B.t.u. gas, liquid fuels, oils, fats, and waxes. Among these processes are direct hydrogenation, low-temperature carbonization, and gasification with subsequent utilization in the methanation reaction or the ammonia, the oxo, or the Fischer-Tropsch synthesis.

In petroleum and natural-gas research, there are equal and parallel needs that Government and industry can fulfill cooperatively. Improved drilling equipment and techniques will continue to offset costs of deeper drilling and rising ratios of drilling to discovery. Better methods of production (pressure maintenance, water flooding, in situ combustion, use of detergents, unitization) will make available material now discounted in reserve estimates. Separation, identification, and characterization of the component parts of crude petroleum and natural gas will provide a sound basis for meeting changes in requirements for the end uses of these materials.

Continued assistance to. the multitude of small operators who in a large measure set the pattern for the production methods of the whole industry will result in industrywide improvements in production methods and will bring the laboratory products closer to application problems, resulting in more efficient use of these resources.

Collection, classification, and dissemination of mineral statistics, properly coordinated with pertinent science and technology, are highly specialized. Such data are a necessity in guiding research by evaluating and interpreting the mineral position on a world-wide scale with a higher degree of precision, coverage, and detail than heretofore and in estimating both domestic and worldwide mineral potentials.

To be most useful and in the greatest public interest, Federal research usually should be integrated, not competitive, with that by others and should be predicated on the best judgment of national needs. Although here and there and from time to time there will be necessity for short-term investigations, Federal research should attempt to anticipate the future and be directed toward long-term objectives much more than privately financed work.

3. MINERAL-INDUSTRY STUDIES

Increased mineral utilization is fostered by mineral-industry studies that point out commercial opportunities resulting from scientific and technical discoveries affecting minerals and from economic advances in other fields. Results of publicly financed research are usually published. Privately financed research of the same type may be published and made available for general use, but more frequently is used as a basis for directing the activities of its sponsor. Discovery of ore bodies, proof of sizable reserves, and development of new or more efficient extraction methods, either in mining or metallurgy, coupled with new or improved sources of energy from mineral fuels to convert such ores to useful products, present consumers with new opportunities.

On the other hand, increasing populations, advancing industrialization, new applications, and other elements of demand may call for a review of latent or partly developed resources as a new or expanded source of supply. Mineral-industry studies will attack such problems of supply-demand adjustment on a national basis and on regional bases where appropriate. Studies of a conservational nature that point the way to intelligent use of latent resources call for proper emphasis.

4. DEVELOPMENT OF SUBMARGINAL RESOURCES

Various deposits in the United States contain such a small percentage of the desired metal or mineral or are so difficult to treat with present methods that development would not be profitable. Such deposits are referred to as economically submarginal. If technologic history can be accepted as a guide, some of these deposits eventually will be mined when the need becomes great enough. Domestic submarginal deposits of some mineral commodities of which imports now constitute much of the supply are so large that the United States could become completely self-sufficient for a long time if cheaper production methods could be devised or if the price of the commodity were to advance.

It can be assumed safely that the United States contains many deposits of submarginal material not yet utilized or discovered. There has been little commercial incentive to search for them or, if they are found in the course of other activities, to develop them; however, demonstration of very large reserves of material, even if presumably of submarginal quality, is a powerful incentive to private enterprise to attempt commercial development. The early phases of research in submarginal resources normally fall to Government, but privately financed organizations frequently undertake such development long before profitable operation can be assured. An example of this type of resource is the oil shale in Colorado, assaying 25 gallons or more per ton, that would yield an estimated 125 billion barrels of oil if all of it could be processed.

5. POSSIBLE NEW OR WIDER USES FOR ABUNDANT RESOURCES

The United States also has large deposits of mineral raw materials for which comparatively little use has been found, as exemplified before World War II by sea water, brines, and magnesite now used as sources of magnesium. Means that would expand the use of the metals or minerals in such deposits, in commodities and in processes for which materials in short supply are required today, could relieve a serious drain on the latter. Outstanding examples of plentiful but unused resources with promising possibilities are high-aluminum clay or anorthosite as raw materials for production of aluminum. Furthermore, if an industrial application for an abundant but little-used material can be devised and demonstrated, the foundation will be laid for new industrial enterprises. Titanium and helium are examples in this category.

6. SUBSTITUTES

Development of substitutes is one means whereby the mineral position of the United States can be improved. Metals, minerals, and mineral fuels in ample supply can be substituted, under certain conditions, for those in scant supply. The maintenance of the German war machine during World War II on liquid fuels synthesized from coal is a spectacular example of substitution in time of need. In the United States molybdenum (of which there is a large domestic supply) has been substituted for tungsten, much of which is imported, in many alloy steels. Glass has conserved tin used in food containers. The mineral position of the United States will be improved to the extent that its abundant substitutes are used.

7. CONSERVATION

Basically, conservation entails wise use—not hoarding. Wise use in the mineral field in turn entails the following: High eventual recovery of material from the ground and of the valuable constituents of this material; recovery at the time needed; fabrication or other preparation for use with minimum waste or loss of the valuable material; utilization or storage of coproducts and byproducts; application to purposes of high economic utility; and utilization of substitutes, where available, for other important materials that either are in limited reserve or have singular use characteristics not common to other minerals.

COMMODITY STATEMENTS

In examining mineral problems, the job has been subdivided on a commodity basis, and a chapter has been written for almost every mineral commodity of commercial importance. The result is a series of 85 commodity statements. In addition, a special statement on mineral research not associated with any particular commodity has been developed to include those fields of activity that affect many commodities in ways that make the single-commodity approach impracticable.

The commodity statements that make up the bulk of this volume conform to a generalized outline arranged under the following five headings, but with deviations and omissions the particular commodities require:

Summary.
Background.
Outlook.
Problems.
Bibliography.

The commodity statements are intended to present the current situation for each commodity. It must be kept in mind, however, that conditions may change radically and unexpectedly and that judgments and forecasts are at best imperfect. Nevertheless, for the time being the statements provide a current review of the situation and carry the discussion to the point of setting forth the outstanding problems in need of study. The following paragraphs are a brief presentation of the points covered in each commodity statement.

BACKGROUND

SIZE AND ORGANIZATION OF INDUSTRY

After a Summary, each commodity statement presents in a Background section a review of the commodity and the industry based on it, giving its history, industrial organization, and other pertinent information. Some commodities, such as gallium, have found only very limited economic uses, whereas others, like coal, iron, petroleum, and portland cement, are the very foundations of modern industry. Commercial prominence, however, is only one criterion for measuring the importance of commodity problems. Some commodities which so far have found little application in industry offer great inducements to use-research because of their abundance. On the other hand, other materials, though widely used, have such limited reserves that search for substitutes is indicated. In addition, military applications, which are less limited by costs, may have major significance, as, for example, titanium, helium, and shale-oil production and use.

GEOGRAPHIC DISTRIBUTION

The geographic distribution of the principal industries that consume each commodity is set forth. Some commodities are widely distributed in nature as well as in consumption; some, because of their low unit prices, are normally mined and consumed locally, whereas others, because of their limited economic occurrence and specialized markets, are important items in international trade. Commodities that are bulky offer special problems, both as to local and national supply; those with defense uses in which the known supplies are largely foreign present serious strategic problems.

TECHNOLOGY

Many commodities face their principal problems in the technologic field and sometimes in a limited part of it. Geology, for example, plays a major part when known reserves and resources are very small compared to current and prospective demand. Mining technology offers important possibilities where very low grade ores must be exploited to meet requirements or where special problems, such as roof support, pumping, and blasthole drilling, must be solved if mineral deposits are to be exploited economically.

Ore dressing and hydrometallurgy are important fields for research where stubborn ores are yielding low recoveries or where ore treatment has not emerged fully from hand methods, as, for example, in the beneficiation of beryl or mica. Smelting, refining, and remelting are of special interest where recoveries are low, the purity of products is unsatisfactory, or possible byproducts are going to waste. The smelting and refining field also offers unusual opportunities in recovering commodities from secondary sources.

Coal research is urgently needed to make better use of this major fuel resource. Mining methods should be improved to increase recovery, and preparation methods should be developed to upgrade marginal coals to coking quality. Low-temperature carbonization of coal should be explored to supplement declining reserves of low-volatile coking coals and to supply sources of new chemical products. Likewise, the constantly increasing demands for natural gas will soon require supplementation with high-B.t.u. gas from coal.

In the petroleum and natural-gas industry technology has been the stabilizing influence in the cost of exploration, drilling, production, transportation, and refining. Improved tools, techniques, and methods have provided increased drilling speeds and more efficient recovery, thereby stabilizing the unit cost. Lower costs of products and better utilization have been attained through vastly improved methods of refining. All these technologic developments have strengthened the energy position of the Nation.

BYPRODUCTS AND COPRODUCTS

As industrialization progresses and wider markets present themselves, the treatment of many commodities calls for reexamination, because of past and current waste of potential byproducts and coproducts. Such investigations not only lead to lowering of costs at established sources of supply but in some instances present major opportunities for utilizing latent resources that could not be commercial under older and less efficient technology, or that may be used as the source of new products and new industries.

SUBSTITUTES AND SYNTHETICS

The question of substitutes presents itself in considering almost every commodity, both as to the possibility that it will invade the market as a substitute for materials already established in use and as to the possibility that some other commodity will displace that in question in one of its established uses. Although immediate decisions in substitution depend on availability, utility, and price, problems may revolve about shifting reliance toward more abundant resources and applying commodities with limited reserves to uses of highest utility. In addition to the simpler forms of substitution—such as aluminum for copper in electric power-transmission lines—laboratory research has developed some promising synthetics, such as quartz crystals precipitated artificially, synthetic mica, and cryolite. The prospective production of synthetic liquid fuels from natural gas, oil shale, and coal is a major example of the importance of substitutes in time of need. These and other substitutes or alternative products have had or will have great significance in relieving the pressure on certain strategic and critical materials in great demand or in limited supply.

SECONDARY RECOVERY

The term "secondary recovery" is used here to denote reuse of materials rather than the reworking of deposits after initial production. The possibilities of increasing the total service of some commodities by reuse pose a major problem. Some minerals and uses lend themselves to much higher percentages of reclamation than others. For example, the lead consumed in storage batteries returns for reuse in 2 or $2^1/_2$ years, with an overall recovery of about 85 percent, whereas the same metal consumed in making tetraethyl fluid is completely dissipated. The fostering of a high return of minerals for reuse and the limiting of degradation in the reclamation processes are very important to conservation.

RESERVES

Available data on reserves have been summarized in the commodity statements. For some commodities the proved reserves are so vast and accessible that little need be said beyond calling attention to this favorable condition, as, for example, the reserves of magnesium in sea water. On the other hand, proved reserves of some commodities are negligible, although substantial resources are certain to exist. Sheet mica is an example. Because discovery of deposits is largely fortuitous, the mica is mined and marketed as the deposits are developed, and estimates of their magnitude cannot be projected very far beyond the development.

In addition, the magnitude of a reserve changes as prices and costs fluctuate, and the trend of the ratio of proved reserves to consumption frequently is more significant than the absolute magnitude of reserves at any one time. Even where the proved reserves are large enough to permit production at current rates only for a very short period, there may be little reason for apprehension if this narrow margin of reserves has been maintained over a long period. Published reports on proved reserves of some minerals—notably petroleum and natural gas—are limited to the quantities that can be recovered from the deposits already found under existing economic and operating conditions. They reflect the composite policy of the industry with respect to quantities that should be proved in advance of production, rather than the total quantities that can be found and produced for future use.

Data on reserves of some minerals are spotty; in many instances too little is known to provide a satisfactory basis for long-range forecasts or even to determine trends. Moreover, many of the known data are closely held company secrets unavailable for general analytical use.

USES

Commodity problems depend largely on commodity uses. Whereas consumption for some uses is declining, for others it is expanding much more rapidly than the average rate for industrial raw materials. Of particular significance are the commodity demands of new developments—such as electronics, for example—where new devices require substantial quantities of material for which earlier markets had been only of laboratory proportions. The demand for germanium for transistors is a case in point, although partial substitution by silicon has relieved apprehensions concerning supply. The demand for helium in industrial operations is added evidence that use may create new and rapid demands on sources of supply.

The accumulation of data on use patterns is of major importance in mineral-resource development, whether applied on a world, national, or regional scale. Many anomalies exist in the present distribution of mineral consumers and their sources of supply. There are interesting possibilities for expanding demand near the sources of supply and of expanding supply near the centers of fabrication and ultimate consumption. Use-pattern information is important to those charged with aiding underdeveloped foreign areas.

STATISTICS

The statistical information available for any commodity tends to become increasingly vague as investigation is carried backward into earlier periods. For many commodities, even the current statistical coverage leaves much to be desired; inadequacies are found both as to fields of coverage (such as production, transportation, processing, and consumption) and geographic areas (such as mining districts or industrial centers) in the United States and abroad.

The improvement of statistics, however, presents a difficult problem because those who are called upon to assume an added reporting burden must be persuaded that a more comprehensive or precise product will justify the extra work. For the most part, the statistical data available are vastly more detailed than the significant totals that can be presented in the space available for commodity statements. The statistical presentations on output ordinarily are confined to mine production for relatively recent years and for the larger political units. Data on consumption also are confined to those for recent years and for major applications. Other data on foreign trade, stocks, prices, costs, and the like are included to the extent that they seem significant.

STRATEGIC CONSIDERATIONS

In view of the fact that national security is a prime consideration for almost every mineral commodity, the degree of United States self-sufficiency is significant. It is also recognized that a healthy domestic mining industry is a powerful deterrent to cartel practices by foreign suppliers, who, for the most part, are much less restrained by antimonopoly legislation than are domestic sellers. Self-sufficiency, however, is not put forward as a goal in itself, and the importance of mutuality in foreign trade and of seeking raw materials for the United States at the lowest real cost are given full weight.

Important considerations that arise from the inability of domestic resources to supply a war economy include the need for the accumulation and storage of stockpiles of such raw materials as otherwise would be in deficient supply if international trade were to be restricted by enemy action. Raw materials of mineral origin are by far the most important both in quantity and value in the National Strategic Stockpile program. In addition to the protection afforded by stockpile accumulation, reliance is placed on the development and maintenance of a healthy domestic mining industry and the promotion of mineral-resource development and friendly relationships, particularly with foreign nations that appear most likely to be available as sources of supply under war conditions.

Strategic considerations also include problems of mine production and processing capacity, vulnerability due to concentration of facilities, and availability of transportation, power, and skilled labor forces. Commercially tested and developed processes for utilizing subgrade and offgrade materials under emergency conditions are important considerations for some commodities.

DEFENSE PROGRAMS

Under current conditions, with the high degree of international tension that has persisted since the end of World War II, all commodity statements consider the extraordinary defense efforts made or sponsored by the Federal Government. It is recognized that many commodity statements have a major defense significance.

PRICES AND COSTS

The records of prices and costs and the trends to be observed in them often have great significance when compared with similar data for competitive raw materials. In instances where commodities have been in production and use over long periods and where use patterns are firmly established, prices and costs tend to move in close consonance with the rises and falls in general wholesale price and cost-of-living indexes. On the other hand, some mineral commodities that have not been widely accepted in industry frequently offer very attractive promotional possibilities if evidence can be assembled showing that a major increase in production would permit a major decrease in cost or that a major decrease in price would be reflected by a major expansion in demand.

TAXES

The problem of taxation impinges on all industrial activities, but there are some substantial and significant differences in the incidence of taxes on different minerals and on the same minerals in different political subdivisions. The provision for percentage depletion under Internal Revenue legislation varies with commodities, and major differences occur between States in the imposition of severance and other forms of taxation.

TARIFFS

Some commodities enter the United States on the free list, whereas others are subject to tariff rates that assure an almost exclusive market for domestic producers. The selection of tariff rates that will interfere as little as practicable with the flow of raw materials from the source of supply cheapest to United States consumers and yet will provide enough protection to maintain a healthy domestic mining industry, where domestic resources permit it, requires careful revaluation of all relevant factors, many of which are not easy to appraise.

TRANSPORTATION

The cost of transportation from mine to market is a controlling factor in most bulky mineral products. Materials like sand and gravel are in almost all instances produced very close to the points of consumption, whereas the transportation costs per unit of value for commodities like the precious metals or gem stones are a negligible part of the sales price to the ultimate consumer. The cost of transporting coal, on the other hand, is a very significant factor, since the preponderance of coal moves long distances by rail at rates that, as an average, add nearly 70 percent to the cost of the coal at the mines. In addition to the general problems of transportation, specific technical problems of modes of haulage have a significant effect in the economics of many commodities, both in movement from mine to market and in handling the materials in production.

CONSERVATION PRACTICES

Conservational considerations are given major weight in all statements. The application of limited high-grade material to the more specialized purposes, the use of scarce materials only where substitutes are not available, the stockpiling of uncommercial residues that promise to yield valuable products with improving technology, early development of new extraction processes for the treatment of submarginal and refractory ores, and improvements in product yield are fostered as conservational measures.

RESEARCH

In presenting the background for each commodity, an effort is made to cover, insofar as information has been easily available, the history and current status of major research by the Bureau of Mines, by other Government agencies, by other institutions, and by industry. The importance of relying or prevailing on others than Government to carry as much as possible of the research expense is emphasized, and those areas of research in which the Bureau of Mines takes the lead are set forth, with their interrelationships to research activities of others where these are known to be significant.

OUTLOOK

SIGNIFICANT TRENDS

A third section of each commodity statement presents the outlook for the commodity. Significant trends are analyzed and their effects on the industry forecast. Major national trends used include records and projections, where available, of population and labor force (fig. 1), gross national product (fig. 2), construction activity (fig. 3), industrial production (fig. 4), price indexes (fig. 5), and energy

TABLE 1.—*Population and total labor force of continental United States*

[Millions]

Year	Population [1]		Labor force [2]
1790	3.9		----------
1800	5.3		----------
1810	7.2		----------
1820	9.6		----------
1830	12.9		----------
1840	17.1		----------
1850	23.2		----------
1860	31.4		----------
1870	39.8		----------
1880	50.2		----------
1890	62.9		----------
1900	76.1		----------
1901	77.6		----------
1902	79.2		----------
1903	80.6		----------
1904	82.2		----------
1905	83.8		----------
1906	85.4		----------
1907	87.0		----------
1909	90.5		----------
1910	92.4		38.1
1911	93.9		38.7
1912	95.3		39.1
1913	97.2		39.5
1914	99.1		39.8
1915	100.5		40.1
1916	102.0		40.3
1917	103.3		42.7
1918	103.2		44.2
1919	104.5		42.0
1920	106.5		41.9
1921	108.5		42.4
1922	110.1		43.0
1923	112.0		43.8
1924	114.1		44.5
1925	115.8		45.0
1926	117.4		46.0
1927	119.0		47.0
1928	120.5		48.0
1929	121.8		49.4
1930	123.1		50.1
1931	124.0		50.7
1932	124.8		51.3
1933	125.6		51.8
1934	126.4		52.5
1935	127.2		53.1
1936	128.1		53.7
1937	128.8		54.3
1938	129.8		55.0
1939	130.9		55.6
1940	132.1		56.2
1941	133.4		57.5
1942	134.9		60.4
1943	136.7		64.6
1944	138.4		66.0
1945	139.9		65.3
1946	141.4		61.0
1947	144.1		61.8
1948	146.6		62.9
1949	149.2		63.7
1950	151.7		64.7
1951	154.4		66.0
1952	157.0		66.6
1953	159.6		[3] 67.4
1954	162.4		[3] 67.8
	PROJECTIONS		
	Low	*High*	
1955	------	164.8	68.6
1960	173.8	177.4	73.0
1965	180.9	189.9	78.9
1970	189.1	204.2	84.8
1975	198.6	221.0	89.5

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[1] Overseas members of Armed Forces included, beginning with 1940. Bureau of Census, Current Population Reports: Ser. P–25, Nos. 78 (projections), 93, 98, and 109.

[2] Includes civilians employed and those unemployed looking for work and Armed Forces. Bureau of Census, Current Population Reports: Ser. P–50 and P–57 (projections are data for April—from Ser. P–50, No. 42—adjusted for seasonal variation from the annual average). Series also given in Chemical Economics Handbook, Statistical Abstract, and Federal Reserve Bulletin.

[3] Not strictly comparable with data for earlier years. Based on improved sample covering a larger number of areas.

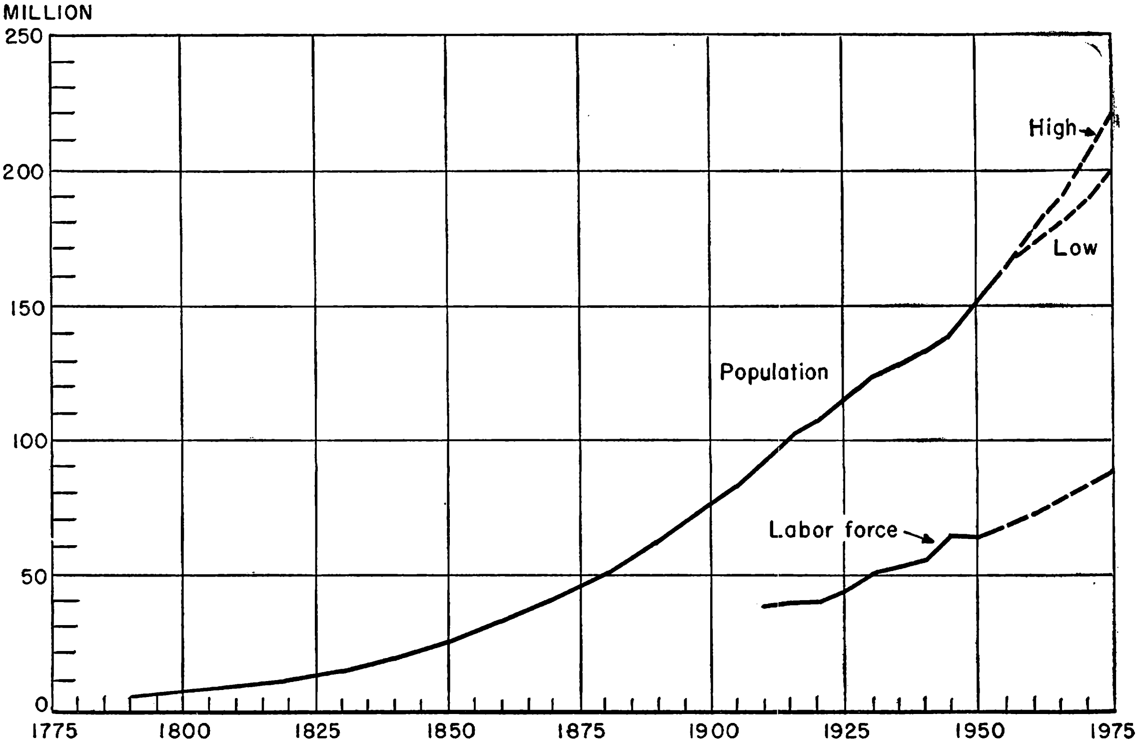

FIGURE 1.—Population and Total Labor Force of Continental United States, Million.

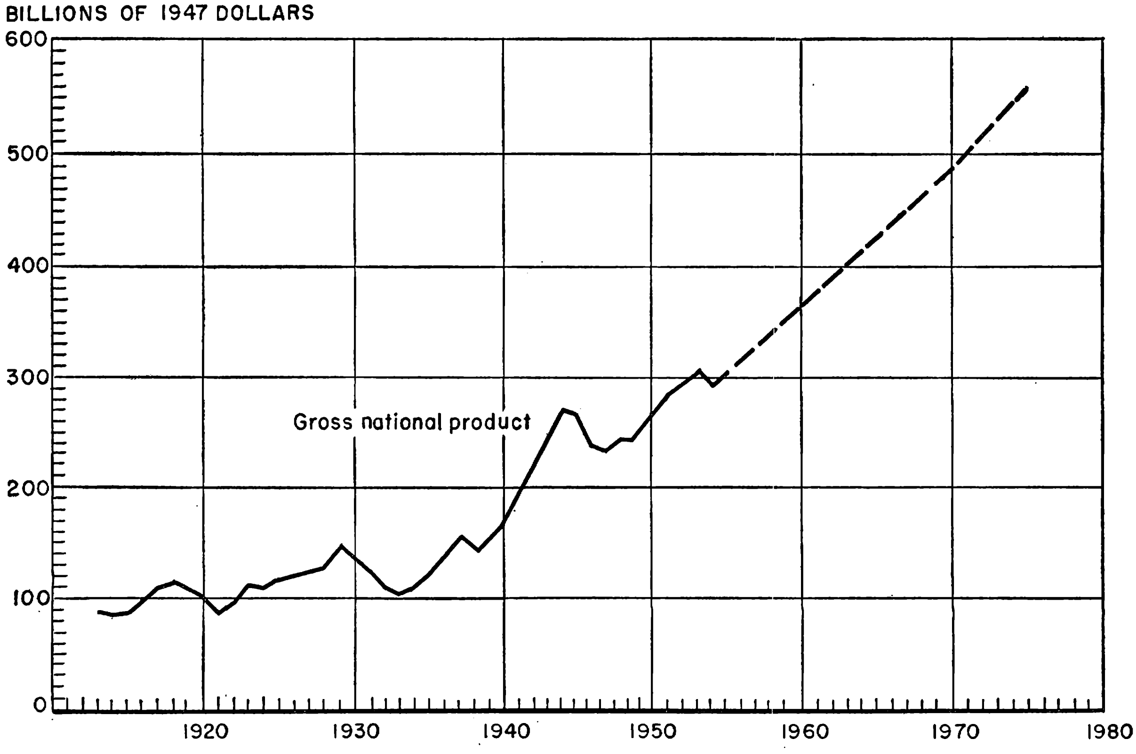

FIGURE 2.—Gross National Product for the United States, 1913–55, Billions of 1947 Dollars.

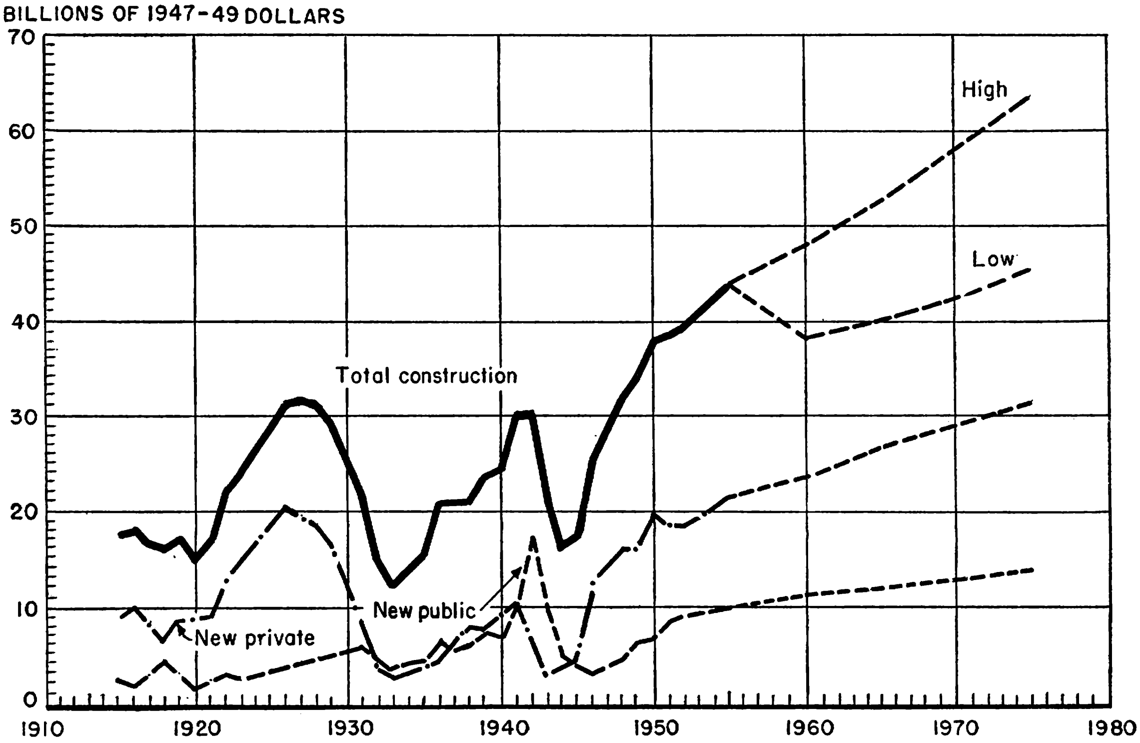

FIGURE 3.—Construction Activity in the United States, Billions of 1947–49 Dollars.

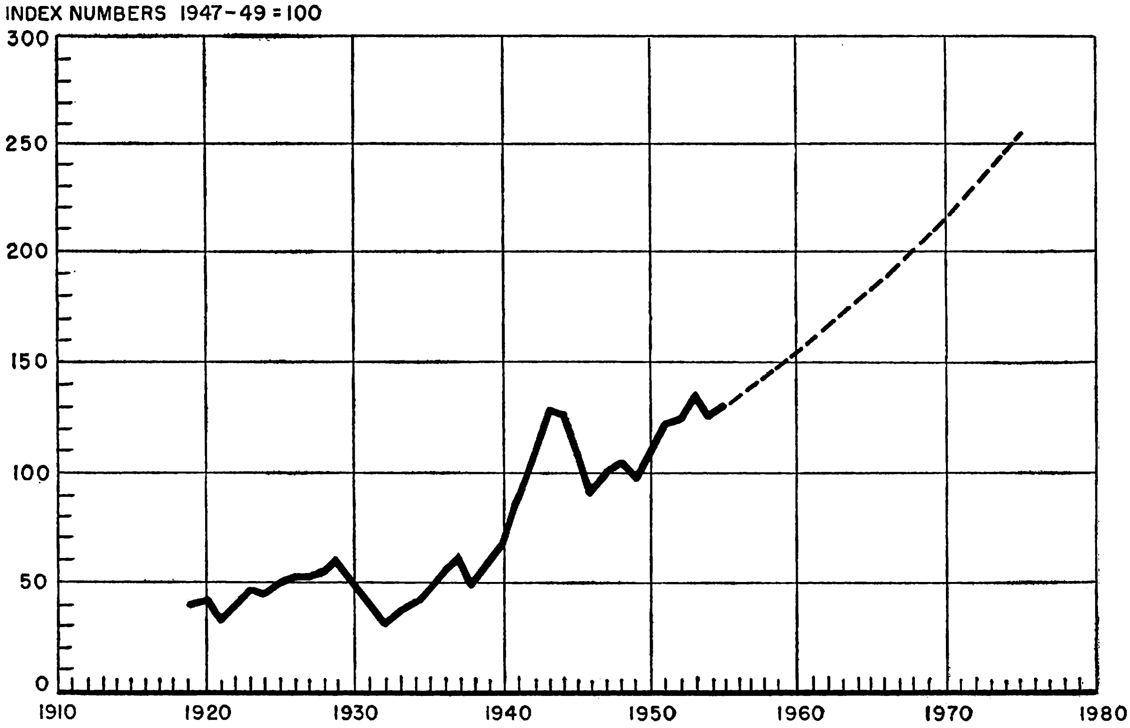

FIGURE 4.—Federal Reserve Board Index of Industrial Production; 1947–49=100.

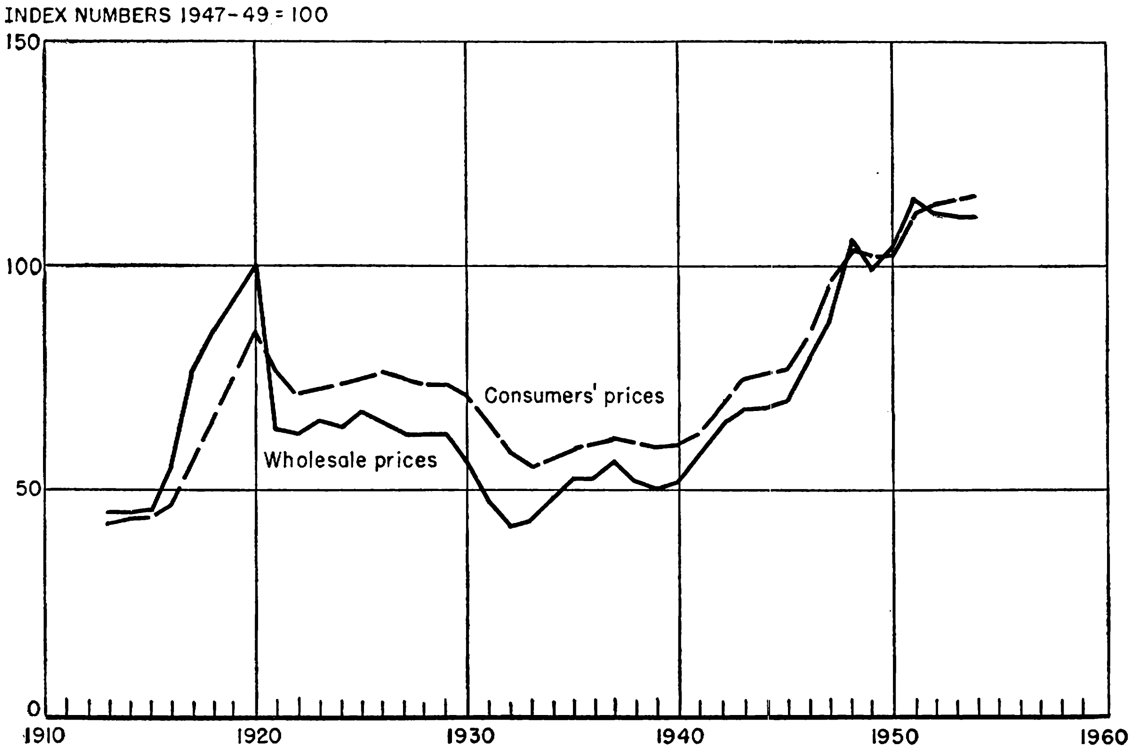

FIGURE 5.—Wholesale and Consumers' Price Indexes for the United States Computed on or Adjusted to 1947–49=100.

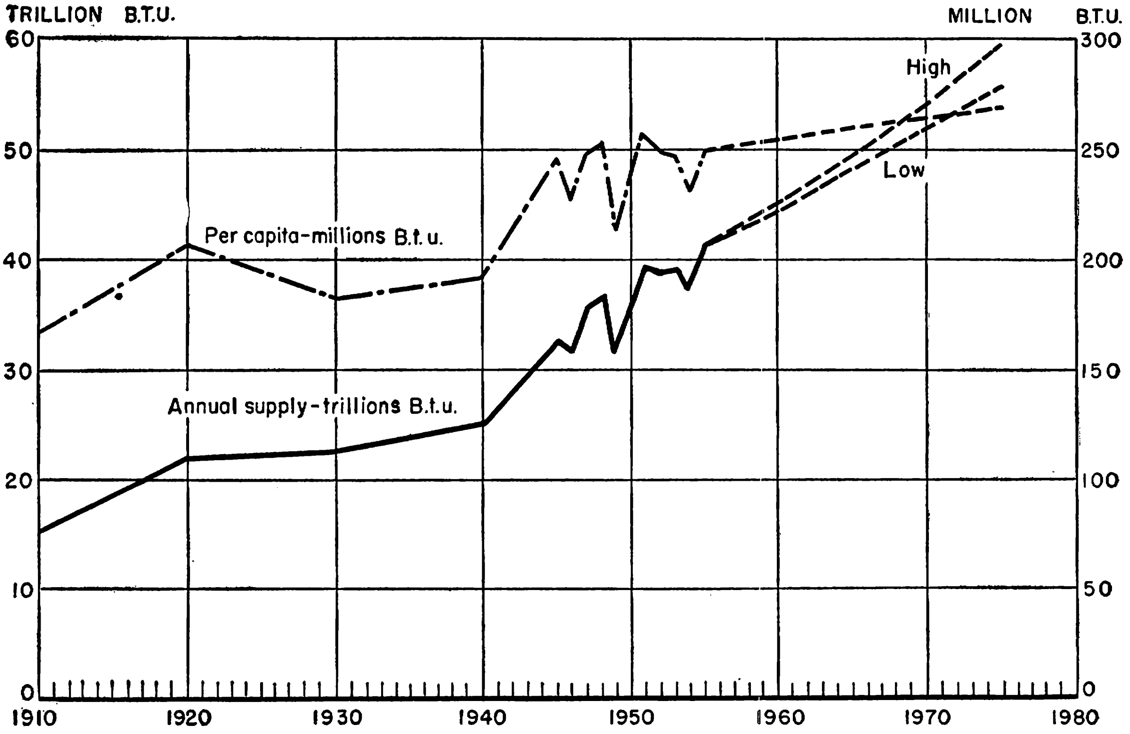

FIGURE 6.—Annual Supply of Energy From Mineral Fuels and Waterpower in the United States, as Related to Population.

TABLE 2.—*Gross national product, 1913–54* [1]

[Billion 1947 dollars]

Year	Gross national product	Year	Gross national product	Year	Gross national product	Year	Gross national product
1913	88. 5	1926	122. 3	1938	145. 9	1951	282. 9
1914	86. 9	1927	123. 0	1939	157. 5	1952	294. 2
1915	87. 4	1928	127. 2	1940	171. 6	1953	306. 6
		1929 [2]	135. 9			1954 [3]	297. 0
1916	100. 0			1941	198. 2		
1917	110. 1	1929	149. 3	1942	223. 6	PROJECTION [4]	
1918	116. 6	1930	135. 2	1943	248. 9		
1919	111. 1	1931	126. 6	1944	268. 2	1955	305
1920	102. 6	1932	107. 6	1945	263. 1	1960	365
		1933	103. 7			1965	423
1921	87. 1	1934	113. 4	1946	233. 8	1970	490
1922	99. 1	1935	127. 8	1947	232. 2	1975	570
1923	112. 1			1948	243. 9		
1924	110. 8	1936	142. 5	1949	241. 5		
1925	118. 3	1937	153. 5	1950	264. 7		

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[1] For 1913–28, unpublished gross national product data estimated by U. S. Department of Commerce (reported by Stanford Research Institute in the Chemical Economics Handbook) adjusted to 1947 dollars by Office of Chief Economist, Bureau of Mines, using consumer price index converted to 1947 base. For 1929–53 (except as indicated in footnote 2), data are as reported in the 1954 National Income Supplement to the Survey of Current Business.

[2] Current dollars total, as reported in 1954 National Income Supplement, adjusted by Office of Chief Economist, Bureau of Mines, in same manner as data for 1913–28, and generally comparable to adjusted values shown here for those years.

[3] Preliminary estimate, Council of Economic Advisors, in Economic Report of the President, January 20, 1955, p. 140.

[4] Series drawn and adjusted to 1947 prices by Office of Chief Economist, Bureau of Mines, through projection for 1960 by G. Colm (National Planning Association), below, Staff, Joint Committee on the Economic Report estimate of 1965 and above 1975 figure derived from Paley Report.

TABLE 3.—*Construction activity in the United States* [1]

[Billion 1947–49 dollars]

Year	New public [2]	New private	Total construction [2] [3]
1915	2. 4	9. 1	17. 5
1916	2. 0	10. 2	18. 0
1917	3. 1	8. 7	16. 8
1918	4. 7	6. 4	16. 0
1919	3. 5	8. 6	17. 1
1920	1. 8	8. 8	15. 2
1921	2. 5	9. 1	17. 2
1922	3. 0	13. 2	22. 3
1923	2. 6	15. 3	24. 1
1924	3. 1	17. 0	26. 6
1925	3. 6	18. 9	29. 4
1926	3. 7	20. 1	31. 1
1927	4. 1	19. 7	31. 5
1928	4. 4	18. 7	30. 9
1929	4. 4	16. 4	28. 9
1930	5. 4	12. 0	25. 1
1931	5. 6	8. 4	21. 0
1932	4. 6	4. 3	15. 3
1933	3. 4	3. 2	12. 3
1934	4. 1	3. 6	13. 8
1935	4. 2	4. 8	15. 8
1936	6. 5	7. 0	21. 3
1937	5. 4	8. 2	21. 2
1938	6. 1	7. 4	21. 0
1939	7. 0	9. 0	23. 8
1940	6. 7	10. 1	24. 7

Year	New public [2]		New private		Total construction [2] [3]	
1941	10. 2		11. 5		29. 9	
1942	17. 1		5. 8		30. 4	
1943	9. 7		3. 2		20. 5	
1944	4. 8		3. 4		16. 4	
1945	3. 7		4. 8		17. 5	
1946	3. 0		12. 5		25. 9	
1947	3. 7		14. 1		28. 7	
1948	4. 6		16. 2		31. 9	
1949	6. 2		16. 0		33. 7	
1950	6. 7		19. 9		37. 7	
1951	8. 3		18. 7		38. 3	
1952	9. 2		18. 4		39. 4	
1953	9. 5		19. 4		40. 6	
1954 (pre.)	9. 6		20. 8		42. 1	
PROJECTIONS [4]						
	Low	*High*	*Low*	*High*	*Low*	*High*
1955	10. 0		21. 5		43. 5	
1960	(5)	11. 0	(5)	23. 5	38. 0	47. 5
1965	(5)	12. 0	(5)	26. 5	40. 0	52. 0
1970	(5)	13. 0	(5)	29. 0	42. 0	57. 5
1975	(5)	14. 0	(5)	31. 5	45. 0	63. 0

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[1] Except as indicated, from U. S. Department of Commerce, Construction and Building Materials Industry Report: *New construction:* 1915–53 data from May 1954 Statistical Supplement, pp. 40 and 42, 1954 data, records of Business and Defense Services Administration; *Maintenance and repair:* Value in current dollars—1915–53 from June 1954 issue, page 5; 1954, not available from BDSA, estimated—deflated by Office of Chief Economist, Bureau of Mines, to 1947–49 dollars by applying to value for each year ratio for that year between total new construction in current dollars and in 1947–49 dollars.

[2] Work-relief expenditures are included in 1933–43.

[3] Includes private and public new construction plus maintenance and repair.

[4] Projections by Office of Chief Economist, Bureau of Mines.

[5] Not considered to be subject to reasonably accurate forecast.

TABLE 4.—*Federal Reserve Board index of industrial production* [1]

[1947–49=100]

Year	Index	Year	Index	Year	Index	Year	Index
1919	39	1931	40	1943	127	PROJECTIONS [2]	
1920	41	1932	31	1944	125		
		1933	37	1945	107	1955	129
1921	31	1934	40				
1922	39	1935	47	1946	90	1960	153
1923	47			1947	100		
1924	44	1936	56	1948	104	1965	181
1925	49	1937	61	1949	97		
		1938	48	1950	112	1970	214
1926	51	1939	58				
1927	51	1940	67	1951	120	1975	252
1928	53			1952	124		
1929	59	1941	87	1953	134		
1930	49	1942	106	1954 (prel.)	125		

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[1] 1919–53 data from Federal Reserve Bulletin, vol. 39, No. 12, December 1953, p. 1324, and vol. 41, No. 1, January 1955, p. 64; 1954 data from Economic Report of the President, transmitted to the Congress, Jan. 20, 1955, p. 164.

[2] Projections by Office of the Chief Economist, Bureau of Mines, assuming 3.4-percent increase per year.

TABLE 5.—*Consumers' and wholesale price indexes* [1] *computed on or adjusted to 1947–49=100*

Year	Consumers'	Wholesale	Year	Consumers'	Wholesale
1913	42.3	45.4	1934	57.2	48.7
1914	42.9	44.3	1935	58.7	52.0
1915	43.4	45.2			
			1936	59.3	52.5
1916	46.6	55.6	1937	61.4	56.1
1917	54.8	76.4	1938	60.3	51.1
1918	64.3	85.3	1939	59.4	50.1
1919	74.0	90.1	1940	59.9	51.1
1920	85.7	100.3			
			1941	62.9	56.8
1921	76.4	63.4	1942	69.7	64.2
1922	71.6	62.8	1943	74.0	67.0
1923	72.9	65.4	1944	75.2	67.6
1924	73.1	63.8	1945	76.9	68.8
1925	75.0	67.3			
			1946	83.4	78.7
1926	75.6	65.0	1947	95.5	96.4
1927	74.2	62.0	1948	102.8	104.4
1928	73.3	62.9	1949	101.8	99.2
1929	73.3	61.9	1950	102.8	103.1
1930	71.4	56.1			
			1951	111.0	114.8
1931	65.0	47.4	1952	113.5	111.6
1932	58.4	42.1	1953	114.4	110.1
1933	55.3	42.8	1954 [2]	114.9	110.3

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[1] These are Bureau of Labor Statistics series with components available in a number of sources including Economic Almanac, Statistical Abstract, Survey of Current Business, and Monthly Labor Review.

[2] Preliminary (consumers' price index based on data through November). Source: Economic Report of the President, transmitted to the Congress, Jan. 20, 1955, pp. 174 and 176.

TABLE 6.—*Annual supply of energy from mineral fuels and waterpower in the United States, as related to population* [1]

Year	Annual supply—trillion B. t. u.[2]	B. t. u. per capita—million B. t. u.[3]	Year	Annual supply—trillion B. t. u.[2]	B. t. u. per capita—million B. t. u.[3]	Year	Annual supply—trillion B. t. u.[2]		B. t. u. per capita—million B. t. u.[3]
							PROJECTIONS [5]		
							Low	*High*	
1910	15, 378	166	1948	36, 989	253				
1920	21, 956	206	1949	31, 982	215				
1930	22, 420	182	1950	36, 225	240	1955	41, 300	41, 300	250
1940	25, 216	191	1951	39, 478	257	1960	44, 900	45, 100	255
1945	32, 552	246	1952	38, 780	249	1965	48, 400	49, 400	260
1946	31, 839	227	1953	39, 256	248	1970	51, 900	54, 100	265
1947	35, 674	249	1954 [4]	37, 500	231	1975	55, 900	59, 700	270

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[1] Prepared by: J. J. Gallagher and I. F. Avery, Bureau of Mines.
[2] Data from Bituminous Coal and Lignite in 1953, MMS 2339, p. 141.
[3] Based on population data in table 1.
[4] Preliminary.
[5] Estimated population, including armed forces overseas. Bureau of Census, Ser. P-25, No. 78, Aug. 21, 1953: High—Bureau of Census estimate assumes the 1950-53 fertility rates remain constant through 1975; low—Bureau of Census estimate assumes 1950-53 fertility rates decline linearly from 1953 to the 1959-60 medium series rates by 1975.

trends (fig. 6). Tables similarly numbered present these same data. These are related to the specific commodity trends to note parallelism, rates of deviation, or other significant correlations.

NEW USES

In addition to the established trends, there are numerous instances where the prospect for new uses for a commodity has major significance. The possibility of obtaining atomic energy from thorium is an example of a new use that could entirely reshape the supply-demand balance for monazite. Other commodities have newly established uses that are growing with unusual vigor, such as the application of magnesium to transportation, particularly in the air. The rapid expansion of the chemical industry and the demand for petrochemicals are other noteworthy examples. Outlooks, however, are not always for expansion of demand, as, for example, the declining use of tin in tinplate, with the advent and rapid adoption of electrolytic tinplate displacing that made by the hot-dipped process.

NEW SOURCES

The prospects for new sources or for cheaper sources of some commodities indicate a substantial change in the use pattern. Where a raw material is becoming relatively cheaper owing to discovery or improved methods of extraction, the possibilities for developing new uses or expanding marginal ones may have major significance. The discovery of bastnaesite in commercial quantities in California promises greatly expanded supply and use of the rare-earth metals. Progress continues in commercial development of processes for treating lateritic nickel ores with recovery of byproduct chromium and cobalt. If they are successful, vast resources of these metals would be made available for industrial use.

LOCAL AND REGIONAL DEVELOPMENTS

With the bulkier commodities, the problems of supply and demand sometimes assume a local and regional aspect, such as for sand and gravel. In areas, like the Pacific Northwest, that are moving rapidly toward industrialization, the needs for expanding local supplies, when possible, to meet new local demand is of utmost significance. New sources of power and growing labor forces bring new requirements for industrial raw materials.

FORECASTS

Forecasting is usually confined to the near term but estimates reaching ahead 10 or more years frequently are needed. In some industries, like iron and steel, where the facilities for handling the raw materials represent enormous aggregations of capital, plans must cover relatively long terms to safeguard investments already made or being made and to form a basis for construction of new facilities, which require much time for completion and decades for amortization.

In developing the mineral outlook, with its forecast of trends and prospects, major alternatives sometimes must be considered. For example, for highly strategic minerals an outlook predicated on easing of international tensions may be contrasted with a future that postulates the present defense program to be the prelude to World War III. With respect to fuels, long-range forecasting for any given commodity is a hazard subject to changes in total energy demand, in consumer preferences, and in availability from either domestic or foreign sources.

PROBLEMS

A fourth section on Problems in each commodity program statement sets forth the commodity's present and anticipated problems, with special attention to their magnitude, importance, and urgency. These problems are considered quite apart from the role the Government in general or the Bureau of Mines in particular might play in solving them. In stating problems it frequently is possible to indicate whether it is principally national security or expanding or changing civilian needs that must be met. In most instances, the immediate problems can be differentiated from the long-range ones.

Although there are many exceptions, Government activities usually gravitate toward national security and long-range problems, whereas private research or other private action is the principal reliance in meeting the shorter range nondefense problems. In the larger and more complex problems the value of coordinated attack by several investigators is recognized. It is also appreciated that there is an endless flux in the position of each mineral commodity and with such changes new problems may arise; old problems resolve themselves, and the relative urgency of problems shift.

ALUMINUM AND BAUXITE[1]

By

H. F. Kurtz[2] and D. D. Blue[3]

ALUMINUM ranks first among the primary nonferrous metals produced in the United States. The phenomenal ninefold expansion of primary-aluminum capacity brought about by World War II and the Korean War indicates the great strategic need for this light metal. Its increasing importance in the civilian economy is apparent from the many new nonmilitary goods that are continually being marketed.

Summary

Although aluminum is the most abundant metallic element in the earth's crust, bauxite is the only ore used for large-scale aluminum production. Of the total bauxite consumption in the United States, 85 to 90 percent is required for the production of alumina (aluminum oxide). Over 90 percent of the alumina is eventually reduced to aluminum. Bauxite also has important uses in the production of artificial abrasives, chemicals, refractories, and cements, and in the iron and steel and petroleum industries. Most of the nonmetallurgical uses of bauxite are expanding; however, the scarcity of high-quality domestic ore threatens to retard the future growth of these applications.

In 1953 approximately two-thirds of the world production of virgin aluminum metal was obtained from reduction plants in the United States and Canada. The United States consumed virtually all of the aluminum it produced, plus a significant part of the Canadian production. Imports comprised about three-fourths of the total bauxite supply for the United States in 1953. Canada depended wholly on imported bauxite or alumina as a source material. Most of the bauxite that these countries imported before 1953 came from Surinam and British Guiana; but Jamaica, which has the world's largest aluminum-ore reserve, was expected to become a major supplier of bauxite and alumina in the future. Figure 1 illustrates the structure of the aluminum industry in the United States and the flow of bauxite and alumina to consumers.

The tremendous expansion of the aluminum industry during World War II and again under the Defense Production Act of 1950 reflected the importance of this light metal as a material for modern warfare. The fact that much of the production capacity installed during these emergency expansion programs subsequently has been used in producing civilian goods indicates the growing demand for aluminum for nonmilitary uses.

As a result of continued expansion, maintenance of adequate supplies of raw materials has been a major problem throughout the aluminum industry. Ore, electrical energy, flux, scrap, ingot, and mill shapes, all of which have been among the deficient items in the past, are expected to continue to create supply problems. Problems that will be faced to relieve these shortages and, of course, reduce costs include more efficient use of ore reserves, development of processes for using submarginal reserves, improvement of techniques for using scrap, and further development of the physical metallurgy of aluminum. Accurate statistics on reserves and industrial operations must be available to determine the best approaches to the problems of the aluminum industry and the Government.

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[1] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

[2] Commodity-industry analyst, Bureau of Mines.

[3] Chief, Electrometallurgical Branch, Metallurgical Division, Region II, Boulder City, Nev.

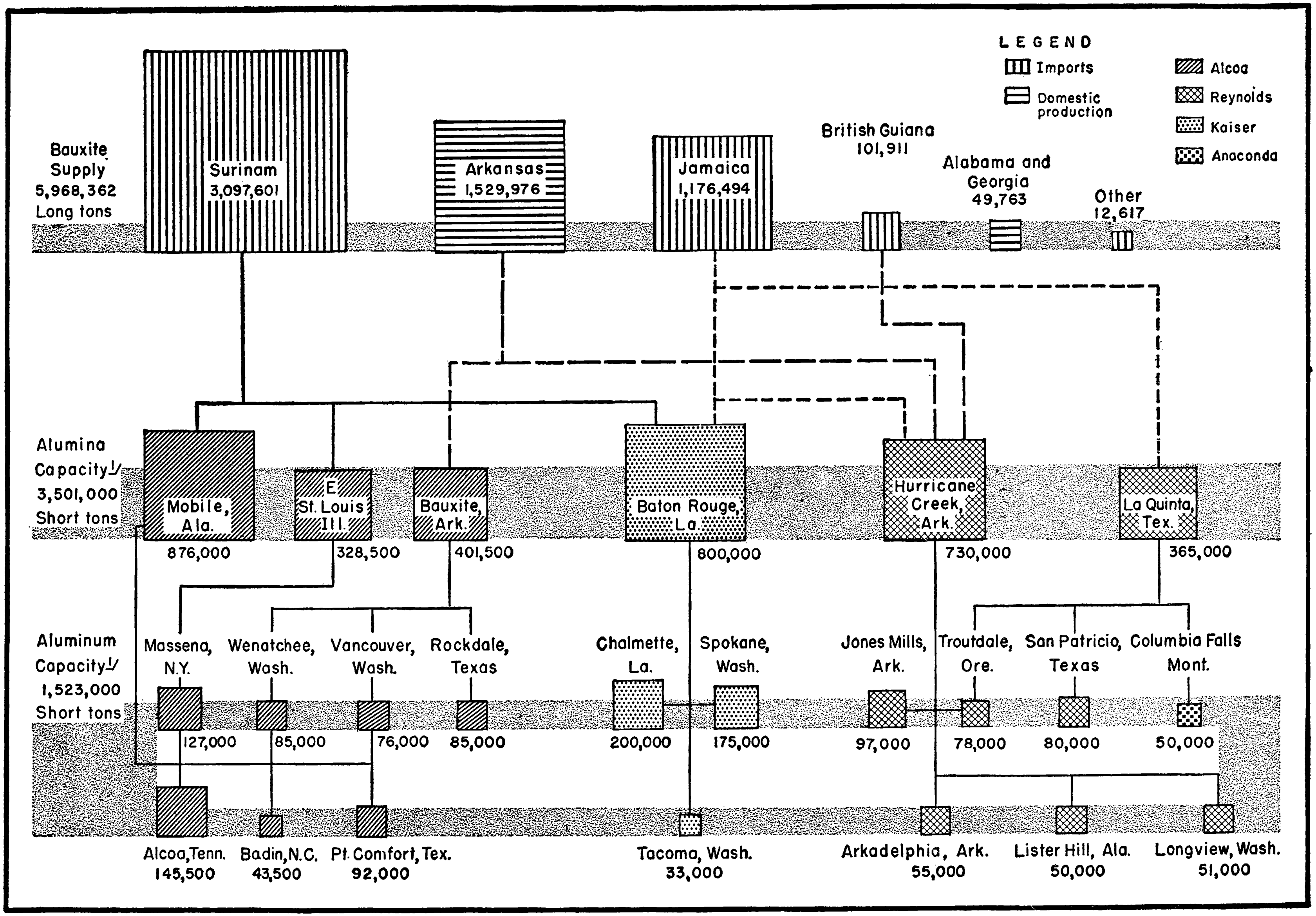

FIGURE 1.—Bauxite Supply and Alumina- and Aluminum-Plant Capacities in the United States in 1953.

[1] Capacities include facilities under construction but not in operation.

BACKGROUND

HISTORICAL SKETCH

The discovery of a continuous electrolytic reduction process for aluminum by both Hall in the United States and Heroult in France in 1886 marked the beginning of the production of aluminum as a commercial metal. Previous to this time the metal had been produced by reducing aluminum compounds by more active metals, but these processes were relatively costly. The electrolytic process reduces relatively pure aluminum oxide (alumina) to metal. In commercial practice most of the alumina has been produced from bauxite. Although aluminum comprises an estimated 8 percent of the earth's crust, it usually occurs in rocks that cannot be utilized economically for producing alumina.

Throughout most of the years of the aluminum industry's phenomenal growth, which has been taking place since 1886, the United States has been the world's major aluminum producer. Germany held the leadership for a few years during its military buildup before World War II, but in 1941 the United States again became the world's largest producer. In 1942 Canada became the second largest producer, and the two North American countries held these relative positions through 1953. Primary aluminum production of the major producing countries and the total world production for 1949 to 1953 are shown in table 1. The principal bauxite producers have been United States, Surinam, British Guiana, France, Hungary, U. S. S. R., Italy, Indonesia, Yugoslavia, and, since 1952, Jamaica. Large undeveloped reserves also exist in French West Africa and Gold Coast. World production of bauxite, by countries, is shown in table 2.

United States production of bauxite began in Georgia in 1889. A decade later the relatively large deposits of central Arkansas were opened up, and these soon became the chief source of domestic ore. Alabama and Georgia have consistently produced small quantities, and for several years bauxite was mined in Tennessee and Virginia. However, over 85 percent of the domestic production has come from Arkansas every year since 1914. Until the early twenties, domestic production far exceeded imports—in fact, in many of those years United States was a net exporter. In 1923 this pattern began to change significantly as imports from the large Surinam and British Guiana deposits began to flow into the United States.

In 1940–41 it became apparent that the aluminum and bauxite industries required a large expansion to meet wartime needs. At that time the Federal Bureau of Mines and the

TABLE 1.—*World production of aluminum by countries, 1949–53*

[Thousand short tons]
(Compiled by Pearl J. Thompson)

Country	1949	1950	1951	1952	1953	Percent of 1953 total
United States	603	719	837	937	1, 252	46
Canada	369	397	447	500	546	20
U. S. S. R.[1]	182	209	220	276	331	12
France	59	67	100	117	125	5
Germany, West	32	31	82	111	118	5
Italy	29	41	55	58	61	2
Norway	39	52	55	56	59	2
Japan	23	27	41	47	50	2
Austria	16	20	29	40	48	2
United Kingdom	34	33	31	31	35	1
Hungary	[1] 15	18	24	26	[1] 33	1
Switzerland	24	21	30	33	31	1
Other	17	16	24	27	28	1
Total	1, 442	1, 651	1, 975	2, 259	2, 717	100

[1] Estimate.

TABLE 2.—*World production of bauxite by countries, 1949–53*

[Thousand long tons]

(Compiled by Pauline Roberts)

Country	1949	1950	1951	1952	1953	Percent of 1953 total
Surinam	2,093	2,048	2,629	3,104	3,221	23
British Guiana	1,758	1,583	2,003	2,388	2,115	15
United States	1,149	1,335	1,849	1,667	1,580	12
Jamaica	--------	--------	--------	376	1,434	11
Hungary	551	563	722	1,153	[1] 1,378	10
France	773	795	1,107	1,097	1,146	8
U. S. S. R.[1]	640	738	837	886	984	7
Other	1,132	979	1,464	1,869	1,965	14
Total	8,096	8,041	10,611	12,540	13,823	100

[1] Estimate.

Federal Geological Survey launched an extensive exploration program that resulted in a substantial increase in the known reserves of ore in Arkansas. In addition, the Government began to stockpile Arkansas bauxite. These activities resulted in expansion of mining operations, and Arkansas production increased to over 6 million tons in 1943.

At the beginning of World War II the Government also embarked on a program that resulted in greatly increased imports of aluminum from Canada and in the construction of 2 commercial and 4 experimental alumina plants and 9 commercial-size aluminum plants. Both large alumina plants and all of the aluminum plants, except one at Tacoma, Wash., operated by Olin Corp., were built and operated by the Aluminum Co. of America. This company was the only domestic aluminum producer until the Reynolds Metals Co. began production in 1941. During the war Reynolds owned and operated two aluminum plants, which were constructed with Government financial aid. The designed capacity of the Government-owned reduction plants at the end of World War II approximately equaled the total capacity of the plants owned by Alcoa and Reynolds. Government-owned alumina and aluminum facilities were disposed of under the Surplus Property Act of 1944. The plants capable of competitive operation were purchased by Reynolds and the Permanente Metals Corp., later renamed the Kaiser Aluminum & Chemical Corp. Over half of the Government-owned capacity, which was constructed for war emergency use only, was considered uneconomical for peacetime operation because of the lack of low-cost electrical power; these plants were dismantled and converted to other uses.

Military requirements for aluminum were reduced toward the end of the war, and primary production dropped from 920,000 tons in 1943 to a low of 410,000 tons in 1946. Between 1946 and 1950 aluminum production and civilian consumption grew steadily. Domestic production of bauxite declined to 1.0 to 1.5 million tons during this period, but imports increased to over 2.5 million tons, or about twice as much as domestic production. The aluminum industry used approximately 85 percent of the total bauxite consumed, while most of the remainder was used in producing artificial abrasives and chemicals. Metal and refractory-grade bauxite was also added to the National Stockpile after World War II.

The outbreak of the Korean War in June 1950 and the decision of the Government to establish a defense economy resulted in passage of the Defense Production Act of 1950. The need for large expansion of the United States aluminum industry and increased supplies of bauxite became evident. An expansion program was planned to create enough new processing capacity to meet essential civilian requirements and military requirements and to leave a surplus for stockpiling. At the start of the program, the capacity of new facilities required to meet these demands was variously estimated at 500,000 to 1,000,000 tons of primary aluminum a year. A number of industrial concerns, consisting principally of established metal fabricators and secondary aluminum producers, investigated the possibilities of producing primary aluminum. Most of these companies did not participate in the program because of the large capital investment required for constructing production facilities with capacities great enough for economical production. The costly integration of producing bauxite, alumina, and aluminum was a practical necessity to achieve a competitive position in the industry. The Harvey Machine Co., Torrance, Calif., a fabricating firm, negotiated with the Government to begin aluminum production but encountered difficulties in obtaining a Government loan. Harvey later joined in a minor capacity with the Anaconda Copper Mining Co. in constructing a reduction plant with 50,000 tons annual capacity at Columbia Falls, Mont. An alumina supply was contracted for with the Reynolds Metals Co. The remainder of the expansion was carried out by the three established producers, Alcoa, Kaiser, and Reynolds.

All new facilities built under the expansion program were financed with private funds. Government aid was furnished in the form of accelerated amortization for tax purposes, guarantees for private loans, market guarantees, and priorities for equipment and construction materials. Unlike some of the facilities constructed during World War II, all new plants were designed and located for continued operation in the postemergency period. The location and estimated annual capacity of United States alumina plants and reduction plants before and after expansion are listed in tables 3 and 4.

As a result of this program, the alumina capacity of the United States was nearly doubled, and United States dependency on foreign bauxite increased. Both Kaiser and Reynolds planned to meet most of their increased bauxite requirements with Jamaican ore. Part of the funds to develop this great reserve of ore was supplied to Reynolds by the Government before enactment of the Defense Production Act. Alcoa planned to increase shipments from Surinam to meet its expanding needs. In addition to bauxite for alumina, requirements for abrasives and other uses have increased. The Aluminum Co. of Canada installed new facilities in British Guiana to provide the United States refractory industry with calcined bauxite to replace the dwindling supply of domestic diaspore. In July 1951 the Government agreed to contribute funds for approved bauxite-exploration projects. Probably as a result of the thorough search for bauxite during World War II, requests for such assistance were few.

TECHNOLOGY

BAUXITE

Although bauxite is occasionally mined by underground methods, in which variations of the room-and-pillar technique are employed, most bauxite is mined in open pits. This may require stripping an overburden several times the thickness of the ore body; but mining recoveries are greater than from underground mines, and many other problems are reduced or eliminated.

Crude, green, wet, or undried bauxite refers to the ore as mined. The only large consumers of crude bauxite in the United States are the alumina plants at Hurricane Creek and Bauxite, Ark. Small quantities of crude ore are sold directly to refractory producers.

Most of the treated bauxite is dried in cylindrical kilns at 200° to 250° F. to drive off free moisture and reduce the shipping weight. The percentage by weight of free moisture in undried bauxite varies from 5 percent to as high as 25 and averages 15 percent for domestic ores. Dried bauxite is produced for the alumina, chemical, cement, and refractory industries. The abrasives industry requires bauxite that

TABLE 3.—*Estimated annual alumina capacity of the United States*

[Short tons]

	End of 1950	Increase, 1951–53	Total
Aluminum Co. of America:			
Mobile, Ala	602, 250	273, 750	876, 000
East St. Louis, Ill	182, 500	146, 000	328, 500
Bauxite, Ark	------------	401, 500	401, 500
Total Alcoa	784, 750	821, 250	1, 606, 000
Reynolds Metals Co.:			
Hurricane Creek, Ark	521, 000	209, 000	730, 000
La Quinta, Tex	------------	365, 000	365, 000
Total Reynolds	521, 000	574, 000	1, 095, 000
Kaiser Aluminum & Chemical Corp.: Baton Rouge, La	440, 000	360, 000	800, 000
Total	1, 745, 750	1, 755, 250	3, 501, 000

TABLE 4.—*Estimated annual primary aluminum capacity of the United States*

[Short tons]

	End of 1949 [1]	End of 1950 [2]	Expansion planned in 1950–51 [3]	Total
Aluminum Co. of America:				
Alcoa, Tenn	145, 300	145, 500		145, 500
Badin, N. C	33, 850	43, 500		43, 500
Massena, N. Y	57, 500	127, 000		127, 000
St. Lawrence, N. Y				
Vancouver, Wash	76, 100	76, 000		76, 000
Point Comfort, Tex	57, 000	57, 000	35, 000	92, 000
Wenatchee, Wash			85, 000	85, 000
Rockdale, Tex			85, 000	85, 000
Total Alcoa	369, 750	449, 000	205, 000	654, 000
Reynolds Metals Co.:				
Jones Mills, Ark	73, 850	[4] 74, 000	23, 000	97, 000
Troutdale, Oreg	72, 000	[4] 76, 000	2, 000	78, 000
Listerhill, Ala	48, 600	50, 000		50, 000
Longview, Wash	30, 500	31, 000	20, 000	51, 000
San Patricio, Tex			80, 000	80, 000
Arkadelphia, Ark			55, 000	55, 000
Total Reynolds	224, 950	231, 000	180, 000	411, 000
Kaiser Aluminum & Chemical Co.:				
Spokane, Wash	108, 125	155, 000	20, 000	175, 000
Tacoma, Wash	24, 250	25, 000	8, 000	33, 000
Chalmette, La			200, 000	200, 000
Total Kaiser	132, 375	180, 000	228, 000	408, 000
Anaconda Aluminum Co.: Columbia Falls, Mont			50, 000	50, 000
Total	727, 075	860, 000	663, 000	1, 523, 000

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[1] Represents economic capacity, having reference to the availability and price of electrical energy (*34*).

[2] Economic capacity plus capacity requiring high cost power.

[3] Data represent capacities as "scheduled" in the expansion program and may differ from "actual" capacities as determined by production. All of this new capacity was installed between 1951 and 1955.

[4] Does not include capacity available from programmed expansion.

has been calcined. In this process, crude ore is heated to temperatures of from 1,700° to 2,400° F. to drive off both the free moisture and the combined water. Natural gas has been used in calcining to control oxidation of the iron for later magnetic separation. The refractory industry uses bauxite calcined in a range of from 2,500° to 2,900° F. A ton of calcined bauxite has the alumina equivalent of 1.5 to 1.6 tons of dried ore. The petroleum industry uses a product called activated bauxite in desulfurizing, deodorizing, and decoloring oils. This material has been partly calcined, crushed, screened, and passed through magnetic separators.

ALUMINA

Although research on the economic manufacture of alumina has continued for many years and is continuing the fact remains that very nearly all alumina commercially produced in 1953 was obtained by a process patented by Karl Bayer in Germany in 1888. A number of modifications and improvements have been made in the process to adapt it to various types of bauxite during the 50-odd years of commercial application; but the basic chemistry—a selective leaching or digestion of the alumina by a causticized soda solution, followed by precipitation of Al_2O_3—has remained the same. This method of production is inexpensive for the preparation of a relatively pure chemical compound. The materials other than bauxite required in the Bayer process are soda ash, lime for causticizing the soda ash, and fuel oil, gas, or coal, all of which are in adequate supply in the United States. The residue after the leaching operation contains inert materials such as Fe_2O_3 and TiO_2, plus a complex $Na_2O \cdot Al_2O_3 \cdot SiO_2$ compound. This compound represents a loss of soda and alumina, and the quantity discarded in the residue depends upon the silica content of the bauxite. Approximately 1.1 units of alumina and 1.2 units of soda are lost for each unit of silica in the ore; thus, bauxites low in silica are required for efficient treatment by the

Bayer process. The upper economic limit of silica for the "straight" Bayer process is about 8 percent.

The reduced supply of imported bauxite during World War II was responsible for commercial application of the "combination process" for recovery of alumina from high-silica bauxites, so named because it combines the Bayer process and an alkaline sinter process. In this process bauxite containing 12 to 13 percent silica is subjected to a Bayer leach. The residue, which contains large percentages of insoluble soda and alumina chemically combined with silica, is not discarded but is treated further by sintering and extraction. This residue from the Bayer plant is superior as a feed material to natural aluminum silicate because it contains two valuable materials, soda and alumina, and the only additional raw materials needed for sintering are limestone and fuel. The additional cost required in capital investment, raw materials, and processing is partly offset by higher overall product yields and soda recovery. A major advantage of the "combination process" is in utilization of bauxite previously regarded unfit for alumina production. The upper limit of silica for use in this process is now about 15 percent, although in practice bauxite containing as high as 25 percent silica is mined and blended with lower silica ore in proportions that will give a feed material of the desired composition.

ALUMINUM

Virtually all virgin aluminum is produced by the Hall process, in which alumina is electrolytically reduced to metal. Most chemical and thermal processes have been unsuccessful because of higher operating costs or inferior products. In the Hall process the alumina is reduced in furnaces known as "pots." A reduction plant or smelter consists of a number of pots electrically connected in series and known as a "potline." The pot assembly is an insulated, carbon-lined, steel box, which, when operating, contains a mixture of molten cryolite and aluminum fluoride in which the alumina is dissolved. Electric current is introduced through carbon electrodes suspended in the molten salts. The current disassociates the alumina into molten metal, which collects on the bottom of the pot, and oxygen, which reacts with the carbon electrodes to form carbon monoxide and carbon dioxide. The process requires a relatively pure alumina because most other metal constituents contained in the feed are simultaneously reduced and appear in the product. The metal is tapped from the pot at intervals and cast into pigs or conveyed to holding furnaces for further treatment. Although the design and size of the pots and electrodes have been modified with usage, the reduction process is basically the same as at the beginning of the industry.

In 1953 an average of 1.94 long tons (dry basis) of bauxite was required to yield 1 short ton of calcined alumina. An average of 1.90 short tons of alumina was required to produce 1 short ton of aluminum; this gave an average of 3.68 long tons of bauxite required to produce 1 short ton of aluminum. The raw materials required in the production of aluminum from alumina are listed in table 5 and are based on 1951 production practices.

In the electrolytic process the aluminum content usually is kept as high as possible; however, alloying materials containing elements such as manganese, silicon, and copper are added to individual pots. After the metal is tapped from the pots it is segregated and treated further according to the chemical analysis. For example, aluminum high in impurities obtained from new cells or relined cells from which impurities are leached is used to make specification alloys, which include the metallic elements found in the cell product. The purest aluminum is used in producing foils, tubes, alclad sheeting, etc., where high malleability is required.

TABLE 5.—*Raw-material requirements*[1] *per ton of virgin aluminum*

	Unit	Quantity
Alumina	Pound	3,825
Power	Kilowatt-hour	17,000
Carbon (total)	Pound	1,385
Green petroleum coke	do	250
Calcined petroleum coke	do	780
Pitch, high softening temperature (105°-115° C.).	do	180
Pitch, low softening temperature (40°-70° C.).	do	125
Coke or calcined anthracite	do	50
Electrolyte, equivalent F_2 weight	do	65
Cryolite	do	47
Aluminum fluoride	do	58
CaF_2 equivalent[2]	do	148
H_2SO_4 equivalent[3]	do	203
Labor	Man-hour	26

[1] Based on industry average in 1951.

[2] Cryolite × 1.23 + AlF_3 × 1.55.

[3] CaF_2 × 1.37.

CONSUMPTION AND USES

BAUXITE

As shown in table 6, most of the bauxite consumed in the United States (86 percent since World War II) has been used in the production of alumina.

The Jacobs process, patented in 1900, marked the beginning of the commercial use of bauxite for abrasives manufacture, the second largest use of bauxite. By this method bauxite is fused in an electric furnace to form a synthetic corundum. The production of this material from bauxite, the only suitable ore, now far exceeds the quantity of natural corundum used.

TABLE 6.—*United States consumption of bauxite by industries, 1949–53*

[Thousand long tons, dry basis]

	1949	1950	1951	1952	1953	Percentage, 1953
Alumina	2, 288	2, 891	3, 365	3, 709	4, 985	89
Abrasive	192	196	304	255	354	6
Chemical	142	163	170	158	169	3
Refractory	23	32	49	53	54	1
Other	33	43	58	53	65	1
Total	2, 678	3, 325	3, 946	4, 228	5, 627	100

Aluminum sulfate is the principal chemical produced directly from bauxite. In 1953, 37 plants produced over 700,000 short tons of commercial aluminum sulfate. Additional quantities were produced at a number of municipal water-treatment plants.

Since 1949 the refractory industry has used increasing quantities of bauxite as a raw material for its high-alumina products as a substitute for Missouri diaspore. Other important uses of bauxite include high-alumina cement, low-density insulating materials, oil purification, and flux for steel and ferroalloy furnaces. The adsorptive qualities of bauxite in various forms have long been utilized for removing impurities in products of the petroleum industry. The calcium aluminate cements made from bauxite have good resistance to attack by chemicals, ability to withstand high temperatures, and quick-hardening features.

ALUMINA

Approximately 90 percent of the alumina consumed since World War II was reduced to aluminum. The remainder of the alumina was used largely in the production of aluminum fluoride, synthetic cryolite, and other aluminum compounds and in the production of white artificial abrasives. Minor quantities of alumina were also employed in the refractory and ceramic industries. As referred to here, the term "alumina" includes hydrates of alumina. Calcined alumina, which is nearly free of water, is required in the production of aluminum and white aluminous abrasives. The trihydrate form is used principally in the chemical industry. Activated alumina, a partly dehydrated alumina trihydrate is utilized as an adsorbent and a catalyst. A tabular alumina is prepared for the ceramic industry.

ALUMINUM

A calculated apparent consumption of primary aluminum (production, producers' stock changes, and net imports of ingot and semimanufactured aluminum) is shown in table 7, with the recovery of aluminum from old and new scrap. This consumption figure does not allow for stock changes at any fabricating or manufacturing plants; however, metal destined for the National Stockpile would be a part of apparent consumption.

The uses of aluminum fall into three broad categories: (1) Wrought products, such as rolled, extruded, and forged shapes, (2) cast products, and (3) dissipative or destructive uses in which the metal loses its identity, such as an alloying element in other metal base

TABLE 7.—*Apparent consumption of primary aluminum and ingot equivalent of secondary aluminum in the United States, 1949–53*

[Thousand short tons]

	1949	1950	1951	1952	1953
Primary: Apparent consumption [1]	634	896	974	1, 072	1, 542
Secondary recovery:					
From old scrap	45	76	77	71	79
From new scrap	136	167	216	233	290
Imports (net)[2]	36	60	17	5	20
Total	851	1, 199	1, 284	1, 381	1, 931

[1] Production and producers' stock changes of crude and net imports of crude and semifabricated products. Net imports may include small quantities of secondary.

[2] Ingot equivalent of net imports (weight×0.90). Imports are largely scrap pig. Some duplication of secondary aluminum occurs because of the small amount of loose scrap imported, which also is included as secondary from old scrap.

TABLE 8.—*Net shipments of aluminum wrought and cast products, 1949–53* [1]

[Thousand short tons]

	1949	1950	1951	1952	1953
Wrought products:					
Plate, sheet and strip	395	582	537	543	684
Rolled structural shapes	102	135	173	222	211
Extruded shapes	75	129	156	174	226
Powder, flake, paste	7	11	12	24	22
Total	579	857	878	963	1, 143
Cast products:					
Sand	61	92	97	97	107
Permanent mold	62	91	80	73	100
Die	49	84	76	85	120
Others	4	5	5	4	2
Total	176	272	258	259	329
Total	755	1, 129	1, 136	1, 222	1, 472

[1] Bureau of the Census, Facts for Industry: Industry Div., Metal and Metals Products Sec., Series M24E and M24-1. Net shipments consist of total shipments to other metal mills for further fabrication.

alloys, as a metallurgical reducing agent, or as a source of aluminum for aluminum chemicals. The data on shipments of cast and wrought products (see table 8) are less than the total apparent consumption figure (table 7), since consumption of aluminum also occurred in other than cast and wrought products.

The expanding number of products in which aluminum is employed results from a unique combination of economic factors and physical properties. Compared with other nonferrous metals, aluminum has been readily available in peacetime and has had a stable, low price. Properties that give the metal competitive advantages include its light weight, electrical and thermal conductivity, heat reflectivity, corrosion resistance, and good strength in alloyed form. In addition, most of the metalworking processes can be applied to aluminum.

As indicated in table 9, military uses, including those of the Atomic Energy Commission, required about one-third of the total aluminum allocations during 1952. As a strategic metal, aluminum's largest application has been in the construction of aircraft. It is also used in many other military items, including airborne equipment, ships, bridges, radar and fire-control towers, rockets, portable shelters, fuel pipes, and miscellaneous equipment used by ground troops.

The major end use of aluminum in civilian applications is in building materials. Hardware, windows, doors, blinds, awnings, roofing, insulation, siding, and wall panels consume appreciable amounts of aluminum. Consumer durable goods, including refrigerators, dehumidifiers, and the like, accounted for 10 percent of the consumption in 1952. Smaller but still significant amounts are consumed by the transportation industry in automobiles, trucks, buses, tractors, trailers, ships, and railroad cars, thereby effecting weight savings and reducing operational expenses. Expanded markets for such items as pistons, bearings, door handles and panels, windows and body trim are developing, and the potential use in automobile radiators, cylinder heads, and engine blocks is even greater. Large quantities of aluminum are also consumed in all types of machinery and agricultural equipment, including such items as oil-field supplies and pipes for irrigation. Aluminum is also used in high-voltage power transmission and electrical equipment. Its use in aluminum cable, steel-reinforced (ACSR), and many other electrical applications have developed rapidly because the price differential in favor of aluminum over copper is even greater when compared on a conductivity-weight basis. Containers, packaging, wrappings, foil, and collapsible tubes all use large amounts of the aluminum consumed.

TABLE 9.—*Shipments of aluminum by consuming industries in 1952* [1]

Industry	Thousand short tons	Percent
Direct military uses	396. 4	29. 8
Building materials	170. 5	12. 8
Consumer durable goods	132. 8	10. 0
Transportation	126. 3	9. 5
(Motor vehicles)	(110. 1)	(8. 3)
Machinery and equipment, excluding electrical	122. 8	9. 2
Construction	122. 5	9. 2
(Electric power construction)	(92. 0)	(6. 9)
Destructive uses	68. 6	5. 2
Electrical and communications equipment	66. 6	5. 0
Containers and packaging	45. 6	3. 4
Exports, chemicals, photography, and others	78. 7	5. 9
Total	1, 330. 8	100. 0

[1] Bureau of the Census and Business and Defense Services Administration, Facts for Industry: Ser. BDSAF-122-03, May 5, 1954.

GEOGRAPHIC STRUCTURE OF INDUSTRY

BAUXITE

Only France among the large bauxite consumers is self-sufficient. The United States, although a large producer of bauxite, imports most of its requirements. World consumption needs are therefore met by a large international flow of ore.

Bauxite production in the United States, as elsewhere, is largely part of a vertically integrated industry; most of the output is mined by companies directly affiliated with consumers. Less than 10 percent of the total 1953 production was by independent companies.

The supply of raw materials has been a major factor in determining plant sites for the various steps required in manufacturing aluminum. If it is necessary to crush, dry, or calcine bauxite before consumption, it is usually processed at plants operated by the mining companies, installed near the mines to save freight.

ALUMINA

Since the major part of United States bauxite supply comes from South America and the West Indies, 3 of the 6 alumina plants in the United States are at ports on the Gulf of Mexico. These plants—at Baton Rouge, La., Mobile, Ala., and LaQuinta (near Corpus Christi), Tex.—account for 58 percent of the total 1954 United States alumina productive capacity. Two alumina plants are adjacent to the Arkansas bauxite deposits. Although these plants occasionally use small quantities of imported ore for "sweetening," production is preponderantly from local ores. For Reynolds, the proximity of reduction plants at Jones Mills and Arkadelphia, Ark., to its Hurricane Creek, Ark., alumina plant offers further transportation savings. The alumina plant at East St. Louis, Ill., uses ore imported from Surinam. Of the six alumina plants, the plant at East St. Louis is the farthest from its bauxite source; however, it is near the fluorspar deposits of southern Illinois and western Kentucky, and part of the alumina produced in this plant is used to produce fluoride chemicals, including AlF_3, synthetic cryolite, and insecticides.

ALUMINUM

Aluminum reduction plants require large quantities of dependable, low-cost power. For this reason, they have usually been situated near hydroelectric developments in areas where there was not a great power demand from industries able to pay premium rates. The aluminum plant at Niagara Falls ceased operations in 1949, largely because of the demand from other consumers. Four plants constructed during World War II were located in metropolitan areas and had to be dismantled after the war because the power situation made them uneconomic to operate. The plants at Massena, N. Y., Alcoa, Tenn., Badin, N. C., and Listerhill, Ala., can be operated at capacity only by using high-cost power.

For the expansion initiated in 1950, the required quantities of hydroelectric power were not available; as a result, the aluminum industry located two-thirds of the new capacity in the South Central States, where thermal power could be generated from low-cost fuels. All of this thermoelectric power comes from natural gas except for one plant at Rockdale, Tex., which uses power generated by burning lignite char.

The widely separated but fixed locations of ore and power sources make transportation costs a major factor in the cost of producing virgin aluminum. Significant transportation costs are also added in distributing the ingot, sheet, extrusions, forgings, etc. To minimize these costs, fabricating facilities are either adjacent to primary plants or in major market areas, such as California and east of the Mississippi. Transportation costs for the secondary aluminum industry are not as large as for the primary industry, since most of the scrap used to make secondary ingot is generated within the smelters' market areas.

SELF-SUFFICIENCY

The United States is not self-sufficient in bauxite. There is a particular shortage in reserves of abrasive-grade and other high-grade bauxites. Table 10 indicates the degree to which the United States has relied on foreign sources of bauxite since 1940. Virtually all bauxite exports during that period were shipped to Canada for the manufacture of crude abrasives. Since most of this material was returned to the United States for final manufacture and consumption, export data have not been included in the self-sufficiency calculations.

Reserve data compared with requirements for the expanding aluminum industry indicate that, if the United States had to be entirely self-sufficient with respect to raw materials for over 5 years, it would be forced to use grades of ore that have never been employed commercially. Low-grade domestic ore can be utilized for alumina by known processes if cost is no consideration. If another wartime emergency should cut off imports from South America, but not Jamaica, the United States could import enough bauxite to maintain its aluminum production without large increases in production cost. In the change from Guiana ore, however, additional equipment would have to be installed

TABLE 10.—*United States self-sufficiency in bauxite for selected years*

[Thousand long tons]

Years	Domestic production (long tons, dry basis)	Imports (crude and dried, long tons as imported)			Percent self-sufficiency $\left(\frac{\text{Domestic production}}{\text{Imports}+\text{domestic production}}\times 100\right)$
		Jamaica	South America	Total	
1940	439	----------	616	630	41
1943	6, 233	----------	1, 548	1, 548	80
1946	1, 104	----------	851	852	56
1949	1, 149	----------	2, 113	2, 688	30
1952	1, 667	265	3, 214	3, 498	32
1953	1, 580	1, 176	3, 202	4, 389	26

at the alumina plants. The ore supply for abrasive and refractory industries would become especially critical with loss of the Guianas.

The National Stockpiles of bauxite and aluminum have been created to provide enough materials to meet anticipated supply deficits during a war emergency period. The accumulation of bauxite represents a measure of security with respect to one raw material, required in the aluminum-manufacturing process. An aluminum stockpile has manifold advantages; it represents a large stockpile of electric power, ocean transportation, domestic transportation, labor, and time and gives assurance to the military services that aluminum will be available for purposes that would best serve the military effort.

As shown in table 11, aluminum imports have historically been smaller than domestic production. Semifabricated aluminum products showed a net import balance for the first time in 1951. Except for 1939 and 1944, crude aluminum has always shown a net import balance, and most of this metal has come from Canada.

The new Canadian smelter at Kitimat, British Columbia, which had not been in production by the end of 1953, was designed for expansion to a capacity of 550,000 tons. It was expected that most of the production from this plant would be exported. Although Canada is strategically well-situated as a supplier to the United States, it depends wholly on imports for its aluminum ore.

TABLE 11.—*United States imports and exports compared to production of primary and secondary aluminum*[1]

[Thousand short tons]

	1940	1943	1946	1949	1952	1953
Production: Primary and secondary	287	1, 234	688	784	1, 242	1, 621
Imports:						
Crude (pig, ingot)	17	136	41	77	128	301
Semifabricated	----------	----------	1	8	16	32
Scrap	1	----------	15	40	7	27
Total	18	136	57	125	151	360
Exports:						
Crude (pig, ingot)	12	57	1	8	1	2
Semifabricated	15	61	15	29	8	8
Scrap	1	----------	1	----------	1	5
Total	28	118	17	37	10	15
Net imports:						
Crude (pig, ingot)	5	79	40	69	127	299
Semifabricated	−15	−61	−14	−21	8	24
Scrap	----------	----------	14	40	6	22
Total	−10	18	40	88	141	345

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[1] Compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

RESERVES, CONSERVATION PRACTICES, AND BYPRODUCTS

BAUXITE

The 1950 estimate of world bauxite reserves shown in table 12 totals 1.6 billion metric tons, about 115 times the 1953 production. A comparison of this reserve figure with 8 previous estimates made at intervals from 1941 to 1953 indicates that new discoveries during that period were made at an average annual rate of about 50 million tons or about 3.6 times the 1953 production. This relationship, when considered with the fact that great areas exist in South America, Africa, and Asia that have never been thoroughly prospected for bauxite, makes the possibility of world bauxite exhaustion very remote.

The only bauxite deposits known on the North American Continent occur in two districts in the United States: (1) Central Arkansas and (2) the southeastern part of the country in two areas: (*a*) The Coastal Plain in Georgia, Alabama, and Mississippi, and (*b*) the Appalachian Valley in Georgia, Alabama, Tennessee, and Virginia. Laterites containing large proportions of iron and/or silica also occur in Oregon and Washington.

The deposits in Arkansas are by far the largest in the United States. They are confined to an area of about 275 square miles in Saline and Pulaski Counties that extends some 25 miles southwestward from Little Rock. Although the reserves of low-silica, low-iron bauxite in the district have been largely depleted, there remains a considerable tonnage of lower grade material suitable for the production of alumina. This district also has large reserves of bauxitic and kaolinitic clays associated with the bauxite.

TABLE 12.—*Estimate of world bauxite reserves as of October 1950*

Country	Million metric tons	Percent of world reserves
British Commonwealth of Nations:		
Jamaica	320.0	19.9
Gold Coast	229.4	14.3
British Guiana	65.0	4.1
India and Pakistan	25.4	1.6
Australia	21.0	1.3
Federation of Malaya	9.6	.6
Subtotal	670.4	41.8
U. S. S. R. and satellites:		
U. S. S. R	30.0	1.9
Hungary	250.2	15.6
Rumania	20.0	1.2
Subtotal	300.2	18.7
Brazil	192.0	12.0
Yugoslavia	105.0	6.6
France	60.0	3.7
Greece	60.0	3.7
Surinam	50.0	3.1
China	50.0	3.1
United States	[1] 40.5	2.5
Indonesia	25.8	1.6
Haiti	23.4	1.5
Other	28.0	1.7
Total—world	1,605.3	100.0

[1] Dry basis, 8 feet or more thick.

All estimates of bauxite reserves are to be taken as approximations, because of the erratic nature of ore occurrence, in pockets and irregular masses. Classification of reserves by grade is difficult because the proportions of minerals often vary widely in a single ore body. Calculations converting wet ore in the ground to mined and dried ore are based on a loss of 15 percent of the ore in open-pit mining and 40 percent in underground mining and on an average of 15 percent free moisture in the mined ore. In Arkansas approximately 85 percent of the ore produced is from open-pit mines and the remainder from underground.

Significant quantities of alumina-bearing minerals are lost in mining and milling bauxite. One of the most obvious examples is the poor recovery in underground mining. In several instances, deposits of metal-grade bauxite have been destroyed as economical sources of ore by selective mining of specialty grades when no market existed for the metal grade.

ALUMINA

The alumina industry, as now operated, produces few byproducts of significant industrial value or application, and the economics of the operations are based only on the production of the primary products. Gallium, which occurs in many bauxite ores, has been recovered from Bayer-process liquors by the Aluminum Co. of America; however, the possible recoveries are less than 1 ounce of gallium per ton of ore. Because of the oversupply of gallium metal and the recovery costs, it is doubtful if this byproduct will contribute much to the reduction of alumina-production costs; however, since gallium is an impurity in the Bayer process and the market price is high, recovery of this byproduct probably will continue.

The largest volume waste product in the manufacture of alumina by the Bayer process is the residue (red mud) remaining after the leaching of the soluble alumina. The "combination Bayer process" plants at Hurricane Creek and Bauxite, Ark., now process this material to recover alumina and soda and in turn produce a residue (brown mud) that is mostly dicalcium silicate.

Alumina plants treating Arkansas ores produce a waste product, designated as "black sands," that is removed from the residue by gravity concentration after the Bayer leaching cycle. These sands contain approximately 4 percent titania and 30 percent iron. Columbium is also concentrated in these sands.

ALUMINUM

In the reduction of alumina to aluminum, carbon monoxide and carbon dioxide gases are discharged from the electrodes. These carry small quantities of fluoride salts and alumina that are discharged with the stack gases, with consequent danger of atmospheric pollution unless gas-cleaning systems are installed. The shortage and increasing prices of cryolite and acid-grade fluorspar, campaigns for reducing pollution in industrial areas, and the indicated harmful effects of the fluorides on animal and vegetable life have demonstrated the necessity for both removing and recovering the fluoride salts and alumina. New plants constructed under the defense expansion program and a number of the older installations have been equipped with gas-cleaning equipment. The collected materials are recharged to the reduction pots with consequent reduction of requirements for fluorides from other sources and recovery of small quantities of alumina.

The increasing shortage of cryolite and fluorides has also prompted another conservation measure for the recovery of cryolite from "spent" pot linings. These linings, a discard item before 1950, are now being processed to recover the cryolite that was absorbed in the lining during its production life.

Aluminum scrap, generated in refining and fabricating operations, is discussed under Secondary Aluminum.

SECONDARY ALUMINUM

Secondary aluminum from marketed sources constitutes approximately one-fifth of the aluminum available for consumption by smelters, foundries, semifabricators, and dissipative users in the United States. The scrap from which secondary aluminum is recovered is designated as "new scrap" and "old scrap." New scrap is (1) waste generated from fabricating operations, such as alloying, stamping, forging, extruding, machining, casting, etc., and (2) rejected semifabricated and manufactured items. New-scrap consumption and metal-recovery data are based on quantities treated outside of the generating plant and do not include "run-around" scrap consumed in the generating plant. Old scrap comes from items that have been used in end products and are collected for metal recovery after being worn out or discarded.

Between 1949 and 1953 secondary recovery from new scrap remained close to 20 percent of apparent primary consumption. This was not true, however, during World War II, when aluminum recovery from new scrap reached a high of 45 percent of primary consumption. The quantity of old scrap available for treatment for metal recovery has shown no consistent trends. In 1947, when a large quantity of aluminum was recovered from obsolete and wrecked aircraft and other ordnance materials, metal recovery from old scrap rose to 29 percent of apparent primary consumption, compared with 5 percent in 1953. Data on recovery of aluminum from scrap are presented in table 7.

Imports of scrap are a third source of secondary aluminum. The major part of the imported scrap is received as "remelt pig" (in ingot form but not of specification grade) to facilitate handling and shipping. Both imports and exports of scrap have usually been small compared with domestic sources of secondary aluminum or with foreign trade in primary metal.

The nonintegrated secondary smelters that produce ingot for resale to the semifabrication industry consume about two-thirds of the scrap. These smelters also use primary aluminum for "sweetening," but their production is based principally on scrap aluminum and controlled blending and refining techniques for production of specification products. The second largest consumers (approximately 28 percent in 1953) are the primary producers who use scrap for blending with primary metal as well as for making secondary ingot. In the first application, metal recoveries are high, and the scrap supplies both aluminum and other elements required for producing specification alloys from "pure" primary metal. Other scrap consumers are foundries, chemical producers, and producers of other metal alloys.

The recovery of secondary aluminum has progressed during the last 30 years from a simple remelting operation producing metal of varying and uncontrolled composition to an operation producing complex alloys of specified composition. The implementation of scrap-segregation processes, the development of sweating, fluxing, and refining techniques for removing objectionable impurities, and the use of the direct-reading spectrograph and other controls on the composition of alloys have resulted in the production of more useful, higher priced products and have strengthened the competitive position of the secondary aluminum industry.

SUBSTITUTES

BAUXITE

There has been much research to develop economical substitutes for bauxite. Of the many proposed substitutes for Bayer-grade bauxite, high-silica bauxite is the only material that has been proved competitive on a commercial scale. Other aluminous materials, such as clay, anorthosite, nephelene and leucite rocks, alunite, and aluminum phosphate, are available in large supply. Although the only plants constructed to use these materials have been of pilot-plant size, the rate of technical progress since 1940 suggests that they will be, at least to some extent, a future source of aluminum.

ALUMINUM

A large part of the increased use of aluminum has been due to its replacement of other metals. It has often replaced iron and steel in the transportation industry; copper, in the electrical industry; tin, in the packaging industry; copper, iron, and enamelware, in the food industry; and a wide variety of products, such as steel, wood, roofing, etc., in the building industry. These replacements have, in most instances, provided less costly, more efficient, and longer lived products. It is thus obvious that other materials could be substituted for aluminum if this becomes necessary. Magnesium has replaced aluminum in some applications, and such replacements will probably increase as new magnesium alloys with higher strengths are developed. The plastics may also be substituted for aluminum in some applications.

COSTS AND PRICES

BAUXITE

Recent data on mine-production costs are not available. Lessors of bauxite properties receive royalties ranging from 40 cents to over $1 per wet long ton. In Arkansas, a State severance tax of 10 cents per short wet ton is also levied. On July 16, 1954, all duties on bauxite were removed for 2 years except the 15-percent ad valorem rate on calcined bauxite imported for use other than for refractories. The tariff on alumina was 0.25 cent per pound.

Since most bauxite is either consumed by the mining companies or their affiliates or purchased under long-term contracts, average quoted prices do not reflect a strongly competitive market but are largely nominal. Historical data on bauxite prices reveal no significant trends. In 1953 the average values of bauxite shipments, f. o. b. domestic mines or producers plants, were as follows: Crude, $7.35 per long ton; dried, $9.82; calcined, $17.57; and activated, $67.34. In comparing the prices of bauxite in the above states of preparation, it is noteworthy that the data are averages of all bauxite, regardless of analysis. Crude ore shipped to the Hurricane Creek and Bauxite, Ark., alumina plants is usually much lower grade than that dried or calcined for the abrasive, chemical, and refractory industries.

ALUMINUM

The major production costs for the three United States primary aluminum producers are shown in table 13. Bauxite and alumina transportation costs, which are included in table 13 as part of the alumina costs, ranged from 0.68 cent per pound of aluminum for Reynolds to 2.00 cents for Kaiser in 1948. Aluminum is sold on a delivered basis, but the cost of shipping the metal was not included. In present practice each mill per kilowatt-hour increase in power costs increases aluminum production costs about 1 cent per pound, which equals about 5 percent of the market price in 1953; however, the reduction process represents a relatively steady power demand, which is desirable from a generating standpoint.

From the time aluminum was first used until 1948, the price of virgin metal declined more or less steadily compared to the Bureau of Labor Statistics Wholesale Price Index and compared to the prices of the major competitive metals—copper, steel, and zinc. Since 1948 the price of aluminum has risen about in proportion to price increases for competitive materials because of increased costs for labor and materials. However, aluminum prices have not fluctuated over short periods, as have those of many other metals.

In contrast to the price stability of primary aluminum, secondary ingot prices have shown large variations. One important factor controlling secondary price is the quantity of primary aluminum available for purchase by ingot consumers. Another is the availability of secondary ingot, which is, in turn, controlled by the availability and price of scrap. At the beginning of 1944, United States primary capacity far exceeded requirements, Canadian metal purchased under a wartime contract was accumulating, and scrap from wrecked and obsolete aircraft was available in large quantities. At this time Secondary Ingot No. 108 and AXS679 averaged 11.0 cents a pound. By late 1948, when stocks of Canadian aluminum held by the

TABLE 13.—*Production costs per pound of aluminum during nine months of 1949 (23)*

[Cents]

Company	Alumina	Carbon electrodes	Power [1]	Net mill costs [2]
Alcoa	3.65	1.31	1.78	10.51
Kaiser	(3)	(3)	1.81	11.63
Reynolds	3.48	1.62	2.32	11.04

[1] Cost after crediting revenue from sales of power to customers and deducting profits from subsidiaries.

[2] Net mill costs represent actual cost after deducting book profits on intracompany transfers, and without allowance for return on investments. Not all subsidiary profits are strictly attributable to aluminum production costs.

[3] Not available.

TABLE 14.—*Average prices of virgin and secondary aluminum ingot and aluminum-base scrap in the United States for selected years*[1]

[Cents per pound]

Year	Primary ingot	Secondary ingot		Scrap	
	(99+ percent Al)	Average No. 108 and AXS679	No. 12	New clippings	Cast
1940	18.7	18.7	14.7	14.5	9.0
1943	15.0	14.2	13.6	7.7	7.0
1946	15.0	14.6	12.2	7.9	5.2
1947	15.0	14.9	14.6	8.6	6.4
1948	15.7	21.4	21.0	11.8	9.4
1949	17.0	18.2	17.4	11.4	7.8
1950	17.7	21.7	21.1	13.2	10.1
1951	19.0	25.8	25.0	15.5	12.1
1952	19.4	20.5	19.4	10.4	7.5
1953	20.9	22.2	21.5	12.6	9.1

[1] Metal Statistics 1954 (American Metal Market).

Government had been completely disposed of, scrap available from military aircraft had been consumed, and total aluminum consumption was increasing, the price of these same grades of secondary ingot soared to 27 cents a pound.

Prices of primary ingot, secondary ingot, and scrap between 1940 and 1953 are given in table 14. In comparing primary and secondary ingot prices it should be noted that the price of primary ingot, 99-plus percent pure, is a base price. At the close of 1953 the prices of many primary aluminum alloys were 1.5 to 3.0 cents per pound above the base price.

In 1953 there were duties of 1½ cents per pound on aluminum pig and ingot and 3 cents per pound on semifabricated shapes. A duty of 1½ cents on imported scrap was suspended between October 1950 and June 1954 by acts of the Congress. Imports of aluminum have been principally in unfabricated form (see table 11), chiefly primary ingot from Canada, which exports over 80 percent of its aluminum production. Available information indicates that Canada and Norway are the only countries that produce aluminum at a lower cost than the United States. The lower production costs are due largely to lower power costs. The Aluminum Co. of Canada has the largest privately owned hydroelectric power system in the world. Additional savings are possible because of the peculiarly ideal geographical concentration of production facilities at tidewater ports.

STATISTICAL INFORMATION

Most of the published statistical information on bauxite and aluminum for the United States is collected and released periodically by the Federal Bureau of Mines and the United States Department of Commerce. Other sources of information include special studies presented in the technical press, usually authored by a staff member of one of the aluminum producers, reports from the Aluminum Association, the Aluminum Smelters Research Institute, or other trade associations, the financial reports of the aluminum producers and investment companies, and reports of projects by other Government agencies such as the Federal Geological Survey and defense agencies.

Statistics for the aluminum industry as a whole are adequate, although some phases should be improved by expanding information or providing more detailed classification. Bauxite-mine-production and primary-aluminum-production statistics, which are among the data most useful for the industry, are also the most accurate because of cooperation among the companies engaged in these operations.

Data on reserves have been collected by the Bureau of Mines and the Geological Survey. Although there are inherent inaccuracies in computing reserves, it is also difficult to keep the data current and catalogued in all the desired forms (by grade, accessibility, etc.).

No statistical data are published regularly on the actual consumption of aluminum in end products; however, statistics on shipments of aluminum mill products, castings, and ingot are published by the United States Department of Commerce and are considered to be a good indication of consumption. Available statistics on alloying materials are inadequate. Data on the consumption of purchased aluminum scrap are published regularly by the Bureau of Mines.

Import-export data are collected monthly by the Bureau of the Census. Statistical information on primary aluminum and bauxite production in foreign countries is published by the Bureau of Mines. The coverage of this survey, although incomplete, is as good as can be expected, considering the international cooperation required.

Data on costs and prices are not generally available other than the statistics on value collected by the Bureau of Mines and the price quotations appearing in trade papers and journals. Employment statistics are collected by the United States Department of Labor, the United States Department of Commerce, and the Health and Safety Division of the Bureau of Mines.

RESEARCH

Research on aluminum since 1939 has been concentrated on utilization of low-grade bauxites and other aluminous ores for the production of alumina, on the development of new alloys with improved properties, on fabrication techniques, and on the development of new end uses. The development of all phases of the bauxite and aluminum industries was accelerated during World War II and again during the expansion program begun in 1950.

During World War II a research program aimed at developing methods for recovering alumina from low-grade ores was undertaken by the Bureau of Mines. Methods for upgrading low-grade bauxites (those with high silica and/or iron content) were investigated. Acid and alkaline processes for extracting alumina were tested, and the most promising methods were investigated on a pilot-plant scale. During this same period other organizations—private, State and Federal—conducted research on similar processes. The United States Government constructed four demonstration plants to test the following acid and alkaline extraction processes:

(1) Laramie, Wyo.—Anorthosite calcined with limestone and soda ash; sodium aluminate leached out of sinter for production of alumina.

(2) Harleyville, S. C.—Clay sintered with limestone; sinter leached with sodium carbonate to form sodium aluminate.

(3) Salt Lake City, Utah—Alunite roasted and leached with sulfuric acid; alumina and potassium sulfate recovered from the alum.

(4) Salem, Oreg.—Clay and ammonium sulfate calcined to form an alum; converted to aluminum hydroxide and calcined to alumina.

When the pressing need for utilizing domestic materials subsided, these plants ceased operation. None had been in operation long enough to test the processes adequately, and 3 of the 4 were converted to other uses. The Bureau of Mines used the remaining plant at Laramie, Wyo., between 1951 and 1954 to investigate the technology, equipment, economics, and product purity of the alkaline sinter process.

Another process for utilizing siliceous aluminum ores produces aluminum-silicon alloys by direct smelting in an electric furnace. This process was tested in a cooperative project by the Apex Smelting Co. of Chicago, Ill., and the Bureau of Mines. As an outgrowth of this research Apex and the American Smelting & Refining Co. formed the National Metallurgical Corp. in 1953 for the construction of a smelter in Lane County, Oreg., to produce "master" aluminum-silicon alloys that can be used to supply both silicon and aluminum in the manufacture of specification ingot. Although consumption of this alloy is limited by the quantity of silicon required for the various silicon-containing aluminum alloys, these noncommercial ores will supply additional aluminum along with silicon, both of which otherwise would have to be produced from high-grade ores by other processes.

There has been much research to develop methods for upgrading low-grade bauxite to make it amenable to established commercial processes and uses. Generally, milling methods for beneficiating bauxitic ores have not as yet proved feasible; however, heavy minerals containing iron and titanium can be concentrated and procedures have been developed for removing iron by magnetic separation preceded by a partial roasting. The development of methods for recovering iron, titanium, and columbium as byproducts of bauxite processing may enhance the economics of using Arkansas ores and at the same time create new sources of titanium and columbium.

A number of processes have been suggested for selectively removing or leaching aluminum from other metallic elements and compounds. The metallic dissolution methods investigated in Germany during World War II can be applied to highly contaminated scrap aluminum and also to crude alloys produced by thermal reduction of aluminous materials. Electrolytic refining of scrap aluminum also shows promise as a means of upgrading contaminated material.

The producers and fabricators of aluminum have given special attention in their research to creating new alloys with improved physical properties, to using techniques of powder metallurgy, to improving fabricating techniques and equipment, and to developing new uses. The fourfold increase in United States apparent consumption of primary aluminum between 1940 and 1950 (reasonably comparable years from the standpoint of economic conditions) indicates the success of this research.

OUTLOOK

Aluminum has very bright prospects for increased consumption and broader applications. World political conditions doubtless will exercise a major influence on the future demand for aluminum. Military and essential civilian demand during a war period would be tremendous; and production would probably be limited by the availability of raw materials, power, transportation, equipment, and manpower. However, consumption predictions based on a healthy, growing, peacetime economy, with a small allowance for defense maintenance, have been made by primary aluminum producers and by Government agencies concerned with aluminum supply. The consensus of their opinion in 1952 was that by 1960 the United States would need over 2 million tons of aluminum annually, excluding National Stockpile requirements. None of the consumption estimates for 1960 has shown a leveling off of the uptrend in use.

Building materials and transportation mediums will remain the chief uses for aluminum, with building materials the likely leader in nonmilitary applications. The large established markets for aluminum construction materials will continue; and the use of many new commodities, such as siding and wall paneling, will develop rapidly. New alloys are expected to broaden the field of structural uses.

The use of aluminum in aircraft reached a high state of development by 1952. Although increased consumption in this field will occur as the use of air transportation increases, it is improbable that aluminum requirements, expressed as percentage weight of aircraft, will increase, and it is even possible that substitution of magnesium and other metals may lower this percentage. The automobile industry is expected to increase its aluminum consumption in a much greater proportion than the aircraft industry. In commercial carriers and shipping containers aluminum will doubtless be preferred, as its use offers substantial economics in lower maintenance and operating costs and the possibility of increasing pay loads.

Many other items of consumption, such as consumer durables, electrical cable and equipment, packaging and containers, agricultural equipment, lightweight irrigation systems, and coatings and finishes are expected to require increased quantities of aluminum. One producer has suggested that in appliances, air-conditioning systems may become the largest use of aluminum.

Improvements in the technology of producing, fabricating, and applying aluminum and its alloys will continue at a rapid pace. Progress in lowering process costs and in developing standby technologies for using ores other than bauxite as source materials will doubtless be made. It is expected that hydroelectrical energy requirements per pound of aluminum produced will be reduced by increasing the efficiency of electrolytic reduction and/or the use of thermally generated power. The greatest technological improvements are anticipated in physical metallurgy. The development of alloys and sintered aluminum powder (SAP) compacts with improved properties and with properties differing greatly from those of known alloys is anticipated. Techniques for fabricating aluminum are expected to improve and be accompanied by lower costs. Solutions of the problems of low-cost joining procedures, soldering, welding or brazing, are being accomplished and should greatly extend the competitive position of aluminum.

The price of aluminum in a peacetime economy will probably increase proportionally less than that of other major metals because of increased production, the availability of high-grade ores in nearby countries, and a fast improving technology. Efforts to shorten the supply lines from mine to alumina plant to reduction plant and even to the ultimate consumer will continue, but the location of low-cost electrical power, transportation costs, the location of existing facilities, and political control of the bauxite reserves are expected to remain as the factors that will control the future geographical distribution of aluminum plants. If foreign ore supplies are cut off and a switch to the use of new raw materials becomes necessary, a large investment in new plant facilities will be required, and production costs and selling prices will increase. Even with this anticipated increase in cost, aluminum probably would maintain or improve its competitive position among the metals.

Although it is undoubtedly true that the United States could be made self-sufficient with regard to the aluminum-ore supply through the use of low-grade materials if cost were not a factor, imports of foreign ores are expected to increase as long as import conditions remain favorable. Changing economic factors, such as production and transportation costs and the recovery of valuable byproducts, could result in an increased use of low-grade domestic ores before the available foreign bauxite is depleted; however, Western Hemisphere deposits are large enough to provide an expanding aluminum industry with high-grade bauxite for many years.

The use of bauxite and alumina in the abrasives, chemical, and refractory industries will continue to be a function of activity in other industries, such as the steel industry and water-purification agencies. Since most phases of the economy are expected to continue to grow, the demand for bauxite and alumina for nonmetal uses will also expand. An increased demand for abrasive and refractory grades of bauxite can be met by shipments from the Guianas. Arkansas, Alabama, and Georgia will also produce low-iron bauxite suitable for manufacturing chemicals and refractories.

PROBLEMS

The long decline in the price of aluminum before World War II and the great increase in bauxite and aluminum production and consumption since then indicate that many major technical problems in these industries have been resolved. However, consumption increases create new problems, particularly with regard to supplies of raw materials. In case of war, many supply problems would become even more acute.

The maintenance of current, accurate, and meaningful data on the economics of the bauxite and aluminum industries is a continuing problem. Since the domestic reserves of high-grade bauxite for alumina, abrasives, chemicals, and refractories are small, maintaining data on reserves is an important aspect of the statistics problem. Data on total bauxite reserves in the United States and nearby foreign countries, classified according to grade and accessibility, must be available for planning research, defense, and industrial operations. Information on the location, quantity, and analyses of possible substitute ores, such as clays, laterite, alunite, nepheline and leucite rocks, anorthosite, and wavellite, is also needed.

Most of the bauxite-mining problems pertain to specific operations and are expeditiously solved by the mining companies. In view of the shortage of high-grade domestic bauxite, one of the basic mining problems is to continue development of deposits of this grade while promoting the conservational objectives of reducing mining losses and preventing the waste of marginal ore and the stripping of ore bodies for high-grade material only.

The problem of increasing the usable reserves of domestic aluminum ores requires development of methods for treating low-grade materials. Improvement of physical or chemical beneficiating processes could extend the ore reserves suitable for abrasive, chemical, and refractory production as well as for alumina. Alumina processes should be frequently reevaluated in the light of new technical information and reagents. The economics of some of these techniques may be improved by using more efficient equipment or recovering byproducts.

Assurance of adequate, steady, low-cost, electrical power has become one of the greatest problems in the reduction phase of the domestic aluminum industry. Power generated from natural gas, coal, and lignite char has been used in place of waterpower, usually at greater cost per kilowatt-hour. Construction of production facilities in foreign countries where hydroelectric potentials are undeveloped may also be considered by United States companies. Procurement of suitable fluxing materials presents another supply problem in producing aluminum.

Most technical problems of the Hall electrolytic process involve product purities and operating efficiencies. The effect of electrolyte impurities, electrolyte composition, alumina particle size, and other factors requires further research. Minute impurities in electrolytes have been shown to have substantial effect on ampere efficiency.

Problems concerning the physical metallurgy of aluminum are numerous. The development of new alloys having greater strength, ductility, corrosion resistance, electrical characteristics, and resistance to high temperatures and the improvement of fabricating and finishing techniques are among the problems demanding major attention.

Government contracts guaranteeing markets for the aluminum produced at several plants constructed during the 1950–54 expansion also provided that specific percentages of production from these plants be made available to fabricators without production facilities for a period of 15 years. This action recognized the independent fabricators' problem of obtaining guaranteed and steady supplies of ingot and crude shapes at competitive prices.

Most technical problems of the secondary aluminum industry concern the improvement of metal recoveries and ingot purities. Companies that use scrap as their chief raw material have serious problems in maintaining competition with other companies because of frequent and wide variations in the price and availability of scrap.

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ANTHRACITE

By

J. W. Buch[1] and J. A. Corgan[2]

PENNSYLVANIA anthracite is one of the purest forms of natural carbon known, containing approximately 86 percent fixed carbon on a dry basis. The larger sizes have a heating value exceeding 13,000 British thermal units per pound on a dry basis. Anthracite does not coke, is low in volatile content, and, under proper firing conditions, burns with a short, blue flame and no smoke. It is not subject to spontaneous combustion and may be stored outdoors for many years without weathering or losing its heating value. These qualities made anthracite for generations the premium space-heating fuel in the New England and Middle Atlantic States areas. Indeed, the early industrial development of the country depended heavily upon anthracite, as it was used extensively in iron and steel metallurgy, for steam generation, and for other industrial purposes.

Summary

The term "anthracite" as used herein refers specifically to the anthracitic or "hard" coal found in the following counties of northeastern Pennsylvania: Carbon, Columbia, Dauphin, Lackawanna, Lebanon, Luzerne, Northumberland, Schuylkill, Susquehanna, and Wayne. These counties contain the greatest reserve of anthracite in the United States, but there are other relatively small deposits in Arkansas, Colorado, New Mexico, Virginia, and Washington. The Pennsylvania anthracite fields, located as they are in a highly industrialized section of the country, not only are one of the Nation's most important natural fuel resources but occupy a prominent position in the total national economy. Throughout the long history of the anthracite industry, over 5 billion net tons has been mined and marketed from this relatively small area in Pennsylvania. However, it is estimated that current reserves can support production at the rate of 30 million tons a year for 300 years.

Peak production was reached in 1917, when 100 million net tons was produced; however, due to inroads made by competitive fuels, particularly fuel oil and natural gas, production declined to an estimated 27 million tons in 1954. This decline in output has created many serious economic and social problems for the industry and the people of the anthracite region. Because of the magnitude and value of the anthracite reserves, it is important to the future economy of the United States and the producing region that some solution be found to these problems. The Bureau of Mines, therefore, in cooperation with the anthracite industry and others, is actively conducting research in the mining, preparation, and utilization of anthracite (aimed at reducing production costs) and in finding new applications for anthracite, particularly in metallurgical and industrial processes. Through such efforts a more diversified demand may be created with less dependence upon the space-heating market, thus stabilizing production and enabling the industry to strengthen its overall economic position. The success of this work would not only improve the economy of the producing region by revitalizing an industry that has been the base of its economy for decades but would enable the industry itself to remain strong enough to expand production rapidly in times of national emergency.

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[1] Chief, Anthracite Research Laboratory, Schuylkill Haven, Pa.

[2] Chief, Branch of Anthracite, Division of Solid Fuels, Bureau of Mines, Washington, D. C.

BACKGROUND

GEOGRAPHICAL DISTRIBUTION OF INDUSTRY

The Pennsylvania anthracite fields underlie a surface area of approximately 484 square miles and are divided by geological conditions into four fields, the Northern, Eastern Middle, Western Middle, and Southern. Mining operations are widely scattered through all four fields. By coal-trade usage, the area is separated into three regions—the Wyoming (which includes the entire Northern field), the Lehigh (which encompasses all of the Eastern Middle field and that portion of the Southern field east

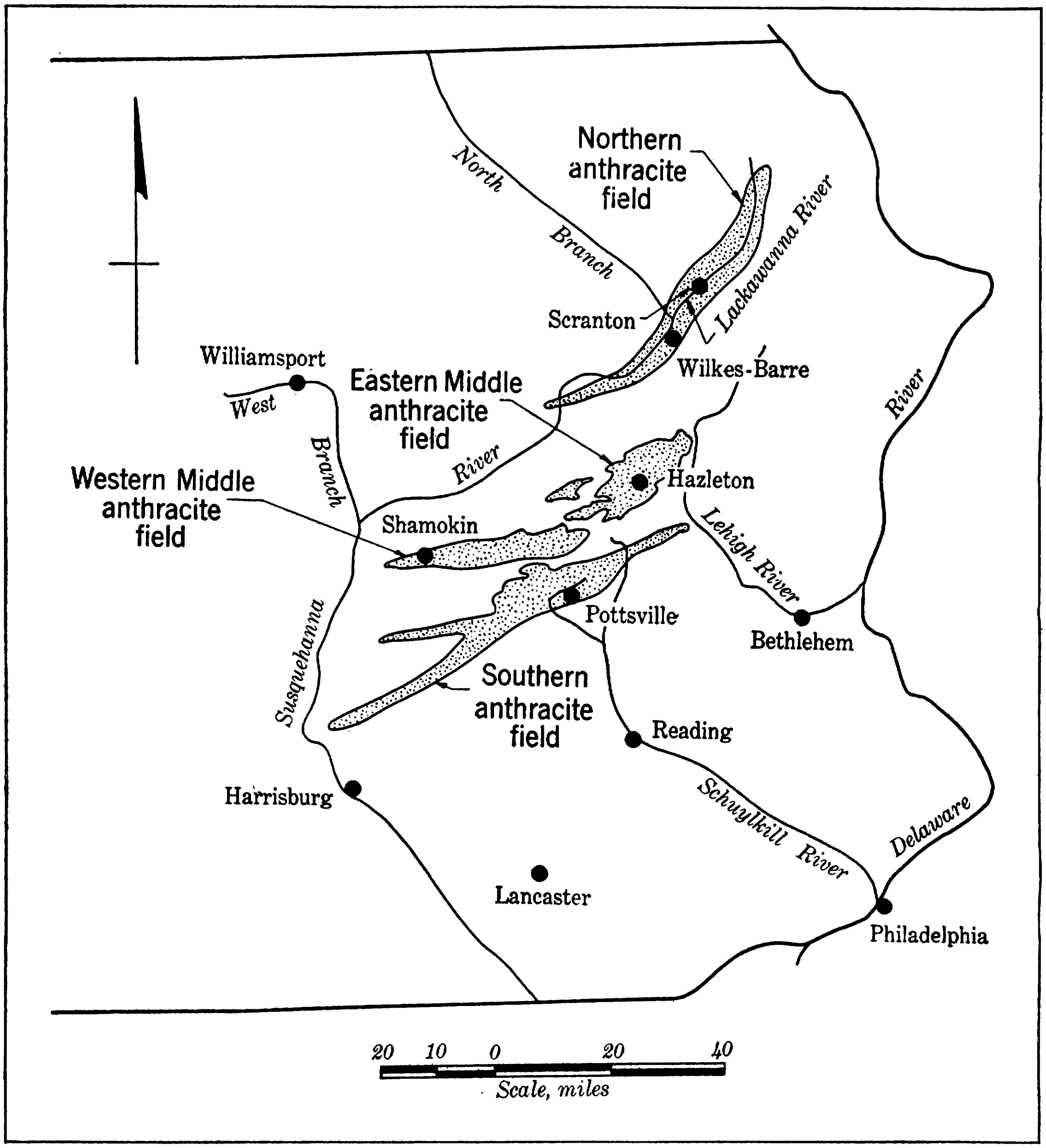

FIGURE 1.—Coal Fields of the Pennsylvania Anthracite Region.

of Tamaqua), and the Schuylkill (which includes the Western Middle field and that part of the Southern field west of Tamaqua). (See fig. 1 for a map of the coal fields.)

The Northern field is about 55 miles long and extends from Forest City in the northeast to Shickshinny in the southwest. About 38 percent of the 1953 total production came from this field. The field is a crescent-shaped basin covering approximately 176 square miles; it contains 14 distinct seams, each of which has been found to be minable at some locality. The seams vary from a minable minimum of 22 inches to nearly 30 feet and are comparatively flat, except near the outcrop, and reach their greatest breadth—about 6 miles—near Wilkes-Barre.

The Eastern Middle field, which centers on Hazleton, produced 8 percent of the 1953 output. It consists of nine distinct canoe-shaped basins extending from the Lehigh River on the east to Catawissa Creek on the west. In some of the basins the coal seams pitch sharply, while in others the beds are relatively flat. This field is the smallest, having a maximum length of 26 miles, a maximum width of 10 miles, and a total area of approximately 33 square miles. Depletion of minable reserves has advanced farther in this field than in any other.

The Western Middle field lies between the Susquehanna River and the headwaters of the Little Schuylkill River, adjoining the Eastern Middle field and extending from it in a general southwesterly direction. The coal seams occur in two principal basins, the Shamokin Basin on the western side and the Mahanoy Basin on the eastern. The field is approximately 36 miles long and 4 to 5 miles wide and covers an area of about 94 square miles. The coal deposits are characterized by thick, steeply pitching beds that flatten at a considerable depth. Of the total 1953 production, 29 percent came from this field.

The Southern field, the largest of the four, extends over approximately 181 square miles of surface area between Mauch Chunk on the Lehigh River to Dauphin on the Susquehanna—a distance of 70 miles. Its greatest width (8 miles) is reached at Pottsville. This field, which contains the largest reserve of anthracite, was subjected at one time to great pressure, with much resultant folding. The deposits are distinguished in this area by steeply pitched coal measures and extremely thick seams. Seam thicknesses of 60 feet have been found in some beds, and in areas of overthrust folding anthracite seams as thick as 100 feet have been found. In 1953, 25 percent of the anthracite output came from this field.

RESERVES

The original reserves of Pennsylvania anthracite totaled about 23 billion net tons, according to a consensus of several authorities. However, by the end of 1953 production and losses had reduced reserves to about 15 billion tons, with the largest amount in the Southern field. The next largest block of reserves is in the Western Middle field, and the Northern field has the third largest reserve tonnage, while the Eastern Middle contains less than 1 percent of the total. On the basis of a 30-million-ton annual production of fresh-mined coal (excluding dredge and culm-bank production) and an overall recovery rate of 60 percent, the estimated reserves will last about 300 years. For details on reserves of Pennsylvania anthracite, by fields, see table 1. For data on reserves of anthracitic coals other than in Pennsylvania, see table 2.

TABLE 1.—*Reserves of Pennsylvania anthracite, by fields*[1]

Field	Reserves	
	Million net tons	Percentage of total
Northern	2, 303	15. 1
Eastern Middle	60	0. 4
Western Middle	3, 442	22. 5
Southern	9, 458	62. 0
Total	15, 263	100. 0

[1] Based upon report of Geo. H. Ashley, State geologist, Commonwealth of Pennsylvania, December 1945: Geol. Survey Circ. 94, 1950; and Bureau of Mines Minerals Yearbook data through 1953.

TABLE 2.—*Reserves of anthracite in the United States other than in Pennsylvania*[1]

[Thousand short tons]

State	Estimated original reserves	Remaining reserves, January 1, 1950	Recoverable reserves, based on 50-percent recovery
Arkansas	230, 000	211, 782	105, 891
Colorado	100, 000	88, 158	44, 079
New Mexico	5, 700	3, 516	1, 758
Virginia	500, 000	480, 816	240, 408
Washington	23, 000	23, 000	11, 500
Total	858, 700	807, 272	403, 636

[1] Averitt, Paul, and Berryhill, L. R., Coal Resources of the United States: Geol. Survey Circ. 94, 1950, 33 pp.

HISTORY

History shows that gunsmiths used coal from the Pennsylvania anthracite region as early as 1755. In 1769 Obadiah Gore, a blacksmith of Wilkes-Barre, Pa., successfully burned anthracite in his forge; following this demonstration, it was used generally by blacksmiths throughout the Wyoming Valley. By 1808 Judge Jesse Fell of Wilkes-Barre, Pa., had succeeded in burning anthracite in an open grate. At that time fuelwood was both abundant and cheap, the smelting of iron ore was still in a primitive stage, and steam engines and railroads lay in the future.

After the turn of the 19th century, fuelwood and charcoal became expensive and scarce in the more heavily populated eastern cities, and the demand for other fuels began to grow. By 1817 a number of small mines had been opened in various areas, but it was not until 1829 that shipments were made from all of the present producing regions. Anthracite was first shipped in "arks," similar in some respects to barges, which were floated down the rivers to markets where the coal was sold and the arks broken up and sold for lumber.

Although "turnpikes" became popular in the first quarter of the 19th century, the cost of transporting freight overland was excessive, and it became apparent that better means of transportation were essential if the anthracite fields were to be developed. It is interesting, then, to observe that the first attempt to establish an inland canal system in the United States had as its primary purpose the marketing of anthracite from the Schuylkill, Lehigh, Lackawanna, and Wyoming Valleys. Several unsuccessful attempts were made to build canals to open up the back country, but it was not until 1825 that the Schuylkill Canal was completed between Pottsville and Philadelphia and fully opened to traffic, although a portion of it was in use in 1818.

The growing demand for anthracite stimulated further canal building. The Lehigh Canal predated the Schuyllkill, as 365 tons of anthracite were shipped over that route to Philadelphia in 1820. These two canals were followed by the Delaware & Hudson, completed in 1828, the Morris Canal in 1831, the Delaware Division Canal in 1832, and the North Branch Canal about 1834. Extensions and connecting links were subsequently added. For half a century or more this canal system had a tremendous influence upon the development of the natural resources of the entire anthracite region, and this in turn caused anthracite to be a most important factor in the early industrial life of the United States. Although the advent of railroads eventually doomed the canals, their remains stand today as monuments to the pioneer spirit and courage that opened up a wilderness in early America.

PRODUCTION

The first statistical record of the production of anthracite is for 1807–20, when 12,000 net tons was produced. Output increased rapidly in the ensuing years until a peak of 100 million net tons was reached in 1917. Inroads made by competitive fuels, especially heating oils, starting in the 1920's, and natural gas, in the 1930's, caused a consistent decline in production, so that by 1938 anthracite output totaled only 46 million tons. Under the impetus of World War II, output naturally increased and in 1944 totaled about 64 million tons. However, after the close of the war and with renewed competition from fuel oil and natural gas, production again declined rapidly to an estimated 27 million tons in 1954—the lowest since 1878. Table 3 gives pertinent statistics on the Pennsylvania anthracite industry for various years in 1890 to 1953.

Anthracite is currently obtained from three principal sources—deep mines, strip pits, and culm banks; and a small amount is recovered by dredges operating in the Lehigh, Schuylkill, and Susquehanna Rivers and their tributaries.

Anthracite as mined consists of run-of-mine material from deep mines and strip pits ranging in size from dust to massive pieces; it contains varying amounts of such impurities as dirt, slate, and rock, all of which must be either removed or reduced to the limits allowed in the specifications for standard anthracite. As anthracite is marketed only as a cleaned and sized product, the modern anthracite-preparation plant is equipped with facilities for crushing or breaking the larger lumps of coal and for separating the broken coal into the sizes required by the market. Impurities are removed in the preparation plant by cleaning devices such as jigs, tables, and dense-medium separators operating on differences in specific gravity between clean coal and refuse. Froth-flotation equipment has been installed at several preparation plants in recent years to clean the fine-mesh particles.

UNDERGROUND COAL

For a number of years underground operations have contributed a decreasing proportion of anthracite, a trend accentuated since the end of World War II—a period of high labor and material costs. As recently as 1947 underground operations yielded 65 percent of the total output; by 1953 the percentage had de-

TABLE 3.—*Statistical trends in the Pennsylvania anthracite industry, 1890, 1900, 1910, 1920, 1930, and 1940–53*

Year	Production (net tons)	Average value per net ton	Exports [1] (net tons)	Imports [1] (net tons)	Apparent consumption [2] (net tons)	Average number of employees	Average number of days worked	Average tons per man per day	Quantity produced by stripping [3] (net tons)	Quantity loaded mechanically underground [4] (net tons)
1890	46, 468, 641	$1. 43	889, 655	16, 962	45, 596, 000	126, 000	200	1. 85	----------	----------
1900	57, 367, 915	1. 49	1, 853, 163	132	55, 515, 000	144, 206	166	2. 40	----------	----------
1910	84, 485, 236	1. 90	3, 384, 222	9, 180	81, 110, 000	169, 497	229	2. 17	----------	----------
1920	89, 598, 249	4. 85	5, 403, 749	31, 748	85, 786, 000	145, 074	271	2. 28	2, 054, 441	----------
1930	69, 384, 837	5. 11	2, 551, 659	674, 812	67, 628, 000	150, 804	208	2. 21	2, 536, 288	4, 467, 750
1940	51, 484, 640	3. 99	2, 667, 632	135, 436	49, 000, 000	91, 313	186	3. 02	6, 352, 700	12, 326, 000
1941	[5] 56, 368, 267	4. 26	3, 380, 189	74, 669	52, 700, 000	88, 054	203	[6] 3. 04	7, 316, 574	13, 441, 987
1942	[5] 60, 327, 729	4. 50	4, 438, 588	140, 115	56, 500, 000	82, 121	239	[6] 2. 95	9, 070, 933	14, 741, 459
1943	[5] 60, 643, 620	5. 06	4, 138, 680	166, 020	57, 100, 000	79, 153	270	[6] 2. 78	8, 989, 387	14, 745, 793
1944	[5] 63, 701, 363	5. 57	4, 185, 933	11, 847	59, 400, 000	77, 591	292	[6] 2. 79	10, 953, 030	14, 975, 146
1945	[5] 54, 933, 909	5. 90	3, 691, 247	149	51, 600, 000	72, 842	269	[6] 2. 79	10, 056, 325	13, 927, 955
1946	[5] 60, 506, 873	6. 83	6, 497, 245	9, 556	53, 900, 000	78, 145	271	[6] 2. 84	12, 858, 930	15, 619, 162
1947	[5] 57, 190, 009	7. 22	8, 509, 995	10, 350	48, 200, 000	78, 600	259	[6] 2. 78	12, 603, 545	16, 054, 011
1948	[5] 57, 139, 948	8. 17	6, 675, 914	945	50, 200, 000	76, 215	265	[6] 2. 81	13, 352, 874	15, 742, 368
1949	[5] 42, 701, 724	8. 38	4, 942, 670	-------	37, 700, 000	75, 377	195	[6] 2. 87	10, 376, 808	11, 858, 088
1950	[5] 44, 076, 703	8. 90	3, 891, 569	18, 289	39, 900, 000	72, 624	211	[6] 2. 83	11, 833, 934	12, 335, 650
1951 [7]	42, 669, 997	9. 51	5, 955, 535	26, 812	37, 000, 000	68, 995	208	2. 97	11, 135, 990	10, 847, 787
1952 [7]	40, 582, 558	9. 36	4, 592, 060	29, 370	35, 300, 000	65, 923	201	3. 06	10, 696, 705	10, 034, 464
1953 [7]	30, 949, 152	9. 67	2, 724, 270	31, 443	28, 000, 000	57, 862	163	3. 28	8, 606, 482	6, 838, 769

[1] U. S. Department of Commerce.
[2] Before 1913 figures on consumption do not include producers' stocks.
[3] Data first collected in 1915.
[4] Data first collected in 1929.
[5] Includes some "bootleg" coal purchased by and prepared at the breakers of authorized operators.
[6] Output per man calculated on authorized tonnages only; "bootleg" purchases excluded.
[7] Data for 1951–53 are not strictly comparable with those for previous years. See Minerals Yearbook 1951.

clined to 58 percent. Increased output from strip pits and culm banks, where output per man per day is generally higher than in deep mines, resulted in increased productivity—from 2.97 tons per man-day in 1951 to 3.28 tons in 1953, the highest rate achieved in the history of the industry.

STRIP-PIT COAL

When strip mining in the anthracite region was begun on a small scale in the early 1800's, recovery by surface mining was extremely small, as the overburden and coal were removed by manual labor or horse-drawn scrapers. In the absence of mechanical equipment, the coal was mined from the outcrops most readily accessible or from seams lying close to the surface. However, with development of internal-combustion engines, strip mining increased, and its relatively rapid rise in the Pennsylvania anthracite region, although impelled primarily by the necessity of reducing overall production costs, was made possible by technological advances in metallurgy, machinery design, and the capacity of earth-moving equipment. In 1953 strip-pit production comprised 28 percent of the total output. Aside from geologic and engineering problems, the primary consideration in strip mining is the ratio between overburden and recoverable coal or the number of feet of earth, slate, or rock to be removed to recover a certain tonnage of coal. Advances in enlarging the capacity of shovels and draglines and in applying, successively, gasoline, electric, and diesel power to mining operations, plus improvement in haulage methods and equipment, have enabled operators to mine economically at ever-increasing depths.

The thick, pitching beds in the Lehigh and Schuylkill regions are more adaptable to stripping than the comparatively level but deeper lying coal seams of the Wyoming region. Therefore, although the Wyoming region led in the total quantity produced until 1951, the Lehigh and Schuylkill regions have surpassed it for many years in percentage of coal obtained from strip pits.

CULM-BANK COAL

No appreciable market existed for the smaller sizes of anthracite, especially Pea and smaller, during the early years of anthracite mining, as the anthracite-burning equipment in use was designed to burn only the larger sizes. Consequently, tremendous quantities of the small sizes and some large masses of refuse containing good coal were piled in large spoil, or culm banks in the anthracite region; however, introduction of automatic stokers and improvements in grates, boilers, and firing techniques at industrial and commercial installations created a growing demand for the smaller sizes. With the increasing demand for the small-size coal, culm banks became more important, since the cost of production from this source was considerably lower than from deep or strip mining. Hence, over the past few decades the culm banks have contributed a substantial output, with a record of 9.6 million tons set in 1944. In 1953 culm banks accounted for 13 percent of the total output. The amount of anthracite remaining in the banks has decreased greatly and this will not be an important source in the future.

RIVER COAL

The rivers and creeks that traverse the anthracite fields of Pennsylvania have been dredged for recovery of anthracite since the late 1800's. In the early history of the anthracite industry, Grate, Egg, Stove, and Chestnut were the principal sizes prepared. Coal smaller than Chestnut was usually discarded as useless and was either dumped with the breaker wastes into surface streams or piled into large culm (spoil) banks. Part of the culm-bank material was subsequently washed into streams; and eventually the beds of the Susquehanna, Lehigh, and Schuylkill Rivers and such creeks as the Shamokin, Mahanoy, Wiconisco, and Swatara were covered with considerable quantities of anthracite.

With the advent of improved grates and boilers, heating and power plants were equipped to burn the fine coal taken from the streams. Around 1890 river coal began to assume importance as an industrial fuel, especially in electric-power generation. From 1909, when production statistics were first collected, through 1940 dredge operators reported the recovery of almost 18 million tons and from 1941 through 1953 an additional 13 million. In 1953 river-coal output was 1 percent of the total anthracite production. Although these figures are but a small fraction of the total output, they are important from the standpoint of conservation of natural resources. With the increased demand for small anthracite and the introduction of State stream-pollution laws, less coal reaches the streams, and therefore the quantity of anthracite available from this source is decreasing rapidly.

DISTRIBUTION AND CONSUMPTION

Because of the location of the region, anthracite has been distributed most widely in the area north of the Potomac and Ohio Rivers and east of the Mississippi. Its primary market area, which normally receives about 85 percent of annual shipments, consists of the New England States, New York, New Jersey, Pennsylvania, Delaware, Maryland, and the District of Columbia. It is in this area and in the Great Lakes States that anthracite has suffered severe tonnage losses to competitive fuels.

The principal reason for the decline in consumption of anthracite has been substitution of competitive fuels, particularly fuel oil and natural gas, for anthracite in its principal markets. The severity of the decline in anthracite consumption in the primary market as a result of competition can best be realized by pointing out that anthracite, which in 1929 comprised almost 83 percent of the total equivalent consumption of anthracite, coke, heating oils, and natural gas for space-heating purposes, decreased to 21 percent of the total in 1952, whereas the percentage for fuel oil, expressed in terms of coal equivalent, increased from 8 percent in 1929 to about 62 in 1952 and natural gas from 5 percent to 16; domestic coke declined from 4 percent to 1. In the 1920's large quantities of anthracite were shipped to the North Central and Lake States. Here again, competition cut anthracite shipments from 10,000,000 tons

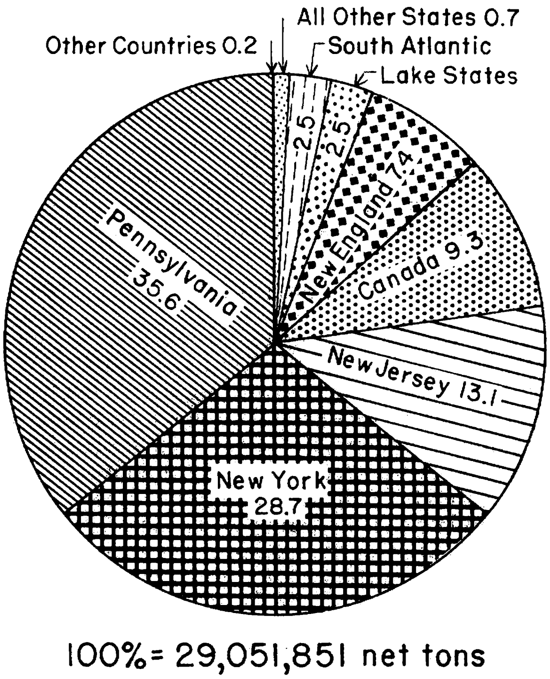

FIGURE 2.—Distribution of Pennsylvania Anthracite, by Destinations, in Percentage of Total, 1953–54.

("Lake States" includes Illinois, Michigan, Minnesota, Ohio, and Wisconsin. "South Atlantic" includes Delaware, District of Columbia, Maryland, and Virginia only.)

TABLE 4.—*Distribution of Pennsylvania anthracite, April 1, 1953, to March 31, 1954, by States, Provinces, and other countries of destination*

[Net tons]

Destinations	Domestic sizes	Steam sizes	Total, all sizes	Percentage of total
United States:				
New England States:				
Connecticut	337, 965	77, 520	415, 485	1. 43
Maine	133, 499	21, 388	154, 887	. 53
Massachusetts	909, 490	191, 864	1, 101, 354	3. 79
New Hampshire	91, 822	81, 793	173, 615	. 60
Rhode Island	115, 329	25, 110	140, 439	. 48
Vermont	108, 819	55, 555	164, 374	. 57
Total	1, 696, 924	453, 230	2, 150, 154	7. 40
Middle Atlantic States:				
New Jersey	1, 670, 570	2, 140, 050	3, 810, 620	13. 11
New York	4, 289, 940	4, 038, 740	8, 328, 680	28. 67
Pennsylvania [1]	3, 802, 940	6, 539, 170	10, 342, 110	35. 60
Total	9, 763, 450	12, 717, 960	22, 481, 410	77. 38
South Atlantic States: [2]				
Delaware	161, 729	43, 052	204, 781	. 71
District of Columbia	82, 655	17, 095	99, 750	. 34
Maryland	248, 539	109, 297	357, 836	1. 23
Virginia	44, 904	23, 428	68, 332	. 24
Total	537, 827	192, 872	730, 699	2. 52
Lake States: [3]				
Illinois	48, 412	73, 531	121, 943	. 42
Michigan	92, 292	51, 405	143, 697	. 49
Minnesota	9, 264	70, 324	79, 588	. 27
Ohio	22, 537	57, 583	80, 120	. 28
Wisconsin	194, 773	101, 793	296, 566	1. 02
Total	367, 278	354, 636	721, 914	2. 48
All other States	50, 050	137, 595	187, 645	. 65
Total, United States	12, 415, 529	13, 856, 293	26, 271, 822	90. 43
Canada:				
Ontario	1, 870, 340	148, 218	2, 018, 558	6. 95
Quebec	321, 351	341, 425	662, 776	2. 28
Other Provinces	25, 698	4, 944	30, 642	. 11
Total, Canada	2, 217, 389	494, 587	2, 711, 976	9. 34
Other countries	37, 599	30, 454	68, 053	. 23
Grand total	14, 670, 517	14, 381, 334	29, 051, 851	100. 00

[1] Includes "local sales."

[2] Shipments to other States generally referred to as being in the South Atlantic area are included in "all other States."

[3] Shipments to Indiana are included in "All other States."

a year before 1925 to less than 1 million tons in the 1953–54 coal year. Table 4 shows distribution of anthracite during the coal year 1953–54, by States, Provinces, countries of destination, and size groups. Distribution by destinations, in percentage of total, is illustrated graphically in figure 2.

Although anthracite is used principally as a space- or comfort-heating fuel, large quantities of the small sizes are used for industrial purposes, especially in the production of electric energy. In 1953 consumption by electric utilities totaled 3.6 million net tons or the equivalent of 12 percent of that year's production. Consumption by railroads for power and other purposes totaled 532,000 tons; 305,000 tons was used in manufacturing fuel briquets and 132,000 tons in the manufacture of gas. Taking into consideration exports and changes in producers' stocks, the apparent consumption of Pennsylvania anthracite in the United States in 1953 amounted to 28 million net tons.

TABLE 5.—*Pennsylvania anthracite exported from the United States, 1945–53*

[Thousand net tons]

(U. S. Department of Commerce)

Country of destination	1945	1946	1947	1948	1949	1950	1951	1952	1953
Canada	3,393	4,514	4,470	4,932	3,581	3,798	3,485	3,607	2,602
Belgium and Luxembourg	135	777	1,368	210	-------	21	227	10	------
France	34	758	1,405	1,425	1,051	51	1,334	486	------
Italy	-------	241	158	53	30	-------	45	11	------
Netherlands	-------	88	42	5	155	-------	491	340	------
Norway	-------	9	201	-------	(1)	-------	190	-------	(1)
Sweden	12	-------	578	-------	-------	-------	-------	-------	------
Switzerland	26	25	115	-------	-------	-------	85	1	------
All other countries	91	85	173	51	126	22	99	137	122
Total	3,691	6,497	8,510	6,676	4,943	3,892	5,956	4,592	2,724

[1] Less than 500 tons.

FOREIGN TRADE[3]

Canada is the anthracite industry's secondary market, receiving over 9 percent of the total shipments in the 1953–54 coal year. It is anticipated that the Canadian market will remain relatively stable for several years; however, should the projected introduction of natural gas into eastern Canadian markets materialize, this important market may suffer also from competition, as has the anthracite primary market area. Although shipments to Europe have been substantial in some of the years since the close of World War II, the overseas demand has fluctuated widely with the availability of European coal. Table 5 shows exports of anthracite to Canada and other foreign countries for calendar years in 1945–53.

PRICES

As in the case of many other commodities, prices of anthracite have advanced greatly since 1940. Material, transportation, and labor costs have increased substantially during this period and have necessarily been largely responsible for the overall gain in anthracite prices. In 1940, for example, the average value per net ton of anthracite at the mines was $3.99 compared with $9.67 in 1953. This is an average value; the smaller sizes have a lower value and the larger space-heating sizes a greater value. Anthracite, of course, costs much more at points distant from the mines because freight rates and handling charges of coal merchants must be added to the f. o. b. mine prices. For example, in Boston, Mass., in November 1954, the prices of Buckwheat No. 1 and Stove sizes of anthracite were $21.30 and $27.95 per net ton, respectively. The Bureau of Mines, in cooperation with the anthracite industry, is conducting research aimed at increasing the output per man per day. If this objective materializes, it is possible that anthracite prices might be decreased.

TECHNOLOGY

MINING METHODS

The recognized conventional systems of mining in the anthracite region are chamber-and-pillar, longwall, breast-and-pillar, and slant chute. The chamber-and-pillar method is used in flat areas in all fields, regardless of coal-bed thickness. With this method, chambers are driven 24 to 30 feet wide and about 300 feet long, leaving pillars 26 to 36 feet wide between the chambers. In most instances, these pillars are mined later.

The original longwall methods were virtually slabbing systems—slabbing of pillars approximately 300 feet wide and 500 to 1,000 feet long. Mining was done on both retreat and advance systems. Since its introduction in 1915, use of this method has been confined principally to thin coal beds—those 4 feet thick or less.

The breast-and-pillar system is a method used in both thin and thick coal beds in steeply pitching areas. With this method chambers or "shrinkage stopes," to define the system more exactly, are driven 30 feet wide straight up the pitch. These breasts are so centered that a supporting pillar is left between them. An attempt is made to mine the supporting pillar later by developing pillar roads or chutes within the pillars and then driving laterally to make contact with the mined-out breasts on either side. The percentage of the deposit recovered decreases with increase in bed thickness and varies from 70 to 40 percent.

The slant-chute method is similar in principle to the breast-and-pillar method and is used in steeply pitching beds where the coal broken by this method will run through chutes into

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[3] Figures on exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

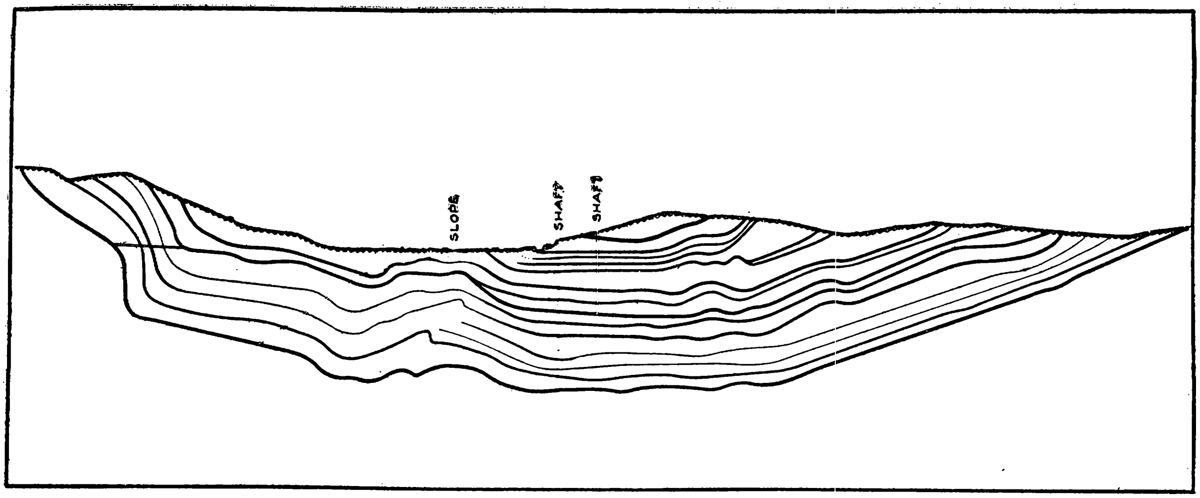

FIGURE 3.—Cross Section of Coal Seams in the Northern Anthracite Field, Pennsylvania.

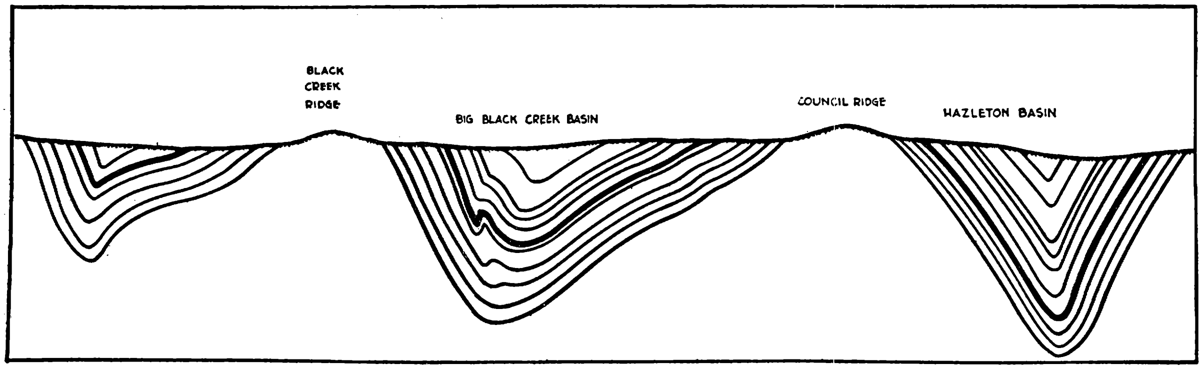

FIGURE 4.—Cross Section of Coal Seams in the Eastern Middle Anthracite Field, Pennsylvania.

mine cars. The essential difference between the slant-chute and the breast-and-pillar methods is that slant chutes are driven across the pitch at an angle of about 35°, whereas breasts are driven at right angles to the strike of the coal bed. With the slant-chute method, pillars are withdrawn progressively downward by the use of long blastholes. The percentage of recovery varies but little with various bed thicknesses and averages 70 percent or more.

Stripping methods have been used for many years in mining outcrops of coal beds. In more recent years there has been a trend toward the mining of deep-seated deposits. The development of modern stripping equipment of large capacity has made this phase of anthracite stripping possible. At present, anthracite strip pits have reached the depth of 400 feet. Cross sections of the anthracite fields are shown in figures 3 to 6.

UNDERGROUND MINE MECHANIZATION

The production record for anthracite and bituminous-coal mechanical loading in underground mines for 1930–53 is shown in table 6. It is interesting to note that in 1932 about the same proportion of anthracite and bituminous-coal underground production was loaded mechanically. If the Wyoming region is considered seperately because its flat-bed area is more comparable with the geological conditions of bituminous-coal deposits, it is to be noted that by 1933 the proportion of underground production of anthracite mechanically loaded was nearly twice that of bituminous coal so mined. The difference in favor of anthracite existed until 1942, when many thin beds for which mechanical loading had been developed were depleted. In the meantime, many thick-bed areas were mechanized. It is also to be noted that much of the so-called hand-loaded anthracite is from pitching seams and is loaded by gravity.

MINE SAFETY

Coal mining has always been considered a hazardous occupation. The trend in the fatality rate for anthracite mining, although some-

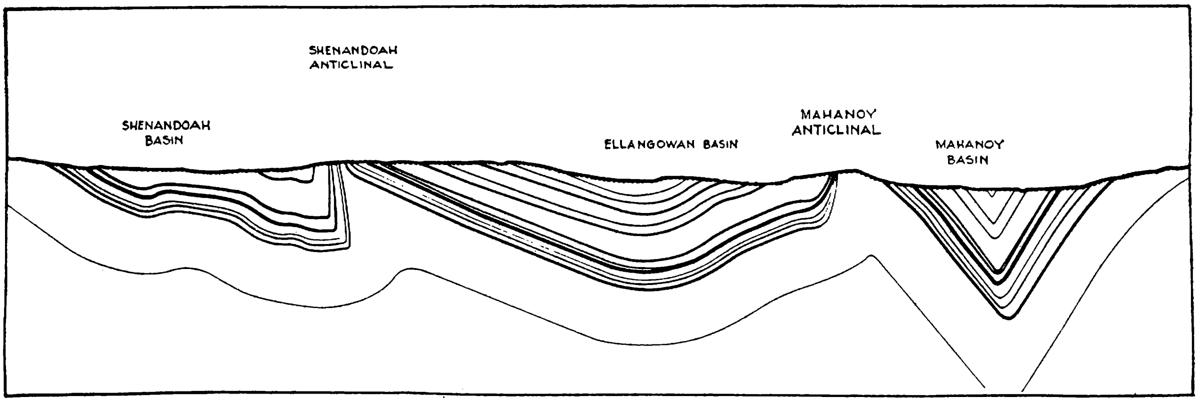

FIGURE 5.—Cross Section of Coal Seams in the Western Middle Anthracite Field, Pennsylvania.

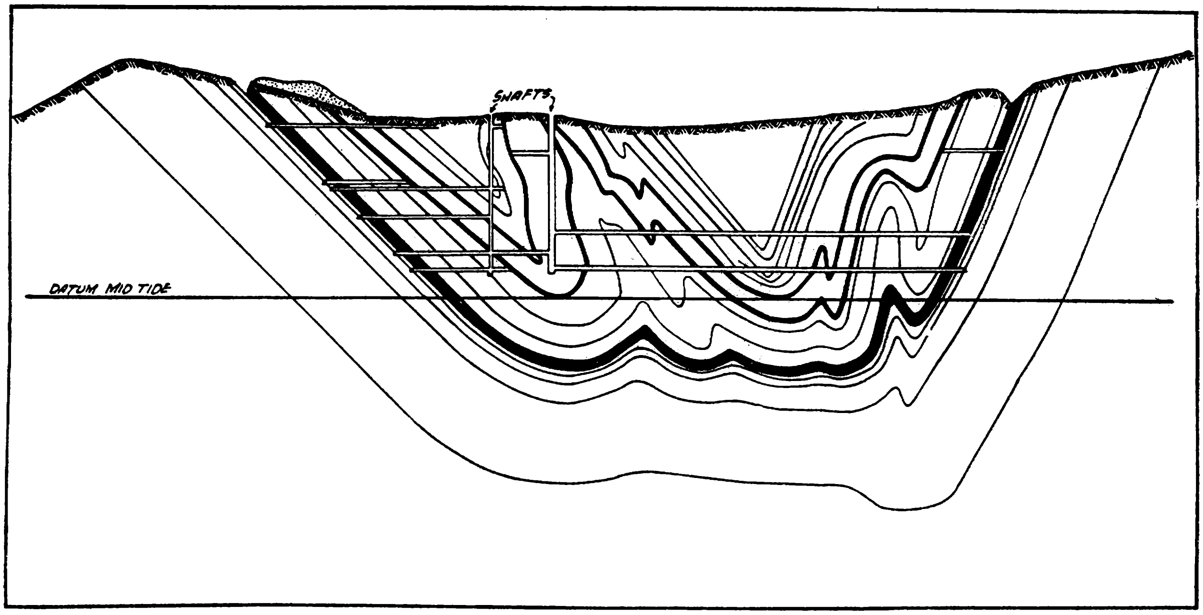

FIGURE 6.—Cross Section of Coal Seams in the Southern Anthracite Field, Pennsylvania.

what erratic, showed marked improvement during the past 22 years, having dropped from 1.76 per million man-hours in 1930 to 1.03 in 1952. The rate of fatalities per million tons mined has decreased from 5.6 in 1924 to 2.1 in 1950 and in 1953. In spite of these decreases, the record is not as good as desirable and calls for further improvement.

Legal protection is provided for the safety of mineworkers by both State and Federal laws. Efforts are made by the producing companies and State and Federal inspectors to see to it that these laws are obeyed. In addition to the inspectors provided by Government agencies, most producing companies have their own safety engineers and safety inspectors. Many companies have gone beyond the safety measures provided by law and have established additional rules for their particular companies.

Improvement in safety is a never-ending problem. In 1949, District No. 1, United Mine Workers of America, established a safety organization directed by one of its officers. This organization contributed to and gained the cooperation of the operating companies in setting up a safety-training program for the rank and file, as provided by instructors from the Federal Bureau of Mines. The safety engineers of the anthracite region are banded in the Anthracite Safety Engineers Association for the purpose of discussing problems in safety and accident-prevention work. The association meets at least once each month to discuss accidents, progress, and planning, and at various times the Federal Bureau of Mines has been invited to provide speakers on subjects of mu-

TABLE 6.—*Percentage of underground production loaded mechanically; anthracite and bituminous coal and lignite, 1930–53* [1]

Year	Anthracite		Bituminous coal and lignite
	All regions	Wyoming region	
1930	6.9	(2)	10.5
1931	8.2	(2)	13.1
1932	12.4	(2)	12.3
1933	16.0	22.9	12.0
1934	19.1	27.2	12.2
1935	21.2	27.6	13.5
1936	24.2	31.6	16.3
1937	25.1	32.7	20.2
1938	26.6	35.9	26.7
1939	27.7	35.7	31.0
1940	29.7	38.3	35.4
1941	30.6	40.9	40.7
1942	32.6	44.5	45.2
1943	34.5	46.6	48.9
1944	35.8	49.5	52.9
1945	39.9	52.2	56.1
1946	41.0	57.1	58.3
1947	43.4	58.9	60.7
1948	42.3	59.4	64.3
1949	43.9	63.0	67.0
1950	43.8	63.5	69.4
1951	41.2	64.9	73.1
1952	40.5	64.4	75.5
1953	38.2	62.4	79.6

[1] Mineral Resources of the United States, pt. 2, 1930–31; Bureau of Mines Mineral Yearbook, 1932–52; and Bureau of Mines Mineral Market Repts. MMS 2323 and 2339

[2] Data not available.

tual interest. Through the cooperation of such organizations with the States and the Federal Bureau of Mines, the mechanism is established for increased safety in mining. Additional coordination, direction, and education are needed. If, through training, a man can be made more safety conscious, he will make mining work safer and, therefore, more attractive for his neighbor as well as for himself. One foremost consideration in developing new mining methods must be whether or not a new method is safer than the one it is intended to displace; if it is not, it should be discarded until a safer method has been devised.

PREPARATION

As anthracite emerges from underground mines it contains impurities and ranges in size from fines to very large lumps. It is the function of the preparation plant to break the large pieces, remove the impurities, and classify the coal into uniform sizes. Table 7 shows standard size specifications as approved and adopted by the Anthracite Committee of the Commonwealth of Pennsylvania.

The first anthracite breaker, capable of preparing 200 tons of coal a day, was erected in the Southern field in 1844. Modern breakers are far removed from this original plant; one that was built not too many years ago is reported to have cost about $5 million and has a production capacity of 15,000 tons of coal per 16-hour day. Wet washing of anthracite has been practiced for many years, and jigs were a favorite cleaning device. This was followed by the introduction of the Rheolaveur process and Chance Cone cleaners. Concentrating tables, flotation methods, and launder screens are used to prepare fine sizes.

UTILIZATION

When first discovered in the late 1700's, anthracite was used as a smithing coal, and about 1840 it began to replace charcoal in the manufacture of iron. The first oceangoing vessel designed to burn anthracite was the U. S. S. *Princeton*, built in Philadelphia in 1843. Anthracite was used to avoid the smoke that might betray its presence to an enemy. For over 100 years, however, anthracite has been utilized principally in heating homes, apartments, hotels, etc. Within a short radius of the region, anthracite is widely used by electric utilities in the production of power, and sizable quantities are shipped considerable distances for use in blending with bituminous coal in the manufacture of metallurgical coke and fuel briquets, in sintering iron ore, and for various other industrial purposes. Although a substantial part of the home-heating market has been lost to fuel oil and natural gas, the industry, in an effort to check this decline, has encouraged development of fully automatic anthracite-burning equipment that is clean and economical for heating homes, apartments, etc. The anthracite industry is looking also to the industrial field for increased sales of its product. To aid the industry in this regard, research is being conducted (1) to create new industrial markets for the larger sizes and (2) to fine-grind the run-of-mine product further, creating a supply of fines to meet present industrial market demands.

TABLE 7.—*Standard anthracite specifications approved and adopted by the Anthracite Committee, effective July 28, 1947*

	Round test mesh, inches	Percent					
		Oversize, maximum	Undersize		Maximum, impurities[1]		
			Maximum	Minimum	Slate	Bone or ash[2]	
Broken	Through 4⅜				1½	2	11
	Over 3¼ to 3		15	7½			
Egg	Through 3¼ to 3	5			1½	2	11
	Over 2 7/16		15	7½			
Stove	Through 2 7/16	7½			2	3	11
	Over 1⅝		15	7½			
Chestnut	Through 1⅝	7½			3	4	11
	Over 13/16		15	7½			
Pea	Through 13/16	10			4	5	12
	Over 9/16		15	7½			
Buckwheat No. 1	Through 9/16	10					13
	Over 5/16		15	7½			
Buckwheat No. 2 (Rice)	Through 5/16	10					13
	Over 3/16		17	7½			
Buckwheat No. 3 (Barley)	Through 3/16	10					15
	Over 3/32		20	10			
Buckwheat No. 4	Through 3/32	20					15
	Over 3/64		30	10			
Buckwheat No. 5	Through 3/64	30	No limit				16

[1] When slate content in the sizes from Broken to Chestnut, inclusive, is less than above standards, the bone content may be increased by 1½ times the decrease in the slate content under the allowable limits, but the slate content specified above shall not be exceeded in any event.

A tolerance of 1 percent is allowed on the maximum percentage of undersize and the maximum percentage of ash content.

The maximum percentage of undersize is applicable only to anthracite as it is produced at the preparation plant.

"Slate" is defined as any material that has less than 40 percent fixed carbon.

"Bone" is defined as any material that has 40 percent or more but less than 75 percent fixed carbon.

[2] Ash determinations are on a dry basis.

The finer sizes are sold, in many instances, in direct competition with residual oil and bituminous coal. As a means of meeting such competition, attention has been directed to lowering the mining costs by and through the introduction of highly developed mechanical mining equipment, new methods of roof control for flat-bed measures, and new and more efficient methods for mining the thick, steeply pitching beds of the Southern and Western Middle fields.

RESEARCH

The Anthracite Institute is the trade association of the anthracite-mining industry of northeastern Pennsylvania. The institute was organized in 1925 to supply the need for a combustion-engineering service in the domestic, commercial, and industrial fuel markets of the United States and Canada. Data regarding combustion tests of various fuels and equipment are furnished to anthracite retailers and other interested parties. Further research is carried on for the benefit of those primarily interested in methods and costs of heating their homes, and in the interests of those chiefly concerned in obtaining safe and reliable automatic heating systems requiring minimum attention. The Anthracite Institute compiles and publishes important trade statistics for the use of members of the industry and keeps the membership informed of important developments affecting the industry.

The efforts of the institute, on behalf of the industry as a whole, are being supplemented by progressive action of the individual producing companies. In some instances, fellowships have been established at such research institutions as the Mellon and the Battelle Memorial Institutes. Since 1932 Pennsylvania State University has conducted a program of research sponsored jointly by the Commonwealth of Pennsylvania and the anthracite industry through the Anthracite Institute. This program has included fieldwork in water-gas manufacture as well as laboratory work in various technical preparation problems and the determination of trace elements in anthracite, such as germanium.

Since its establishment in 1910, the Federal Bureau of Mines has made analyses and combustion tests on anthracite and has endeavored to improve health and safety in anthracite mining. In 1944 the Bureau undertook extensive investigations relating to anthracite-industry problems. By act of the Congress in 1942, erection of an Anthracite Research Laboratory at Schuylkill Haven, Pa., was authorized. It was specifically stated that this laboratory was to serve as a center of information on matters pertaining to the conservation of anthracite reserves; efficient mining, preparation, and utilization; development of new uses for anthracite; promotion of safety, health, and sanitation in anthracite-mining operation; and solution of special problems that eventually may be assigned to the Bureau of Mines. At present, research is in progress covering underground mine-mechanization equipment and roof-control methods; the crushing and grinding of coarse sizes to produce the maximum amount of desirable fine sizes; the cleaning of fine coal; the use of large-size anthracite in the metallurgical industry; and the gasification of anthracite for industrial use. A survey is also being made of market possibilities for special uses, including frost control and crop drying. Bureau of Mines activities are carried on in cooperation with various technical industry advisory committees.

OUTLOOK

Although the demand for anthracite has decreased greatly during the past 2 decades, the potential market for this fuel within a radius of 150 miles of the anthracite fields is substantial. The New England and Middle Atlantic States are highly industrialized and consume huge quantities of fuels in the form of coal, oil, and natural gas. The lowering of anthracite prices would place this fuel in a much stronger competitive position with respect to other fuels. A large part of the research conducted by the Bureau of Mines through its Anthracite Research Laboratory has been devoted to improving mining methods so as to bring about a greater output per man per day, thus paving the way for lower prices to consumers. Extensive research is also necessary on new uses for anthracite, especially in metallurgy and gasification. The latter is a most promising possibility for future consumption of large tonnages of anthracite for the production of high-B. t. u. gas and synthesis gas for chemicals and oils.

PROBLEMS

Many problems confronting the anthracite industry, such as the necessity of increasing the tonnage output per man per day, of finding new uses, and of expanding current markets have been discussed briefly in this publication. In addition to the problems discussed, the industry faces possible inundation of coal reserves by water impounded in underground pools and by infiltration of tremendous quantities of water into the mines from surface streams and pools. The many tons of water pumped to the surface for each ton of anthracite mined add materially to the cost of production. The need for a solution to the drainage problem is acute and is being studied intensively by the Commonwealth of Pennsylvania, the anthracite industry, and the Federal Government.

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ANTIMONY

By

Abbott Renick[1]

ANTIMONY is often regarded as a nuisance from the metallurgical standpoint, but it has many industrial and military applications.

Summary

An adequate supply of antimony is vital for national defense. Antimony is essential as a metal for the production of antimonial lead for storage-battery plates, bearing-metals alloys, type-metal alloys, and type metal and as antimony trioxide (Sb_2O_3) for making ceramic enamels and paints and for flameproofing plastics and textiles. The last use has special significance from a military standpoint.

The total world production of antimony from 1943 to 1953, inclusive, is estimated at 441,000 metric tons. During this period the United States consumed about 174,000 metric tons of primary antimony. The principal producing countries were Bolivia, Mexico, and the Union of South Africa. Mines in the United States made a significant contribution to the world total during and immediately after World War II. Until the start of the Sino-Japanese War in 1937, China was by far the world's largest producer and supplied nearly two-thirds of the world output. China's reserves constitute well over half of the known world reserves, estimated at about 3½ million metric tons of antimony metal.

The principal Free World consuming countries are the United States, the United Kingdom, France, Belgium, Japan, Italy, and Germany.

All of these major consuming countries have smelting industries. The United States now has the largest smelter capacity for the production of antimony metal and oxide; it also has large resmelting and secondary production capacity to meet the requirements for oxide and secondary antimony metal.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

HISTORY

Antimony occurs in many ore minerals. The most common ores, with their compositions, are:

Name of mineral:	*Formula*	*Antimony, percent*
Stibnite	Sb_2S_3	71.4
Stibiconite	$H_2Sb_2O_5$	74.5
Senarmontite	Sb_2O_3	83.3
Cervantite	$Sb_2O_3Sb_2O_5$	78.9
Valentinite	Sb_2O_3	83.3
Livingstonite	$HgS{\cdot}2\ Sb_2S_3$	53
Jamesonite	$Pb_2Sb_2S_5$	29.4

Antimony is one of the longest known metals, and the natural sulfide was familiar to ancient peoples. Women of Biblical times used it as a medicine and as a cosmetic for eyebrow painting. A vase found in Chaldea, cast in metallic antimony, discloses the fact that about 4000 B. C. the Chaldeans knew the art of reducing the metal. A copper ewer and basin of the Fifth or Sixth Dynasty (about 2500–2200 B. C.) discovered in Egypt, covered with a thin coating of metallic antimony, have shown that the ancient Egyptians knew something of the art of surface plating.

Basil Valentine, a German monk of the 15th century, and Agricola wrote of liquation, precipitation, and "starring" of regulus, the refined antimony metal. In the 15th century, antimony was in use for printers' type and in mirrors and bells. The 16th century witnessed its use as a medicine. Precipitation of metal from the sulfide by iron is described in writings of the 17th century (Ercker), and in the 18th century roast-reduction procedure came into use. The early 1830's were marked by the reverberatory furnace; 1844, by the French volatilization process; and 1896, by the first appearance on the market of electrolytic antimony.

Antimony is a strategic metal, as the mines of the United States yield relatively small quantities of antimony ores, and byproduct recovery from other ores is not large. China once was the principal source of the 10,000 to 18,000 tons of metal required each year by United States industry in peacetime. However, since the beginning of World War II, antimony supplies from China have been curtailed, and the United States has been forced to rely largely upon Mexican, Bolivian, South African, and domestic sources. The United States supplies of antimony were greatly improved by development of the Yellow Pine mine at Stibnite, Idaho, and by installation of augmented smelting facilities for treating foreign ores.

Antimony ores range from the low grades requiring concentration in milling operations to high-grade stibnite containing 70 percent antimony. "Chemical-grade" ore, so-called because it is desired in the production of such near chemically pure products as "regulus", oxide, sulfide, and other chemical compounds, contains a minimum of 60 percent antimony and a maximum of 0.5 percent lead and arsenic (impurities) combined. The low grades of ores are used in the production of antimonial lead products. In this case, lead is not considered an impurity in the ore, and although many grades are frequently used for this purpose, "metallurgical-grade" ore is classified as containing a minimum of 40 percent antimony and a maximum of 1 percent arsenic and copper (impurities) combined and is preferred. These definitions apply mainly to sulfide ores. Oxide ores with lower antimony content are used for both purposes, provided they are within the maximum impurities respectively stated.

The intermediate primary forms of antimony are metal, oxide, and sulfide. Antimony metal, regulus at least 99 percent pure, is a hard, silver-white, crystalline metal and is used to impart hardness and stiffness to soft metals, particularly lead, with which it is usually alloyed. Upon solidification, a lead-antimony-bismuth alloy expands, a property that makes it ideal for manufacturing such items as castings and type. Chemical-grade ore is preferred for metal production, although large quantities of mixed or oxide ore, which have wide ranges of antimony content and varying impurities (except lead), are used.

Antimony oxide is theoretically 83.3 percent antimony, but commercial grades range between 80 to 85 percent. Oxide produced from some ores may have objectionable percentages of an acid radical that would make it unsuitable for production of flameproofing compounds and some paints; however, such oxide may be used for other purposes, such as frits and ceramic enamels, sulfide, glass and pottery, and miscellaneous chemicals.

Antimony sulfide contains about 70 percent of antimony and is of 2 kinds—"needle" or "liquated" and precipitated. Needle or liquated is produced by melting direct from sulfide ores, whereas precipitated is produced from either ore or oxide by precipitation.

As a consequence of foreign importations, the United States mine production of antimony has declined to less than 10 percent of the domestic requirements. This country, as the leading world consumer, must import most of its supply of primary antimony, however, under wartime pressure, record production was established for 1942, 1943 (alltime high), and 1944. The Yellow Pine mine at Stibnite, Idaho, operated by the Bradley Mining Co., in 1951 produced about 75 percent of the domestic mine output (including antimony contained in lead-silver ores, which, however, cannot be utilized in nonmetallic uses, such as for flameproofing requirements) largely in the form of high-grade oxide. The Bradley Mining Co., however, suspended operations in midyear 1952 because of unfavorable market conditions. In addition to the ore production, the United States annually produces, as a byproduct of lead smelters, about 3,000 tons of primary antimonial lead from primary lead ore, residues, and bullion.

The new supply of primary antimony in 1953, in terms of recoverable metal, was 14,000 short tons. A breakdown of this supply shows that domestic antimony ores contributed 2 percent (340 tons—in terms of recoverable metal content calculated at 92 percent of mine and mill production); domestic and foreign lead-silver ores, 20 percent (2,800 tons); and imports 78 percent (10,900 tons). The antimony imported arrived as follows: Ore and concentrates, 7,200 tons; metal, 2,600; and antimony oxide, 1,100. The supply from secondary sources was 22,000 short tons.

In addition to the antimony obtained from antimony ores, 1,600 to 2,800 tons of antimony is recovered from certain lead and lead-silver ores. These ores come from both domestic and foreign sources, although the foreign material does not appear as imports of antimony because it is imported as lead ore.

OCCURRENCES

Antimony deposits may be classified into 2 rather distinct general types, but gradations exist between the 2. The first is the simple type, consisting primarily of antimony minerals in a siliceous or carbonate gangue with only small or negligible quantities of other metals. In almost all deposits the original antimony mineral is stibnite, the antimony sulfide, but native or metallic antimony is occasionally found. Where the ores have been exposed to extensive oxidation, the original stibnite has been entirely converted to antimony oxides. Most ores of Mexico and Algeria are of this type. Many other ores consist of a mixture of stibnite and antimony oxides. Commonly the simple type of antimony deposit is also characterized by rather simple geologic relations, but there are many exceptions. Most of the deposits of Mexico, Bolivia, Peru, China, Yugoslavia, Algeria, Hungary, Czechoslovakia, Italy, and Japan are of the simple type.

The second type of deposit is mineralogically complex and very commonly is also structurally complex. In many places stibnite is the antimony-bearing mineral, but often the metal is only one component of minerals that also contain lead, copper, silver, and mercury. In addition, in most deposits of the complex type, other non-antimony-bearing metallic minerals are present and are commonly more abundant than the antimony-bearing minerals. With few exceptions, the ores are mined primarily for lead, gold, silver, mercury, zinc, copper, or tungsten. The antimony may be a byproduct of only minor value; or, in some instances, the value of the ore decreases because of its antimony content. In such ores the production of antimony may bear little relation to its price. Much or most of the antimony that is produced in the United States, Australia, Union of South Africa, and Canada is from the complex-type deposit.

A few deposits, such as those of the Yellow Pine district in Idaho, contain enough antimony so that its value exceeds that of other metals in the ore when the price of antimony is high; but when it is low, antimony becomes secondary to the value of other metals. Where a choice of ores containing different proportions of metals is available, price may be a major factor in determining the rate of production.

More antimony was produced in Bolivia than in any other country during World War II. The annual output of Bolivian ore from 1939 to 1953, inclusive, averaged over 11,000 tons of contained antimony, most of which was shipped to the United States. The greater part of the ore is produced from small workings by native operators. Inferred reserves are large and are believed adequate for some decades at the current rate of production. Bolivia is capable of expanded output if stimulated by moderately high prices.

Mexico has been the dominant source of antimony in North America for many years, generally supplying at least two-thirds of the total. Although China and Bolivia have usually exceeded Mexico, their outputs have been erratic and subject to many uncertainties. Mexico's production has also varied greatly, but the extent of the variations has not been as pronounced because of its favored position in the United States market. This position is due largely to proximity to the United States and to the fact that the ores are transported by rail to the Laredo, Tex., smelter and do not depend upon long and at times uncertain ocean transport.

The United States is, without question, highly dependent upon Mexico and Bolivia for antimony, and consequently a strong effort should be made to insure a healthy condition of the

TABLE 1.—*World production of antimony (content of ore),*[1] *by countries, 1944–48 (average) and 1949–53*

[Metric tons]

(Compiled by Pauline Roberts)

	1944–48 (average)	1949	1950	1951	1952	1953
North America:						
Canada [2]	518	72	292	591	1, 134	694
Honduras	21	9	(3)	(3)	----------	----------
Mexico [2]	8, 112	5, 753	5, 868	6, 824	5, 531	3, 686
United States	3, 806	1, 484	2, 265	3, 150	1, 960	337
South America:						
Argentina	28	(4)	(4)	(4)	(4)	(4)
Bolivia	[5] 8, 613	[5] 10, 275	[5] 8, 781	[5] 11, 816	[5] 9, 806	[6] 2, 200
Peru	1, 390	729	971	1, 107	505	(4)
Europe:						
Austria	258	[7] 379	[7] 409	[7] 498	[7] 389	[6] [7] 450
Czechoslovakia	2, 811	(4)	2, 000	(4)	(4)	(4)
France	198	338	413	611	470	(4)
Greece	----------	49	350	500	350	(4)
Hungary	[6] 310	(4)	(4)	(4)	(4)	(4)
Italy	457	503	671	794	779	[6] 470
Portugal	22	21	15	19	141	(4)
Spain	181	259	200	167	261	[8] 600
Yugoslavia	[6] 1, 450	2, 789	3, 205	1, 973	2, 237	[6] 1, 840
Asia:						
British Borneo	1	1	2	----------	----------	----------
Burma [6]	371	70	40	200	(4)	(4)
China	1, 158	[6] 4, 000	[6] 6, 000	[6] 8, 000	[6] 8, 000	[6] 8, 000
Indochina	6	----------	----------	----------	----------	----------
Iran	1	[9] 175	[9] 230	[6] [9] 230	160	(4)
Japan	222	172	161	221	209	[6] 320
Thailand	62	265	87	65	70	(4)
Turkey	184	460	1, 288	2, 166	1, 400	(4)
Africa:						
Algeria	304	1, 326	1, 250	1, 462	1, 321	[6] 1, 860
French Morocco	343	700	689	957	839	[6] 50
Southern Rhodesia	53	41	24	62	100	[6] 25
Spanish Morocco	134	150	353	213	431	(10)
Union of South Africa	3, 038	4, 461	8, 311	15, 858	7, 211	[6] 3, 600
Oceania:						
Australia	316	198	227	420	243	[6] 70
New Zealand	1	3	----------	----------	6	(4)
Total (except U. S. S. R.)	34, 000	37, 000	44, 000	61, 000	45, 000	29, 000

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[1] Approximate metal content of ore produced, exclusive of antimonial lead ores.
[2] Includes antimony content of antimonial lead.
[3] Negligible.
[4] Data not available; estimate included in total.
[5] Exports.
[6] Estimate.
[7] Excludes Soviet zone, data for which are not available. Estimates included in total.
[8] Includes Spanish Morocco.
[9] Year ended March 20 of year following that stated.
[10] Included in Spain.

antimony industry in those countries. These countries afford the only relatively dependable major sources of primary metal outside of the United States, and domestic production cannot provide more than a small part of essential requirements, even under the most favorable circumstances.

Consolidated Murchison (Transvaal) Goldfields & Development Co., Ltd., the only South African antimony producer in Transvaal, in 1953 confined development to a gold lens; broken stibnite ore was drawn from stopes as required. Cobbed ore and concentrate stocks increased, but overall output decreased to 4,773 short tons (12,958 tons in 1952); sales were 9,450 short tons (11,229 tons in 1952). It is believed that this rate of production can be increased and maintained in future. Antimony production is currently curtailed solely because the market cannot absorb full production. The Murchison property could supply the United States with at least 12,000 tons of concentrates annually, containing about 7,000 tons of antimony.

At present the only important source of antimony in the United States is the byproduct yielded in the treatment of ores of silver, copper, lead, and zinc. In such operations, production of antimony is less erratic, depends upon the mining of the other metals, and is relatively independent of the price and need for antimony. The Coeur d'Alene district, Idaho, is an outstanding example of this type.

A number of ore deposits, consisting essentially of quartz, stibnite, and antimony oxides, with only small or negligible contents of gold, silver, and other metals, are known in a number of the Western States. These deposits are mined primarily for their antimony content; therefore, production from them is closely related to the price of antimony; most of them are very small and individually are not reliable sources for the metal.

Table 2 shows salient statistics for antimony in the United States, 1944–53.

United States chief antimony-mine producers and smelters

Mines:
- Bradley Mining Co., Stibnite, Idaho.
- Sunshine Mining Co., Kellogg, Idaho.

Smelters:
- American Smelting & Refining Co., Omaha, Nebr.
- American Smelting & Refining Co., Perth Amboy, N. J.
- Bradley Mining Co., Stibnite, Idaho.
- Harshaw Chemical Co., Gloucester City, N. J.
- McGean Chemical Co., Cleveland, Ohio.
- Metal & Thermit Corp., Piscatawaytown, N. J.
- National Lead Co., Laredo, Tex.
- National Lead Co., Evans Lead Div., Charleston, W. Va.
- Sunshine Mining Co., Kellogg, Idaho.

Antimonial lead smelters and refiners:
- American Smelting & Refining Co., Omaha, Nebr.
- American Smelting & Refining Co., Selby, Calif.
- American Smelting & Refining Co., Perth Amboy, N. J.
- American Smelting & Refining Co., East Helena, Mont.
- American Smelting & Refining Co., Leadville, Colo.
- American Smelting & Refining Co., Federal, Ill.
- American Smelting & Refining Co., El Paso, Tex.
- Bunker Hill Smelter, Kellogg, Idaho.
- Eagle-Picher Co., Galena, Kans.
- Goldsmith Bros. Smelting & Refining Co., Chicago, Ill.
- St. Joseph Lead Co., Herculaneum, Mo.
- U. S. S. Lead Refinery, Inc., East Chicago, Ind.
- U. S. Smelting, Refining & Mining Co., Midvale, Utah.

MINING METHODS

The nature of the occurrence of antimony deposits presents a problem in ore extraction. Most of the ore bodies are small, irregular, and scattered and do not lend themselves readily to exploitation by the large-scale methods so prevalent in mining many other metals in the United States.

Typically, the mining of antimony ores is carried on by hand methods, especially in China, Mexico, and Bolivia.

SMALL-SCALE MINING OPERATIONS

The small-scale or intermittent mining of antimony ores is accomplished through careful, selective hand mining, ore and waste being broken separately to prevent dilution. Mining must then be followed by hand sorting to produce an ore of smelting grade if concentration facilities are not available.

TABLE 2.—*Salient statistics of antimony in the United States, 1944–53*

[Short tons, antimony content]

	1944	1945	1946	1947	1948	1949	1950	1951	1952	1953
Production:										
Primary:										
Mine (shipments)	4, 735	1, 930	2, 505	5, 316	6, 489	1, 636	2, 497	3, 472	2, 160	372
Smelter	20, 000	21, 000	12, 422	13, 782	14, 308	8, 099	9, 471	13, 800	11, 860	7, 100
Secondary	15, 886	17, 148	19, 115	22, 984	21, 592	18, 061	21, 862	23, 943	23, 089	22, 000
Imports for consumption	17, 373	23, 270	8, 496	15, 153	17, 038	9, 429	15, 354	15, 673	12, 789	11, 478
Ore and concentrates	17, 080	22, 643	5, 903	9, 257	13, 464	7, 473	9, 746	11, 746	7, 945	7, 778
Metal	293	627	2, 593	5, 879	3, 201	1, 853	4, 632	2, 231	3, 354	2, 612
Oxide	------	------	------	------	------	46	963	1, 692	1, 466	1, 076
Sulfide	------	------	------	12	373	57	13	4	24	12
Exports of ore, metal and alloys [1]	745	333	462	808	327	485	154	168	161	24
Consumption of primary antimony [2]	23, 756	25, 761	17, 515	16, 647	15, 455	11, 266	15, 167	17, 370	14, 988	14, 300
Average price of antimony at New York ...... cents per pound [3]	15. 84	15. 84	17. 31	33. 45	36. 67	38. 73	29. 41	44. 17	44. 02	35. 90
World production [4]	39, 700	29, 800	28, 700	42, 000	49, 600	40, 800	48, 500	67, 200	49, 600	32, 000

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[1] Gross weight.

[2] Excludes antimony contained in domestic and foreign silver-lead ores, recovered at primary lead refineries and marketed as antimonial lead.

[3] American Metal Market.

[4] Exclusive of U. S. S. R.

Even in the larger antimony districts of China, ore is broken largely by hand drilling and blasting and is then hand-sorted both at the working face and on the surface. Such methods are employed owing to the irregularity of the deposits and an abundance of low-wage labor. In recent years there has been substantial progress toward mechanization, involving the use of power drilling and modern ore-dressing plants.

It is customary in the United States and elsewhere that irregular deposits, too small or scattered to be mined by mechanical methods, be mined by hand on a piecework or "buscon" basis—compensation directly based on individual production. In addition, parts of old mines, where company operations have become marginal, are leased to groups of miners on a production rather than on a wage basis.

LARGE-SCALE MINING OPERATIONS

Where the scale of operations and regularity of production warrant, the development and operation of antimony properties follow, to a limited degree, the standard principles and procedures that prevail in the broader metal-mining fields (copper, lead, and zinc), that is, the usual cycles of drilling, blasting, and removal of broken ore, shaft sinking, crosscutting, drifting, stoping, pumping, and ventilation.

The Bradley Mining Co. Yellow Pine mine at Stibnite, Idaho, is the only large-scale open-pit antimony mine in the United States. This mine is a gold-tungsten-antimony development and was converted to an open-pit operation to mine large reserves of antimony-gold-silver ore. The property was equipped to produce 2,400 tons of ore per day and included a crushing plant of 300-ton-per-hour capacity at the pit. Diesel-powered trucks of 15 to 20 tons capacity hauled the ore from the mine to the mill—2½ miles.

A few antimony mines in Mexico, Nevada, and elsewhere have been operated by systematic underground methods on a moderate scale.

METALLURGY

CONCENTRATION

Pyrometallurgical methods of extracting antimony from its ores were used almost exclusively until the last few years, when electrometallurgical methods received considerable attention in the United States. Ore dressing for concentrating low-grade ores has been used only in a very limited way for antimony, but both experimental work and some practical plant results indicate increased use of such methods in the future. This applies particularly to flotation. One important advantage in beneficiating low-grade antimony ores is removal of arsenic at an early stage. Flotation plants have been operated for antimony concentration in various localities, including the Djinti-Kaya mine in Turkey, the Wiluna gold mine in Australia, the Cucama mine in Czechoslovakia, and the Yellow Pine mine, Stibnite, Idaho, to cite examples. Wet concentration with jigs or float-sink equipment has the disadvantage of high slime loss. By normal tabling this disadvantage is at least partly overcome, and dry concentrating tables show promise, as judged from experimental results. The antimony content of commercial ores ranges from 1 to 60 percent and determines the method of treatment for recovering the metal. In general, the lowest grades of sulfide ores, 5 to 25 percent antimony, are volatilized, 25 to 45 percent antimony ores are smelted in a blast furnace, and 45 to 60 percent antimony ores are liquated or treated by iron precipitation. The blast furnace is generally used for mixed sulfide and oxide ores and oxidized ores containing up to about 40 percent antimony; the direct-reduction method is used for the richest oxide ores.

OXIDE VOLATILIZATION

Removal of the antimony from the ore as the volatilized trioxide is the only pyrometallurgical method suitable for low-grade ore. The sulfide content of the ore supplies some of the heat; hence, the fuel requirement is not great. Arsenic oxide can be substantially removed because it is more volatile than antimony trioxide. There are many variations of the volatilization process, the principles employed being the same but the equipment differing. In every instance, the sulfur is burned away (roasting), the trioxide formed and volatilized, and the oxide recovered in flues, condensing pipes, a baghouse Cottrell precipitator, or combinations of these means. Roasting and volatilization are effected almost simultaneously by heating the ore in contact with charcoal, coke, or petroleum coke, with controlled aeration, in such equipment as a shaft furnace, a rotary kiln, a converter, or a blast roaster. Usually the oxide produced in this manner is impure and is used for reduction to metal, sometimes after briquetting. A comparatively high grade oxide is, however, obtainable by direct treatment of ores, with special attention to choice of charge, volatilization conditions, and selection of product.

LIQUATION

In the treatment of comparatively rich sulfide ores, antimony sulfide is readily but inefficiently separated from the gangue by heating to 500° to 600° C. in a perforated pot and collecting the molten sulfide in a lower container. A more efficient method is to use a reverberatory furnace and liquate continuously, but this has the difficulty of needing an atmosphere reducing enough to prevent oxidation and loss by volatilization if the trioxide is not all caught or by formation of stable tetroxide in the residue. The residue, containing 12 to 30 percent antimony, is usually treated by the volatilization process to recover additional antimony. The liquated product, called "crude" or "needle" antimony, is sold as such for applications of antimony sulfide, or it is converted to metallic antimony by carefully roasting it to the oxide and reducing in a reverberatory furnace or by iron precipitation.

SMELTING

Metallic antimony is recovered from the concentrated trioxide produced by volatilization methods, from stable tetroxide produced by dead-roasting rich ores or concentrates, from the liquated sulfide, or from intermediate grades of ores by direct smelting. The last is the newest method. Reduction of antimony oxides in crucibles has been succeeded by larger scale, more efficient reduction in reverberatory furnaces. An easily fusible alkaline flux, as of soda, potash, and Glauber's salt, is commonly used to minimize volatilization and dissolve residual sulfides. Part of this slag is frequently reused in subsequent charges. Loss of antimony from the charge by volatilization is high—as much as 12 to 20 percent—even with use of ample slag and careful control, and it may be higher. This makes necessary the use of effective Cottrell precipitators or baghouses and considerable runaround of oxide.

IRON PRECIPITATION

Rich sulfide ore or liquated antimony sulfide (crude antimony) is reduced to metal by iron precipitation in some localities, as in England, Hungary, Japan, and Mexico. This method is based on the greater attraction of sulfur for iron than for antimony and consists essentially of heating the molten antimony sulfide in crucibles with slightly more than the theoretical amount of fine-iron scrap required to remove the sulfur. Sodium sulfate and carbon are added to produce sodium sulfide, or salt is added to form a light fusible matte with the iron sulfide and facilitate separation of the metal. Since the metal so formed contains considerable iron and some sulfur, a second fusion with some liquated antimony sulfide and salt is made for purification.

BLAST-FURNACE SMELTING

Water-jacketed blast furnaces are being used in several modern plants to handle intermediate (25 to 40 percent) grades of oxide or sulfide or mixed ores, liquation residues, mattes, rich slags, and briquetted fines or flue dusts. In general, lead-blast-furnace practice is followed. A high smelting column, comparatively low air pressure, and separation of slag and metal in a forehearth are used. Considerable slag is made and is desired, since it tends to reduce volatilization losses. The slag usually runs under 1 percent antimony.

The world's largest antimony smelter, at Laredo, Tex., uses blast-furnace reduction in making a metal of high purity and supplies much of the antimony used in the United States.

ELECTRIC-FURNACE SMELTING

At Stibnite, Idaho, the principle reduction unit in the smelter is an electric furnace, where a crude bullion is produced and the waste products are removed in a slag. Further refining is accomplished in oil-fired furnaces, where iron, sulfur, and arsenic are removed. The refined antimony metal is converted to antimony oxide in additional furnaces.

PROPERTIES AND USES OF ANTIMONY

Antimony is a brittle, silvery-white metal having a specific gravity of 6.68 and a melting point of 630° C. It is similar in many properties to arsenic and bismuth. Elemental antimony may exist in two allotropic forms—the metastable yellow or alpha form and the rhombohedral beta form. The metastable form is electrodeposited from an antimony trichloride solution as alpha antimony containing the trichloride in solid solution. This solid solution is known as "explosive antimony" because, if heated or scratched, it transforms suddenly to the beta form, giving off clouds of antimony trichloride.

The primary use of primary antimony is for hardening lead. Hard lead serves a variety of uses.

ALLOYS

BATTERIES

The largest use of antimony, normally about 35 percent of the total United States consumption, is in the manufacture of grids in lead storage batteries, used principally in automobiles. The antimony content of antimonial lead used in storage-battery grids ranges from 5 to 11 percent, with an average composition of 7 to 9 percent. Antimonial lead, with an antimony content of about 4 percent, is used for the terminals.

BEARINGS

Antimony required for manufacturing bearing metal totals about 20 percent of the annual consumption in the United States. Bearing metal, sometimes known as white metal, is of two types—white tin-base and lead-base; the former is known commercially as "babbitt metal" or "babbitts." Babbitt metal generally contains less than 10 percent antimony and the lead-base white metal less than 15 percent antimony.

TYPE METAL

Type metal is an alloy, principally of lead, antimony, and tin, containing up to about 30 percent antimony.

BRITANNIA METAL AND PEWTER

Britannia metal is essentially a tin alloy, with antimony in variable proportions and smaller quantities of copper, zinc, lead, bismuth, or other metals. Pewter, an alloy of tin-antimony-copper, is similar to high-grade Britannia metal.

CABLE SHEATHINGS

Cable sheath for telephone and electric cables is ordinarily made of lead hardened with about 1 percent antimony. In a normal year about 1,000 short tons of antimony in antimonial lead is required for this use.

ANTIMONIAL LEAD

Antimony is commonly used to harden lead; however, recent improvements in battery-grid alloys are represented chiefly by reduction in antimony content and increase in arsenic and silver content. Arsenic improves resistance to growth and corrosion and slightly increases mechanical strength. Silver has somewhat the same effect. The antimony content of antimonial hard lead depends upon the use to which the metal is put and ranges from 6 percent or lower for rolled sheets for linings and roofing to 28 percent for special purposes, such as nozzles.

The chemical specifications of antimonial lead purchased for the National Stockpile follows:

	Percent
Antimony	10.70 to 11.30
Arsenic	.20 maximum
Bismuth	.05 maximum
Copper	.10 maximum
Tin	.15 to .55
Nickel	.005 maximum
Iron	.01 maximum
Silver	.03 maximum
Sulfur	.005 maximum
Lead	Balance

COMPOUNDS

The principal industrial compounds of the metal are the trioxide, Sb_2O_3; the sulfide, Sb_2S_3; the sodium antimonate, $Na_2H_2Sb_2O_7$; and the chloride, $SbCl_3$. Peacetime uses of these are mainly in ceramic enamels, paints and lacquers, plastics, glass, and flameproofed textiles.

ENAMELING

As an opacifying agent for white enamels, antimony trioxide and, to a larger extent, zirconium oxide, titanium oxides, and Uverite (a calcium-antimony fluotitanate) have almost completely replaced tin oxide, mainly because of the high price of the latter. Moreover, antimony compounds are, in certain respects, superior to tin compounds. Antimony oxide lessens the blistering defects that frequently come through use of tin oxide and gives a more brilliant finish to the enamel. Sodium antimonate is used in opaque glass and in cast-iron enamels.

The use of the oxide for these purposes is due to its refractive index, as follows:

Oxide:	*Refractive index*
Sb_2O_3	2.09
SnO_2	2.04
ZrO_2	2.13–2.20
TiO_2	2.50

PLASTICS

Antimony oxide has found widespread usage as a stabilizer and flame-retardant in plastics (including polyester, polyethelene, polyvinyl, etc.).

PIGMENTS

"Antimony white" is the trioxide of antimony, which is formed as a very fine powder in the flues and dust chambers of antimony-roasting furnaces. It can be collected in this form and used without further treatment for certain paint purposes, but a specially ground and prepared oxide, called Timonox, of superior texture, opacity, and whiteness, is produced in England and by the National Lead Co., at Laredo, Tex. It has become useful in the pigment field. Other domestic brands having similar qualities include antimony oxide, KR brand (Harshaw Chemical Co.) and Elk brand (Bradley Mining Co.).

"Antimony black" is metallic antimony deposited electrically or chemically as a fine powder from an antimony solution. It is used as a bronzing pigment for metals and plaster casts.

"Antimony vermilion," a red trisulfide of antimony, may be formed by precipitating an antimony salt from solution with sulfureted hydrogen. As a pigment, it is in some ways superior to red oxide of lead, red chromate of lead, or mercury vermilion.

"Antimony yellow" is produced by slow oxidation of the sulfide; various shades of yellow are formed by mixing with red lead and zinc white.

"Antimony blue," "antimony violet," etc., are other pigments produced by mixing antimony yellow with other mineral compounds.

RUBBER VULCANIZING

Manufacturers of antimony sulfuret, for use in the rubber industry, customarily use the words "golden" and "crimson" to indicate pentasulfide and oxysulfide, respectively, and "golden-crimson" when both are present.

MORDANTS

Tartar emetic and antimonine are the principal compounds of antimony used as mordants.

AMMUNITION PRIMERS

A small tonnage of antimony sulfide is used annually as a constituent in military- and sporting-arms primers and as a smoke producer for smoke screens, range-finding shells, and tracer bullets.

FIRE-RESISTANT MATERIALS

The largest single use of nonmetallic antimony in recent years is for preparing an antimony compound for rendering certain essential war and industrial fabrics fire-resistant (tenting, truck tarpaulins, and papers for industrial use). This use employed approximately 10,000 tons of the metal a year during World War II. Materials treated with the compound quickly extinguish flames that accompany initial combustion by virtue of an automatic chemical reaction as the compound itself is heated. United States Patent 2,299,612 (October 1942) gives the composition of such a compound:

	Percent
Chlorinated paraffin containing 30 to 50 percent chlorine	25
Antimony trioxide (Sb_2O_3)	25
Petroleum solvent (Stoddard solvent)	50

In United States Patent 2,427,997 (September 1947) antimony chloride ($SbCl_3$) is used

TABLE 3.—*Industrial consumption of primary antimony, 1944–48 (average) and 1949–53*

[Short tons, antimony content]

Product	1944–48 (average)	1949 [1]	1950	1951	1952	1953 [2]
Metal products:						
Ammunition	55	6	9	4	3	3
Antimonial lead	6, 586	2, 588	4, 440	2, 282	2, 196	2, 300
Battery metal	(3)	1, 521	1, 738	2, 774	2, 253	3, 000
Bearing metal and bearings	2, 441	873	1, 518	1, 308	1, 119	1, 000
Cable covering	180	172	72	95	43	60
Castings	165	49	126	79	80	80
Collapsible tubes and foil	109	14	23	18	32	60
Sheet and pipe	266	306	300	180	70	170
Solder	151	155	162	123	145	200
Type metal	1, 244	587	766	709	624	700
Other	(3)	364	145	52	61	127
Total metal products	11, 197	6, 635	9, 299	7, 624	6, 626	7, 700
Nonmetal products:						
Ammunition primers	29	9	9	18	24	30
Antimony sulfide (precipitated)	(4)	(4)	(4)	68	67	50
Fireworks	(5)	(5)	(5)	20	36	50
Flameproofed coatings and compounds	(5)	(5)	(5)	463	980	450
Flameproofed textiles	3, 086	273	369	2, 590	2, 059	780
Frits and ceramic enamels	1, 313	1, 155	1, 462	1, 476	959	1, 000
Glass and pottery	368	296	579	570	579	700
Matches	23	28	56	31	22	20
Paints and lacquers	1, 965	874	267	962	853	340
Pigments	(5)	(5)	(5)	705	766	780
Plastics	(5)	498	737	747	632	560
Rubber products	(5)	55	103	19	66	20
Other [6]	1, 846	1, 443	2, 286	2, 077	1, 319	1, 820
Total nonmetal products	8, 630	4, 631	5, 868	9, 746	8, 362	6, 600
Grand total	19, 827	11, 266	15, 167	17, 370	14, 988	14, 300

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[1] Data exclude certain intermediate smelting losses, which are included for subsequent years.
[2] Estimated 100-percent coverage based on reports from respondents that consumed 89 percent of the grand total antimony in 1952.
[3] Included with "Antimonial lead."
[4] Not reported as an end-use product.
[5] Included with "Other nonmetal products."
[6] Antimony trichloride and sodium antimonate included to avoid disclosure of individual company operations.

instead of antimony trioxide. The fabric to be fireproofed is first impregnated with sodium carbonate solution, dried, and then impregnated with a 15-percent solution of antimony chloride in an organic solvent, which may be a petroleum or coal-tar solvent or alcohol. Certain new fireproofing pigments and compounds of antimony base, employed in ship construction and in the mastic industry, are finding expanding use. The demand expands enormously in wartime.

OTHER USES

Antimony is used in various forms in the manufacture of matches, medicine, fireworks, toys, etc. Table 3 shows United States consumption of antimony in short tons, 1944–48 (average) and 1949–53.

SECONDARY PRODUCTION

The production of secondary antimony annually contributes 50 to 60 percent of the total available supply in the United States. Although secondary antimony has always been an important part of the total supply, it has increased in importance over the past 25 years. During 1925–29 secondary antimony was approximately 40 percent of the total supply. During the postwar years 1946–49, it comprised 52 percent of the total supply. Table 4 presents secondary production for antimony in the United States in 1944–53.

TABLE 4.—*Secondary production of antimony in the United States, 1944–53*

Year:	*Thousand short tons*
1944	16
1945	17
1946	19
1947	23
1948	22
1949	18
1950	22
1951	24
1952	23
1953	22

Most secondary antimony is recovered as antimonial lead from lead-tin or lead scrap in the form of discarded storage batteries; however, large quantities are also recovered from babbitt metal, type metal, and antimonial lead drosses. This recovery process is carried on at secondary smelters, remelters, refineries, and, to a more limited extent, primary-lead refineries. The scrap treated at these plants is largely resmelted; some, depending upon the degree of purity of the scrap, is merely brought back to specification through the addition of pure lead, tin, or antimony.

Because secondary antimony is a minor constituent of lead scrap, its supply varies with the supply of lead scrap, and the price of the scrap is related more closely to the price of lead than to the price of antimony.

SUBSTITUTES

Although significant tonnages of antimony are used as an opacifier for enamels and in pigments for paint and lacquers, in recent years latest developments in the enamel industry have eliminated almost entirely the use of mill addition opacifiers by substituting highly opaque titania and zirconia enamels; titania or zirconia is added to the batch of raw materials and then smelted and fritted.

Antimony in type metal is a revolving supply but requires significant quantities of new metal. Plastics are, however, finding some use in printing and could possibly replace antimonial lead.

Lead-base babbitt requires considerable antimony but serves purposes for which tin-base would otherwise be required. Other lead-base metals could be substituted in part, but with little advantage. Pipe, sheet, and other products, in which antimony is now used as the hardening metal, can largely be replaced by lead-calcium and, to some extent, lead-tellurium alloys. Use of metallic antimony in cable sheathing is being somewhat reduced by the substitution of plastics (polyethylene) for 1-percent antimonial lead cable.

Storage batteries consume more antimony metal than any other use; however, it has been found that 0.06 percent calcium could be substituted for 8 percent antimony as the hardening agent to produce batteries that in telephone service last more than twice as long and have but a fifth of the power loss compared with antimony-lead batteries.

MARKET REQUIREMENTS

Antimony is usually marketed in units 10 by 10 by 2½ inches, variously known as blocks bars, and pigs. Nomenclature in this regard is apparently not fixed. The average weight is about 55 pounds to the bar.

Metallic antimony is sold under brand names of the producing companies, each brand having certain limits in impurities. The antimony content may range about 99.0 to 99.91 percent, with lead, arsenic, and sulfur the chief impurities. The most used American brands of metal are RMM (99.5 percent antimony with arsenic not to exceed 0.1 percent) and Lone Star (99.8 percent with arsenic guaranteed to be below 0.05 percent). Typical analyses show 99.8 and 99.9 percent antimony, respectively. The higher purity commands a half cent per pound premium. A very high purity antimony (99.97

percent) has been prepared by distillation, but no market has developed for it. All brands are sold as starred slabs or pigs.

When cast under a protective "starring slag" of the proper composition, antimony develops on its top surface a beautiful crystalline pattern known as "starring," although, in fact, the pattern more closely resembles fronds of palms or ferns.

PRICES

The price of antimony has been marked by notorious instability. The trend of antimony prices has been upward over a long period of years. During the World War II period the average price of antimony metal was about 15.5 cents. In 1953 quoted prices for antimony metal, New York City, varied between a low of 30.47 and a high of 36.47 cents per pound. The average price for the metal during 1953, according to the American Metal Market, was 35.90 cents per pound compared with 44.02 cents in 1952. Table 5 covers price fluctuations and averages of antimony metal in New York since 1910.

The selling prices of antimony ore have varied widely, indicating comparative sensitivity to demand. Antimony production—all phases of it, mining, smelting, and refining, but particularly mining—cannot be increased or decreased rapidly; hence, the problem of relating world production to demand has been serious, with consequent wide swings in price.

In 1953, $3.30 to $3.75 per unit was paid for ore of a minimum grade of 60 to 65 percent antimony. The price per unit follows a sliding scale, decreasing for lower grade ores.

The introduction of the term "unit" in the buying of certain ores has sometimes been confusing. The corresponding value per ton of the ore may be determined by considering that a "unit" represents each 1 percent of antimony in the ore. If, on analysis, the ore shows a content of 60 percent antimony and the price quoted is $3.75 per unit, the value will be (60×3.75) or $225.00 per ton of ore. By some, the term "unit" is used to represent each 1 percent of a ton or 20 pounds. Using the same quotation and percentages as above, at $3.75, an ore showing 60 percent antimony would be worth 60 percent of $3.75 or $2.25 for 20 pounds, equivalent to $225.00 per ton of 2,000 pounds.

According to the E&MJ Metal and Mineral Markets, the quotation for 60- to 65-percent antimony ores on December 2, 1954, was $4.35 to $4.45 per unit of 20 pounds.

TABLE 5.—*Antimony prices for metal*

[Cents per pound]

(Extreme price fluctuations and annual averages of ordinary grades of antimony in New York. Hungarian and French brands through February, 1915. Chinese brands, 1915 to 1934 inclusive and subsequently American brands)

Year	High	Low	Average
1910			8. 24
1911	9. 12½	6. 90	7. 48
1912	9. 50	6. 62½	7. 63
1913	9. 00	6. 00	7. 43
1914	18. 00	5. 30	8. 53
1915	40. 00	13. 00	29. 52
1916	45. 00	10. 50	25. 33
1917	36. 00	13. 62½	20. 73
1918	14. 75	7. 62½	12. 55
1919	9. 62½	6. 50	8. 16
1920	11. 62½	5. 20	8. 38
1921	6. 00	4. 45	4. 92
1922	7. 00	4. 15	5. 42
1923	10. 00	6. 25	7. 81
1924	17. 50	8. 25	10. 77
1925	24. 50	11. 00	17. 50
1926	25. 00	9. 50	15. 91
1927	15. 50	10. 25	12. 34
1928	11. 37½	9. 25	10. 30
1929	9. 87½	8. 25	8. 94
1930	9. 25	6. 75	7. 67
1931	7. 50	6. 00	6. 72
1932	7. 00	5. 00	5. 62
1933	7. 50	5. 40	6. 51
1934	14. 00	7. 15	8. 92
1935	16. 75	12. 50	13. 62
1936	13. 75	10. 62½	12. 25
1937	17. 37½	13. 75	15. 35
1938	13. 75	10. 75	12. 35
1939	14. 00	11. 25	12. 36
1940	14. 00	14. 00	14. 00
1941	14. 00	14. 00	14. 00
1942	16. 05	14. 00	15. 55
1943	16. 05	15. 84	15. 92
1944	15. 84	15. 84	15. 84
1945	15. 84	15. 84	15. 84
1946	29. 62½	15. 79	17. 31
1947	34. 53	29. 55	33. 45
1948	40. 17	34. 53	36. 67
1949	40. 23	33. 78	38. 73
1950	33. 78	26. 28	29. 41
1951	50. 00	32. 00	44. 17
1952	51. 85	36. 47	44. 02
1953	36. 47	30. 47	35. 90

TARIFF

The small quantities and low grade of the antimony ores in the domestic reserves combine to make such high working costs that domestic mining is limited to periods of extraordinary demand and high prices. Accordingly, the tariff on antimony is designed or written to protect the smelting rather than the mining industry. The present tariff rates on the major antimony products follows:

Tariff rates on antimony

[Cents per pound or percent ad valorem]

Ore and concentrates	Free
Metal	2
Oxide	1
Liquated	¼

IMPORTS[2]

During 1953 the contained antimony imported for consumption—the lowest recorded since 1949—totaled 11,500 tons, a 10-percent decrease compared with the 12,800 tons imported in 1952. In terms of recoverable metal, however, imports totaled 10,900 tons, comprising 7,200 tons of ore and concentrates, 2,600 tons of metal, 1,100 tons of antimony oxide, and a small quantity of antimony sulfide.

Imports of ore and concentrates, principally from Bolivia and Mexico, decreased only 2 percent, the grade of ore averaging 45 percent. Imports of metal, chiefly from Mexico, Yugoslavia, and Belgium-Luxembourg, decreased 22 percent. Imports of antimony oxide, 71 percent of which came from the United Kingdom, decreased 27 percent, and imports of antimony sulfide (supplied solely by the United Kingdom), decreased 50 percent.

Table 6 shows imports for consumption and table 7 general imports, by countries.

GOVERNMENT SUPERVISION

WAR AND DEFENSE PROGRAM

Various regulations regarding many commodities, including antimony, were promulgated by the Government during the World War II period, with the objective of obtaining an adequate supply, controlling distribution, and preventing excessive prices.

Under the Strategic Minerals Act of June 7, 1939, the expenditure of $500,000 annually by the United States Department of the Interior for investigating domestic resources of strategic minerals was authorized. The expenditure of $100,000,000 over a 4-year period by the Reconstruction Finance Corp. for acquiring stocks of strategic and critical materials (including antimony) for use in the national defense was also authorized, and on June 28, 1940, the Metals Reserve Company was created to develop the program on a larger scale. Substantial quantities of Chinese antimony were bought, and arrangements were made for deliveries of antimony from the Texas Mining & Smelting Co. at Laredo, Tex., produced from domestic and Mexican ores.

Commencing in 1941, American financial aid through the Metals Reserve Company was given to expand antimony production in foreign countries, notably Bolivia and Mexico, to offset the increasing scarcity of Chinese antimony due to Japanese control of the normal trade routes.

OFFICE OF PRICE ADMINISTRATION

A price of 14.00 cents per pound for domestic antimony (at New York) had prevailed from October 1939 to March 23, 1942. An advance of ½ cent was posted in August 1941 but was withdrawn at the request of the Office of Price Administration. On March 23, 1942, OPA permitted an advance to 15.955 cents. Effective April 13, 1942, the price was advanced to 16.013 cents to cover a 6-percent increase in freight rates.

Up to this time the OPA control of prices had been on an informal basis, but on April 28, 1942, antimony was officially placed under the General Maximum Price Regulations. Effective December 1, 1942, a 3-percent excise tax on freight rates raised the official price to 16.049 cents. On June 3, 1943, a reduction in the freight rates was passed on to the buyer, reducing the price to 15.839 cents, at which point it remained throughout the balance of the war period.

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[2] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines from records of the U. S. Department of Commerce.

TABLE 6.—*Antimony imported for consumption in the United States, 1947–53* [1]

(U. S. Department of Commerce)

Year	Antimony ore			Antimony sulfide		Antimony metal		Type metal and antimonial lead [2] (short tons)	Antimony oxide	
	Gross weight (short tons)	Antimony content		Gross weight (short tons)	Value	Short tons	Value		Gross weight (short tons)	Value
		Short tons	Value							
1947	28, 471	9, 257	$2, 672, 249	17	$7, 914	5, 879	$3, 487, 126	187	--------	--------
1948	41, 610	13, 464	4, 312, 431	533	314, 809	3, 201	2, 022, 676	1, 569	--------	--------
1949	17, 855	7, 473	2, 488, 271	81	42, 537	1, 853	1, 242, 582	654	56	$27, 290
1950	22, 307	9, 746	1, 850, 162	19	8, 895	4, 632	2, 204, 091	1, 936	1, 160	428, 386
1951	26, 698	11, 746	4, 571, 974	6	5, 936	2, 231	1, 780, 576	465	2, 039	1, 525, 016
1952	18, 246	7, 945	3, 200, 889	34	20, 719	3, 354	2, 338, 938	1, 494	1, 766	1, 056, 286
1953	17, 242	7, 778	2, 035, 125	17	8, 678	2, 612	1, 402, 226	1, 350	1, 296	579, 600

[1] Does not include antimony contained in lead-silver ore.
[2] Estimated antimony content.

TABLE 7.—*General imports of antimony into the United States, 1944–48 (average) and 1949–53*

[Short tons antimony content]

(U. S. Department of Commerce)

	1944–48 (average)	1949	1950	1951	1952	1953
Ore and concentrates:						
Australia			20			
Bolivia	4, 450	3, 153	5, 488	4, 626	4, 967	5, 558
Canada	35	49	258			
Chile [1]	520	544	877	492	86	261
France			55	49	2	22
French Morocco	19		79	90		
Greece				7	15	50
Honduras	27	8		5		1
Iran			66			
Japan			23			
Mexico	7, 847	2, 985	3, 127	4, 003	2, 272	1, 662
Mozambique				90		
Peru	712	727	358	251	251	212
Portugal	3	7				
Thailand	18		16			
Turkey	11					
Union of South Africa	12			1, 904	348	
United Kingdom					4	12
Total	13, 654	7, 473	10, 367	11, 517	7, 945	7, 778
Metal:						
Belgium-Luxembourg	53	384	936	320	545	548
Canada		11		23		
China	2, 193	297	92			
Czechoslovakia			44	17	356	
France			277	217	5	253
Germany, West			325		77	
Italy	6	44		19	41	11
Japan	175					6
Mexico	55	768	1, 478	1, 094	1, 055	815
Netherlands		11	138	22	3	6
Peru	26			103	25	
Spain			52	11		
Sweden			55			
Switzerland			11			
Thailand	1					
Turkey					143	
United Kingdom	6	78	285	134	320	361
Yugoslavia	31	472	795	271	819	627
Total	2, 546	2, 065	4, 488	2, 231	3, 389	2, 627
Oxide:						
Belgium-Luxembourg		37	196	159	203	303
Bolivia			17			
Canada			42			
France			32	9		
Germany, West						13
United Kingdom		9	676	1, 524	1, 263	760
Total		46	963	1, 692	1, 466	1, 076
Sulfide:						
Belgium-Luxembourg			4	5	8	
China	77	57	9	1		
United Kingdom						11
Yugoslavia					16	
Total	77	57	13	6	24	11
Grand total	16, 277	9, 641	15, 831	15, 446	12, 824	11, 492

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[1] Imports from Chile probably were mined in Bolivia or Peru and shipped from a port in Chile.

WAR PRODUCTION BOARD

Antimony supply was comparatively scarce throughout 1941, but no mandatory priorities were invoked by the Office of Production Management during the year, even though some metal had been withdrawn from Government stockpiles to relieve shortages. On March 30, 1942, allocation control over antimony was ordered by the Director of Industry Operations of the War Production Board, under General Preference Order M–112, partly effective at once and fully effective May 1. Under this order antimony in all forms (except alloys containing less than 50 percent) were placed under strict allocation, with deliveries restricted to those specifically authorized by the Director of Industry Operations. The use of antimony in alloys was limited to that required by purchasers' specifications. Effective July 1, 1942, the order prohibited the use of antimony in paints, enamels, lacquers, toys, and ornamental or decorative objects and limited its use in battery grids to a maximum of 7.5 percent by weight.

On March 8, 1943, Order M–112 was amended to permit unallocated deliveries of metal and alloys in amounts not exceeding 2,240 pounds in any 1 month. At the same time the limitation of use in storage battery grids and other industrial alloys was omitted, and antimony content in pigments was permitted up to 2 percent. On December 4, 1943, effective January 1, 1944, allocation control was removed; consumers using more than 2,240 pounds per month were still required to make periodical reports but on a quarterly basis instead of monthly.

On February 3, 1944, effective February 10, allocation and monthly reports were reinstated, owing to increasing demand, but no specific limitations on uses were imposed. On May 12, 1945, the regulations were tightened to the extent of reducing the unallocated limit from 2,240 pounds per month to 224 pounds and restricting inventories and deliveries to a 30-day supply of metal or a 45-day supply of ores and concentrates. The order in this form was in effect at the end of the war and was retained for several months of the postwar era, while supply and demand were being brought back to normal conditions.

NATIONAL PRODUCTION AUTHORITY—ORDER M–39

As a result of the defense program and with the continued high-level demand, antimony received a high rating on the strategic list for military purposes. The NPA placed limitations on inventories of antimony and materials containing antimony, other than ores and concentrates. Effective February 15, 1951, Order M–39 set limitations on acceptance of deliveries to the smallest quantity of material reasonably required to meet the current scheduled rate of operations for a 60-day period. Antimony ore and metal are among the materials stockpiled.

OFFICE OF PRICE STABILIZATION

Effective November 21, 1951, the Office of Price Stabilization issued Ceiling Price Regulation 96, covering sales of antimony metal, antimony oxide, and sodium antimonate, including any sold by an importer or exporter. In general, the regulation set basic ceiling prices for carlots, with adjustments to reflect certain transportation charges and with specified differentials for less than carlots and for special packing.

Ceiling prices for carlots of antimony metal were set at 49½, 50, and 50½ cents per pound, f.o.b. shipping point, depending on grade. A ceiling price for delivered carlots of antimony oxide in bags was set at 48 cents a pound; for sodium antimonate the price was set at 42 cents. Effective December 13, 1951, the OPS issued General Overriding Regulation 9, Amendment 10. This exempted from price control all sales of domestic antimony ores and concentrates.

DEFENSE MINERALS EXPLORATION ADMINISTRATION

The search for additional antimony deposits is being carried on through exploration projects contracted for by DMEA under authority of the Defense Production Act. As of January 1, 1953, DMEA has granted exploration assistance to 12 projects that have a total contract value of $462,700, with the Government's participation at $347,000 (75 percent of costs).

TRANSPORTATION COSTS

The principal world ports from which antimony ore and metal are shipped are Marseilles, France, and Vera Cruz, Mexico. Bolivia uses the ports of Arica and Antofagasta, Chile and Mollendo, Peru. Alaska uses the ports of Seward and Nome. The Union of South Africa uses the port of Lourenço Marques, Mozambique. The principal ports of entry are New York, Baltimore, Seattle, and Boston, in the United States; Marseilles and Le Havre, France; and Liverpool and Portsmouth, England.

The topography of Bolivia, Peru, and Alaska has made the mining and marketing of antimony ore difficult and expensive. For example, Alaska antimony ore from the Sawtooth Mining Co. operations 65 miles (airline) northwest of Fairbanks to Seward costs about $50 to $60 per ton for freight alone.

WORLD RESERVES

World reserves of antimony metal in ores are estimated to be of the order of 3½ million metric tons, of which roughly 60 percent are in countries now dominated by U.S.S.R. (China, Czechoslovakia, and U. S. S. R.).

Reserves in Free World countries are of the order of 1,300,000 metric tons or 40 percent of the total. Since 1944 the world production of primary antimony has averaged about 39,000 metric tons annually, of which about 75 percent originated in Free World countries.

The United States consumes annually, about 40 to 50 percent of the Free World's output and contributes virtually nothing to the Free World reserve position for 50 to 60 percent grade ores.

The President's Materials Policy Commission Report (the Paley Report) estimates world reserves of antimony as follows:

TABLE 8.—*World reserves of antimony* [1]

	Metal in ore, thousand metric tons [2]
Free-World:	
Bolivia	500
Mexico	500
Union of South Africa	150
United States	[3] 110
Yugoslavia	50–75
Minimum Free World total	1,300
Communist-World:	
China	[4] 2, 000
Czechoslovakia	40–60
U. S. S. R.	75–150
Total Communist-World	2, 200
Approximate world total	3, 500

As very little authoritative information is available on this subject, the foregoing estimates can represent little more than orders of magnitude.

[2] Measured, indicated, and inferred. Measured fraction insignificant, includes all material down to 2 percent Sb.

[3] Includes 75,000 tons measured and indicated and 35,000 tons inferred; corresponding grade, about .2 percent Sb.

[4] Estimated at 4 million tons by Chinese Geological Survey.

OUTLOOK

Free World production of antimony in future will chiefly, as in the past, continue to come from Mexico, Bolivia, and the Union of South Africa. The immediate outlook is for the world output of antimony to continue at about 30 to 35 thousand metric tons a year. Increases in production, principally in Yugoslavia, Greece, and Turkey, and possibly Peru, should offset, to some extent, declines in other countries. No significant new sources of antimony are in sight in the United States, although exploration work now in progress in Alaska may augment moderately the potential supply.

Mechanization of mining and milling operations in Bolivia could increase antimony production by several thousand tons a year; however, under the existing political conditions, investments to effect such mechanization seem unlikely. In Mexico the large antimony mines are becoming more costly to operate, and the average grade of ore is declining.

Among the most promising expanding sources of antimony is the Union of South Africa. Production is confined to the Murchison range in northeastern Transvaal. In 1953 the Union of South Africa produced about 12 percent of the estimated Free World production of antimony. The possibility of large Chinese production becoming available to the Free World cannot be discounted completely.

NEW USES IN PROSPECT

Two potential new uses for antimony have been mentioned recently in the technical press, which, if successfully developed, would create significant demands on the available supply of the metal. One of these is in electroplating, the other in electronics.

Antimony's silvery appearance and resistance to tarnish and corrosion have inspired numerous attempts to use it as a plating metal, but these have been defeated in the past by its brittleness and the difficulty of developing suitable plating techniques. In recent years, two methods have been devised by which steel and other metals can be plated satisfactorily with antimony. It is reported that these coatings are adherent, ductile, and readily polished, comparing well with nickel and chromium plate. Bradley Mining Co. is sponsoring research at Battelle Memorial Institute to perfect an antimony plating process, and it is believed that work along the same line is in progress in other laboratories. If these investigations should lead to industrial adoption of antimony plating and if antimony were substituted to a significant extent for nickel and chromium, substantial tonnages would be required. A further important consideration, besides the increased demand for antimony, would be the saving of the more highly strategic metals in an emergency.

A second new use for antimony is in the form of intermetallic compounds for semiconductor uses. Extensive investigations by both Government and industry are being conducted on these compounds, the most promising of which are aluminum-antimony and indium-antimony.

Antimony semiconductor compounds show promise for use where previously known semiconductor substances (germanium and silicon) are not suitable due either to limited availability or to limitations in electrical or physical characteristics and, in the case of germanium, due to deterioration in electrical functioning with a moderate rise in temperature.

The research work to date indicates that aluminum-antimony should be superior to both germanium and silicon for high-temperature operation in electronic devices (there is a critical need for semiconductor materials that can operate at high temperatures in military aircraft, guided missiles, and digital computers). Aluminum-antimony also is of importance because of the ready availability of its components.

It is believed that the properties of indium-antimony will lead to several important new uses.

In view of the high rate of research (both in the United States and abroad) on antimony semiconductor compounds, it is expected that their development will proceed rapidly. It is difficult to estimate the potential market for antimony in this field other than to say that, because of the many possible new applications in the rapidly expanding semiconductor industry, a large tonnage future requirement for high purity antimony is indicated.

Another new use under development is the use of antimony compounds in reinforced plastics, namely, the polyester resins with fiber glass. There is a potential large and growing use for antimony oxide as a flame proofing agent for reinforced plastics in the building industry in such applications as panels.

PROBLEMS

Many problems of the antimony industry in the United States arise from the strategic nature of the metal. As this country normally has depended largely on imported antimony ore and as the demand has fluctuated sharply in periods of military emergency, the domestic mining industry has experienced repeated cycles of activity and depression. Technologic developments in production and consumption occasionally have contributed to this instability, as have also Government restrictions and stockpiling programs.

A continuing problem of the mining industry arises from the fact that the domestic reserves are relatively small and low-grade, particularly as compared with those in Mexico, Bolivia, and the Union of South Africa. Thus, the presence or threat of overwhelming foreign competition is of paramount importance to domestic antimony producers.

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ARSENIC

By

Abbott Renick[1]

THE POISONOUS quality of arsenic that has made it a murder vehicle from early historic times now finds a useful application as the active ingredient for many insecticides and fungicides.

Summary

The principal use of white arsenic[2] is in the manufacture of insecticides and weed killers. Other important uses include wood preservatives, manufacture of lead shot, toxic gases, and alloys and in the glass industry.

The United States is in a favorable position to meet its arsenic requirements. It is unlikely that any new uses for arsenic or its compounds will tax the existing arsenic-producing capacity of the Nation.

Arsenic normally occurs as a constituent of various metallic ores, in which it is usually an undesirable impurity. The bulk of the supply, accordingly, is obtained as a byproduct at copper and lead smelters in the recovery of other metals, notably gold, silver, copper, and lead.

Domestic arsenic output is limited to white arsenic, virtually all arsenic metal being imported from Sweden and the United Kingdom. Mexico, which has been the principal foreign source of arsenical ores and concentrates, supplied 93 percent of that imported in 1953.

Domestic and world arsenic resources are extensive. Sweden is the largest producer, followed by the United States, Mexico, Germany, Belgium, Australia, Japan, and other countries. It is believed that the United States could supply much more of its requirements from common arsenic-bearing minerals, provided that the price for arsenic was attractive to the producers. At present the apparent consumption of white arsenic in the United States approximates 15,000 tons annually. Mexico, Canada, and South America could supply the small quantity of white arsenic or metal necessary to complement the United States output to meet all requirements during a war emergency.

There is no serious security problem in the supply of arsenic or its compounds. This commodity has not been classified as critical or strategic, and none is being accumulated for the National Stockpile. Table 1 shows salient statistics for white arsenic for 1914–53.

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[1] Commodity-industry analyst, Bureau of Mines.

[2] The term "arsenic" in commerce is usually applied to white arsenic (refined grade, minimum 99 percent), the trioxide (As_2O_3), instead of the elementary or metallic arsenic, which has a relatively small consumption. Compounds of arsenic are referred to by their trade names.

TABLE 1.—*Salient statistics of white arsenic in the United States, 1914–53*

[Short tons]

Year	Production	Sales	Imports	Exports [1]	Apparent consumption [2]	Producers' stocks	Price per pound [3]
1914	4, 670	(4)	1, 594	------------	6, 264	(4)	$0. 03¾
1915	5, 498	(4)	1, 400	------------	6, 898	(4)	. 03½–. 04½
1916	5, 986	(4)	1, 071	------------	7, 057	(4)	. 03⅞–. 08¼
1917	6, 151	6, 151	1, 178	------------	7, 329	(4)	. 08 –. 20
1918	6, 323	6, 323	1, 847	------------	8, 170	(4)	. 09 –. 15
1919	6, 029	6, 029	4, 389	------------	10, 418	(4)	. 09 –. 12
1920	11, 502	11, 502	3, 740	------------	15, 242	(4)	. 10¼–. 18
1921	6, 158	4, 786	1, 669	------------	6, 455	(4)	. 05¾–. 09½
1922	9, 350	10, 027	1, 081	------------	11, 108	(4)	. 06 –. 15½
1923	14, 902	14, 271	10, 152	(4)	24, 423	(4)	. 09 –. 15½
1924	20, 177	14, 453	8, 877	(4)	23, 330	(4)	. 13½–. 06½
1925	12, 119	12, 317	9, 316	(4)	21, 633	(4)	. 06¼–. 03½
1926	6, 759	11, 805	7, 703	(4)	19, 508	(4)	. 03 –. 03½
1927	11, 730	11, 560	12, 517	(4)	24, 077	(4)	. 03½–. 04
1928	14, 163	11, 767	11, 153	(4)	22, 920	(4)	. 04
1929	16, 605	14, 546	13, 157	(4)	27, 703	(4)	. 04
1930	17, 057	17, 425	10, 471	(4)	27, 896	(4)	. 04
1931	17, 137	13, 777	7, 791	1, 400	20, 168	(4)	. 04
1932	12, 704	12, 483	6, 882	2, 000	17, 365	(4)	. 04
1933	10, 650	11, 797	10, 583	2, 000	20, 380	(4)	. 04
1934	13, 096	15, 623	14, 110	2, 700	27, 033	(4)	. 03½–. 04
1935	14, 237	12, 670	15, 075	800	26, 945	(4)	. 03½
1936	15, 379	15, 581	17, 586	1, 000	32, 167	(4)	. 03½
1937	16, 814	17, 636	19, 256	2, 200	34, 692	(4)	. 03½–. 03
1938	16, 685	13, 160	14, 238	2, 300	25, 098	(4)	. 03
1939	22, 341	22, 439	14, 674	3, 200	33, 913	5, 505	. 03
1940	24, 983	23, 339	9, 929	1, 639	31, 629	6, 944	. 03 –. 03½
1941	32, 481	34, 784	10, 292	1, 616	[5] 43, 460	4, 518	. 03 –. 04
1942	28, 681	31, 038	16, 350	305	[5] 47, 083	[6] 2, 187	. 04
1943	31, 202	32, 423	16, 112	1, 975	[5] 46, 560	[6] 1, 138	. 04
1944	36, 094	34, 472	9, 965	2, 401	[5] 42, 036	[6] 2, 760	. 04
1945	24, 349	24, 810	13, 149	858	37, 101	[6] 2, 299	. 04
1946	10, 211	12, 039	13, 821	[7] 1, 000	24, 860	471	. 04 –. 06
1947	18, 755	18, 188	13, 940	[7] 1, 000	31, 128	1, 038	. 06
1948	18, 639	14, 965	9, 336	------------	24, 301	4, 712	. 06 –. 06¼
1949	12, 795	10, 181	4, 696	------------	14, 877	7, 326	. 06 –. 05½
1950	13, 273	17, 330	14, 774	------------	32, 104	2, 479	. 05 –. 06½
1951	16, 190	14, 351	14, 518	------------	28, 869	4, 834	. 06½
1952	15, 673	9, 244	4, 483	------------	13, 727	11, 263	. 06½–. 05½
1953	10, 873	11, 315	4, 717	------------	16, 032	10, 820	. 05½

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[1] Figures for 1943-45 reported by U. S. Department of Commerce; figures for all other years reported by producers to Bureau of Mines.
[2] Producers' shipments plus imports minus exports.
[3] Refined white arsenic, carlots, as quoted by Oil, Paint and Drug Reporter.
[4] Data not available
[5] Consumption based on allocation data of the War Production Board was 40,442 tons in 1941, 41,520 tons in 1942, 51,083 tons in 1943, and 43,500 tons in 1944.
[6] Excludes Government stocks as follows: 1942, 2,693 short tons; 1943, 1,018; 1944, 3,029; 1945, 1,987.
[7] Estimated.

BACKGROUND

HISTORY

The writings of the ancient Greeks refer to sulfides that have since been identified as arsenic minerals. What is now known as realgar (As_2S_2) was described as "sandarach," and the yellow arsenic sulfide, orpiment (As_2S_3) was apparently "arsenicum" and "auripigmentum." Derivation of the term "arsenic" has been variously attributed to the Greek word meaning valiant or bold, alluding to the great energy with which it acts on other metals; to the Arabian arsa naki; and to the Greek term derived from the Persian word for arsenic, zarnick, zirnuk, or zirne. Strangely, these early writers mentioned several medicinal properties of arsenical preparations but failed to note the toxic properties of white arsenic. Apparently the first mention of metallic arsenic is in a fragmental writing of the third or fourth century attributed to Zosimos. The preparation of white arsenic by roasting the sulfide is described by Olympiodorus, a writer of the fifth century, but he refers to the product as "white alum." In the 17th century, N. Lemery obtained metallic arsenic by heating white arsenic with potash and soap; it was also made by J. Schroder and J. F. Henkel. In 1733 G. Brandt showed that white arsenic was the calx or oxide of the metal arsenic.

Many recorded assassinations by poison were doubtless accomplished by means of arsenic. As late as the latter half of the 17th century professional poisoners were permitted to practice their nefarious arts almost openly and were a fearful factor in the political and domestic life of the times. Arsenical compounds, sold ostensibly for use as cosmetics or for some other innocent purpose, were used by such infamous adepts as Lucretia Borgia in Italy and the Marchioness de Brinvillier and M. de St. Croix in France. The poisoned shirt that caused the death of the Duke of Savoy is said to have been soaked in a concentrated solution of an arsenical preparation.

During World War II the first commitment for the purchase of arsenic and its compounds by the United States was made in 1942. As of October 31, 1944, the Metals Reserve Company reported that arsenic ranked 29th relative to the cost of deliveries to the Government.

OCCURRENCES

Arsenic is not an abundant element but is widely diffused in nature. Traces of arsenic are common in natural waters and in organic matter. The native metal occurs but is not common. Arsenopyrite or mispickel, a sulfarsenide of iron ($FeAsS$), is a familiar mineral; it is found chiefly along with other metallic sulfides and arsenides. It frequently contains cobalt as well as gold; and when the cobalt comprises 4 to 10 percent, the mineral may be called danaite. Leucopyrite or lollingite—the arsenide of iron ($FeAs$)—is fairly abundant in certain localities. Realgar, the red sulfide, As_2S_2, has been mined in small quantities in Washington, and both realgar and orpiment have been produced as commercial minerals in Europe and Asia to some extent. Arsenic occurs also in combination with the heavy metals as arsenides and sulfarsenides.

The world supply of arsenic is now recovered mainly as a byproduct from the flue dusts of copper and lead smelters. Often the arsenic content of the ores treated is very small; at many plants it is considered only a nuisance and rarely is paid for.

ESTIMATES OF ORE RESERVES

Two properties of Gold Hill (the United States mine of the United States Smelting, Refining & Mining Co. and the Western Utah Copper Co. mine) have been important producers in the past, and substantial ore reserves are still available at these mines. Likewise, ores that could yield arsenic directly in connection with milling or treatment for other valuable metals, chiefly gold and silver, as at the Jardine mine in Montana and the Getchell mine in Nevada, could be operated in the future. The ore reserves of white arsenic in the United States have been estimated as follows (*15*).[3]

	Recoverable white arsenic, tons
Class I. Byproduct of smelting operations	1, 000, 000
Class II. Milling ores (gold and silver)	1, 000, 000
Class III. High-grade arsenic ore (20 percent As_2O_3)	500, 000
Total	2, 500, 000

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[3] Italicized numbers in parentheses refer to items in the bibliography at the end of this chapter.

DESCRIPTION AND PROPERTIES (*9*)

Arsenic metal in its ordinary form is a silver-gray or tin-white substance with a metallic luster, but it turns black when exposed to the air. It is one of the lighter metals, its specific gravity (5.6 to 5.9) being intermediate between that of aluminum (2.7) and iron (7.9). Though moderately hard (hardness—3.5 on Mohs' scale; 147.0, Brinell), it is far too brittle to be used alone for purposes demanding even a moderate degree of mechanical strength. Its poisonous character and the fact that it is appreciably volatile at the temperature of the boiling point of water render it further unfit for most of the purposes for which the common metals are employed.

Three other allotropic forms of arsenic, unlike the more stable steel-gray metal, are not conductors of electricity. One is a yellow powder like flowers of sulfur, another brown, and another gray to black. The garliclike odor of burning arsenic has been attributed to the presence of yellow arsenic. While interesting from a scientific standpoint these forms of arsenic have no present practical uses.

In alloys arsenic generally tends to cause brittleness and to lower the melting point. It whitens copper and often tends to make a brighter color. With most metals arsenic combines chemically (in atomic proportions), forming arsenides.

In the periodic chart of the atoms arsenic is element 33, following the rather rare element germanium and preceding selenium. It is in the same vertical column with phosphorus, antimony, and bismuth. Arsenic may act as an electropositive element, but in most of its chemical reactions it has the characteristics of an electronegative element; it is on the borderline between distinctly metallic or base-forming elements and nonmetallic or acid-forming elements. Like phosphorus and antimony, it forms gaseous hydrides, usually arsine (AsH_3). The valence is 3 and 5. Prime white arsenic (As_2O_3) is a fine, white powder.

Commercial specifications require that the refined product shall contain 99 percent or more of arsenic trioxide; usually it contains 99.5 percent. It should be free from lumps, and for some uses a certain standard of solubility in nitric acid may be specified. Crude arsenic is grayish white, and sales contracts generally specify 95 to 98 percent. For sale to large consumers who buy in car lots, white arsenic is not only shipped in bulk but is also packed in barrels containing 550 pounds each. For jobbing purposes it is packed in casks of various sizes from 100 pounds up, and for the retail trade it is put up in packages of 1 to 10 pounds. Other important commercial compounds are arsenates or salts of arsenic acid. Both the arsenites and arsenates, except those of the alkali and alkaline-earth metals, are soluble with difficulty if at all in water but readily dissolved by acids.

LOCATION AND PLANT CAPACITY

The arsenic production of the United States is now obtained from Anaconda Copper Mining Co. at Anaconda, Mont. (copper smelter); United States Smelting, Refining & Mining Co. at Midvale, Utah (lead smelter); and American Smelting & Refining Co. at Tacoma, Wash. (copper smelter).

The Tacoma smelter has an annual capacity of about 18,000 tons of crude As_2O_3 a year; the arsenic dust is collected in a Cottrell plant. Arsenic-bearing copper-gold concentrates are obtained mostly from abroad.

The Anaconda arsenic plant has an annual capacity of approximately 10,000 tons of As_2O_3.

The United States Smelting, Refining & Mining Co., at Midvale, Utah, has an annual capacity of 2,000 tons of high-grade crude arsenic.

The plant capacity of these three plants totals 30,000 tons annually.

In addition to the foregoing plants, the El Paso smelter, operated by the American Smelting & Refining Co., is a potential producer of white arsenic. This smelter has an annual capacity of 4,800 tons of As_2O_3 which is collected in a baghouse. El Paso has not produced arsenic since 1949.

Crude arsenic dust has also been produced at the Getchell mine, Humboldt County, Nev., as a result of roasting arsenical gold ores preparatory to cyanidation.

The Murray smelter, at Murray, Utah, formerly operated by the American Smelting & Refining Co., has been dismantled. Former operations were at a rate of 7 to 10 million pounds of white arsenic a year. During recent years the smelter acted as a scavenger plant—roasting speiss and baghouse and Cottrell-fume dust from various company plants (particularly Garfield and Selby) and from East Helena and Midvale.

Arsenic is recovered from Boliden in such quantities as to make Sweden the world's largest producer. White arsenic, now stored in concrete silos in that country, is believed to exceed 400,000 tons. The Bolidens Gruv A-B has established several arsenic-salt timber-impregnation plants in Sweden; however, the problem in Sweden has been disposal of huge quantities of crude white arsenic. The United States is easily the largest producer of arsenic in the Western Hemisphere and by far the largest consumer, as it absorbs, in addition to its own output, the bulk of arsenic produced in Mexico; further substantial quantities are available from Canada, South America, and Europe. If market conditions warranted, imports from these sources could be increased sharply. The Canadian production in 1951 totaled about 1,200 tons of white arsenic compared with 400 tons in 1950.

TECHNOLOGY

The arsenic output of the Anaconda smelter is produced largely from copper ores mined by the Anaconda Copper Mining Co. at Butte, Mont., supplemented with minor quantities of custom ores. Enargite ($3Cu_2S.As_2S_5$) is one of the most abundant copper minerals of the Butte deposits, second only to chalcocite.

At Anaconda, Mont., for example, enriched arsenic dust from the main Cottrell treaters adjacent to the stack is returned for roasting in six hearth furnaces of the McDougall type. Four of these furnaces are connected in series for the gas flow. The first or roasting furnace is fired with natural gas on the third and fifth hearths and is equipped with a special type double apron feeder completely enclosed and dust tight. One side is used for feeding dust; and the other may be used for feeding reducing agents, such as fine coal or copper concentrate.

The other three furnaces in each set serve merely for condensing the arsenic. All of the inside hearths have been removed, and a central vertical partition has been installed in each shell. The gas enters each condenser at the top, passes down one side of the partition which has an opening at the bottom, and up the other side and out the top. After passing through three of these condensers, the gas goes to the main dust chamber and flue system. Crude arsenic containing about 90 percent As_2O_3 accumulates in the bottom of the condensers and is removed daily. The residue from the roasting furnace containing about 5 percent As_2O_3 is stockpiled and sold for its bismuth and lead contents.

Dust from the main flue may be treated similarly or sent directly to the reverberatory furnaces. If it is treated in the arsenic-roasting plant, the residue is sent to the reverberatory furnaces, and some arsenic may be recovered in the condensers.

The refining plant consists of 2 reverberatory furnaces with hearths 34 feet long and 12 feet wide. The hearth is composed of siliceous material, and furnaces are fired with natural gas. One furnace, if required, can treat about 21 tons of 90-percent As_2O_3 crude a day and produce 15 tons of refined white arsenic.

The charge is conveyed to the furnace by a screw conveyor and delivered to five double hoppers, each single hopper being just a little off center. The charge is dropped and rabbled about 3 times on each 8-hour shift. If a furnace is operating 24 hours a day, the residue is cleaned out and sent to the smelter daily. A temperature of 600° C. is maintained in the furnace compared with 700° C. in the McDougall roaster. The recovery is about 80 percent of the arsenic in the charge.

The sublimed white arsenic is caught in condensing chambers. Each chamber is 225 feet long and divided into 39 compartments or kitchens. The kitchen nearest the furnace is larger than the others and is kept at a temperature of 295° C. to allow only nonarsenical dust to settle out. The bulk of the arsenic settles in those kitchens where the temperature ranges from 180° to 120° C. The gases leave the kitchen at a temperature of 90° to 100° C. The kitchens nearest the furnace contain a fused amorphous arsenic carrying about 95 percent As_2O_3; this is black and has to be refined again. In succeeding kitchens, the arsenic content increases to 99.9 percent As_2O_3, and the product is white and crystalline, changing in the last kitchen to a fine, white powder.

Metallic arsenic (about 125 tons a year) is imported into the United States for use in hardening lead and promoting the formation of certain copper alloys.

Basically, the process for producing metallic arsenic is as follows: The charge, with proportions consisting of about 1 pound of wood charcoal to 4 pounds of arsenic trioxide, is placed in a rotary mixer. The mixed batch for charging to a small furnace weighs about 200 pounds and to a large furnace about 450 pounds. The furnace is equipped with pyrometers, and gas is used as fuel. The temperature varies between 1,200° and 1,300° F.

USES

Arsenic metal composes a relatively small part of world consumption of the element. In the United States, rarely more than 100 tons, and often less, of metallic arsenic is required annually; this is used for hardening lead shot and for promoting the formation of certain copper alloys. In most of these alloys the arsenic content is well under 1 percent, although there are bronzes, chiefly speculum metals, that contain as much as 2 percent, and occasionally a trifle more. In lead-base alloys for battery grids, bearings, electric cable sheathing and anodes (zinc electrolysis), the proportion is usually well under 0.5 percent, and shot lead carries only 0.2 percent. Britannia metal often contains arsenic, and a number of brasses for high-temperature work and copper-boiler tubes and plates for locomotive fireboxes (used in Europe) may contain arsenic. A little of the metal may be used in medicine as an amalgam.

Arsenic is usually considered a deleterious element in alloys. In 60 : 40 brass, for instance, as little as 0.12 percent reduces ductility as much as 50 percent; and in steel amounts above 0.20 percent cause an effect similar to that of phosphorus, namely, coldshortness.

There are two leading commercial sulfides of arsenic: (1) Arsenic disulfide (As_2S_2), an orange-red compound found in nature as the mineral realgar, but ordinarily made artificially, and containing 70 percent arsenic; and (2) arsenic trisulfide, (As_2S_3), a lemon-yellow material containing 61 percent arsenic and found in nature as orpiment or manufactured artificially. Realgar is used as a paint pigment, as a depilatory in tanning, and for making a white (Bengal) fire used in pyrotechnics. The red color is a desirable feature in treating hides or skins; but, as sodium sulfide and calcium compounds are cheaper and are nonpoisonous, they tend to displace the arsenic sulfide for this purpose. Orpiment or king's yellow is used as a paint pigment and a reducing agent in chemical work; it is very bright yellow and has good covering power but is not permanent when exposed to light and cannot be mixed with many other colors.

The pentasulfide, As_2S_5, has a more limited use, in paint pigments, blue fire, and Bengal lights.

The "arsenic" produced in the United States is arsenious oxide (white arsenic). During 1953 about 40 percent of the total production of this product was used in the manufacturing of insecticides, which require arsenic of the highest grade; about 50 percent was used in weed killers and wood preservatives, which are made from crude arsenic (including "black" dust, "flue" dust, and "treater" dust); and about 10 percent went into the glass industry, which, like the insecticidal uses, requires refined white arsenic. Other minor uses require smaller quantities of the crude and refined product.

Insect infestations and fungus infections depend, to a large extent, on weather conditions and other natural factors, few of which are predictable to any satisfactory degree. If serious epidemics should become widespread, large tonnages of pesticidal formulations would be needed promptly to avoid heavy losses by growers. A striking example is in the Cotton Belt where, with heavy rainfall in June and July, boll-weevil populations might increase quickly to enormous numbers. Growers may find their sources of supply exhausted if bad outbreaks should occur. This situation would be serious where formulators have bought only enough stocks to meet current needs.

Calcium arsenate and lead arsenate are the arsenicals ordinarily used as insecticides. The consumption of calcium arsenate exceeds that of lead arsenate owing to the greater effectiveness of the former product in combating the ravages of the boll weevil in the cotton-producing States of the South. It is estimated that boll-weevil control affords a potential outlet for virtually all the arsenic now produced in the United States. Lead arsenate finds its chief use in poison sprays and dusts for destroying the codling moth, plum curculio, cabbage worm, potato bug, tobacco hornworm, and other pests that injure fruits and vegetables. Paris green (copper arsenite), magnesium arsenate, and manganese arsenate are used as insecticides.

Arsenic in various forms, either alone or mixed with other substances, has been used as a wood preservative for many years, and many poles, ties, and other materials treated with it are now in service. The consumption of wood preservatives since 1949 has averaged about 1,400,000 pounds a year.

Arsenic, usually regarded as an impurity, has been found to add desirable properties to some alloys. The solder used for "filling" in automobile bodies in the years immediately preceding World War II contained 15 to 50 percent tin, a maximum of 1 percent antimony, and the balance lead. The generally accepted and specified composition in the years from 1939 to 1941 was as follows:

Element:	*Amount, percent*
Tin	20.0
Antimony	1.0 maximum
Lead	Balance

The industry during this period believed that reduction of tin below 20 percent in a lead-tin solder for filling purposes would result in an inferior and unsatisfactory production-line operation. This assumption has proved to be unfounded.

Government restrictions on the use of tin immediately after the war made it mandatory for the solder producers and automobile-body manufacturers to develop a filler solder containing low tin. Many compositions were alloyed and tested in this program. The majority of these were lead-base alloys with antimony and tin as the principal alloying elements and small additions of various combinations of arsenic, bismuth, silver, copper, cadmium, and indium.

The results of these tests was that the industry generally agreed that a low-tin solder was feasible for the filling operation. The nominal composition of the solder now being used follows:

Element:	*Amount, percent*
Antimony	4.00–5.25
Tin	2.50–5.00
Arsenic	0.60 maximum
Lead	Balance

The arsenic used in this solder has a marked effect on grain refinement, resulting in a much finer textured material while in the plastic working range.

Recent improvements in battery-grid alloys are represented chiefly by the reduction in antimony content and an increase in arsenic content. An average widely used alloy would be represented by 7 to 9 percent antimony, 0.25 percent tin, 0.05 percent arsenic, 0.03 percent copper, 0.02 percent bismuth, 0.0005 percent iron, 0.001 percent nickel, and the balance lead.

Arsenic is still one of the most practical and persistent materials for permanent soil sterilization; for this purpose, white arsenic has been found to have distinct advantages over soluble forms (*2*).

Chemical debarking is the process of applying a toxic chemical to the living tree during the sap-peeling season, which causes the tree to die and the bark to loosen after a few months. (A solution of 30 to 40 percent sodium arsenite is the most effective chemical for debarking.) The bark remains loose from then on and may be peeled at any convenient time, except during the winter months, when it is frozen to the wood. This allows pulpwood operators to spread their work over a longer period of time, producing peeled pulpwood 8 or 9 months of the year (*16*).

Sodium arsenite has been used with satisfactory results in controlling aquatic weeds in ponds and lakes. This has permitted freer use of the water. Organic matter that has been bound up in the vegetation is released to produce food for fish. Forage fish that have been sheltered by dense weed stands are exposed to predatory game fish, which consequently make better growth.

Clinical studies at the Harvard Medical School, with radioactive arsenic were reported (*13*) to have saved a young girl from permanent brain injury or death from a brain tumor. A small quantity of radioactive arsenic was injected into the girl's veins, and 2 hours later her head was placed between 2 scintillation counters that recorded the positrons emitted by the arsenic carried to the brain. The scintillation counters were mounted on a carriage that swept backward and forward automatically beside the head, dropping about a third of an inch at each sweep. In this manner a complete map of the brain tumor was made. The tumor stood out on the radiation map like mountains on a flat landscape, permitting its extent to be known.

SUBSTITUTES

In recent years non-arsenic-containing chemicals and compounds have competed with arsenic compounds with increasing success as insecticides, rodent controls, medicinals and weed killers. DDT is rapidly replacing lead arsenate. A dust containing DDT and benzene hexachloride is being increasingly used in place of calcium arsenate to control the boll weevil. Sodium arsenate is being replaced as a weed killer by 2,4–D, and modifications of this organic compound (Crag Herbicide 1). Other replacements for arsenic compounds include thallium sulfate as a rodenticide (over 10 times as poisonous as arsenic compounds used for this purpose) and penicillin in place of arsenicals used in treating venereal diseases. To sum up, the use of organic chemicals and improved techniques in employing the more familiar herbicides have made heavy inroads into the requirements for arsenical compounds.

DOMESTIC PRODUCTION

United States shipments of refined and crude arsenic by producers in 1953 totaled 11,315 short tons valued at $539,000 (cost value), an average of 2.38 cents a pound. Of the domestic white arsenic produced in 1953, 95 percent was crude (79–89 percent As_2O_3) and 5 percent was refined (99+ percent As_2O_3).

During 1953 the apparent consumption of white arsenic was 16,032 short tons. The major portion is employed in manufacturing lead and calcium arsenate insecticides. A strong trend developed in consumer preference for organic insecticides over arsenicals in 1944, (namely, benzene hexachloride, toxaphene, and chlordane); this was a factor chiefly responsible for the accelerated trend away from arsenicals. Table 2 shows the production and shipments of white arsenic in 1944–53.

In 1944, the year of peak production, 91 million pounds of lead arsenate was manufactured in the United States. Production and the domestic disappearance of lead arsenate for the crop years 1950–53 are shown in table 3. Lead arsenate was in critically short supply for a few months late in 1951 and early 1952 because of a shortage of lead. Normally, when the supply of metallic lead improves, adequate quantities of lead arsenate are available.

Production of calcium arsenate, like that of lead arsenate, has decreased from the peak of 84,136,000 pounds in 1942. Table 4 shows production and domestic disappearance of calcium arsenate during the four latest crop years. The chief use of calcium arsenate at present is to control boll weevils on cotton.

NOTE.—According to the data compiled in the Bureau of Mines, 1 pound of white arsenic will make 2.69 pounds of calcium arsenate, 3.47 pounds of lead arsenate, and 1.36 pounds of paris green. Thus the factors would be 0.372 for calcium arsenate, 0.288 for lead arsenate, and 0.736 for paris green.

Sodium arsenite is a soluble arsenical compound widely sold for many years in weed control where all plant growth must be removed. Over 10 million pounds, on the basis of white arsenic content, was used annually in herbicides

TABLE 2.—*Production and shipments of white arsenic by United States producers, 1944–53*

Year	Crude			Refined			Total		
	Production (short tons) [1]	Shipments		Production (short tons)	Shipments		Production (short tons)	Shipments	
		Short tons	Value		Short tons	Value		Short tons	Value
1944	31, 182	29, 159	$1, 370, 602	4, 912	5, 313	$326, 217	36, 094	34, 472	$1, 696, 819
1945	21, 358	22, 180	1, 041, 614	2, 991	2, 630	155, 447	24, 349	24, 810	1, 197, 061
1946	8, 981	10, 448	557, 986	1, 230	1, 591	97, 091	10, 211	12, 039	655, 077
1947	17, 636	17, 119	1, 424, 316	1, 119	1, 069	109, 440	18, 755	18, 188	1, 533, 756
1948	17, 213	13, 749	1, 141, 213	1, 426	1, 216	119, 054	18, 639	14, 965	1, 260, 267
1949	12, 289	9, 597	713, 984	506	584	50, 527	12, 795	10, 181	764, 511
1950	11, 903	15, 778	955, 739	1, 370	1, 552	113, 240	13, 273	17, 330	1, 068, 979
1951	15, 485	13, 656	972, 832	705	695	69, 242	16, 190	14, 351	1, 042, 074
1952	15, 046	8, 719	563, 719	627	525	46, 751	15, 673	9, 244	610, 470
1953	10, 345	10, 816	495, 673	528	499	43, 383	10, 873	11, 315	539, 056

[1] Excludes crude consumed in making refined.

TABLE 3.—*Production and domestic disappearance of lead arsenate, crop years, 1950–53* [1]

[Thousand pounds]

Crop year:	*Production*	*Domestic disappearance*
1950	39, 434	27, 490
1951	25, 416	30, 174
1952	14, 286	17, 452
1953	14, 196	16, 000

[1] Production and Marketing Administration, U. S. Department of Agriculture.

TABLE 4.—*Production and domestic disappearance of calcium arsenate, crop years 1950–53* [1]

[Thousand pounds]

Crop year:	*Production*	*Domestic disappearance*
1950	45, 348	38, 842
1951	40, 900	39, 583
1952	7, 634	4, 735
1953	7, 260	7, 000

[1] Production and Marketing Administration, U. S. Department of Agriculture.

for several years before 1950. Sodium arsenite is also used to some extent for dipping cattle, for termite control, and so forth. The overall consumption in the United States at present is about 4½ million pounds a year. Railroads throughout the country have found this compound a cheap and effective means of keeping their roadbeds and rights-of-way free from grass and weeds. Table 5 shows the production of some major pesticides for 1949–53.

FOREIGN TRADE [4]

IMPORTS

White arsenic imports totaled 4,717 short tons in 1953 compared with 4,483 tons in 1952, 14,518 tons in 1951, and an average of 13,042 tons in 1944–48. Of the tonnage imported in 1953, 93 percent came from Mexico, long the

TABLE 5.—*Production of some major pesticides, 1949–53* [1]

[Thousand pounds]

Pesticide	1949	1950	1951	1952	1953 [2]
Benzene hexachloride (gross)	27, 937	76, 698	116, 988	[3] 86, 290	[3] 61, 000
Benzene hexachloride (gamma equivalent)	3, 787	10, 200	17, 100	[3] 14, 000	[3] 10, 650
Calcium arsenate	16, 006	45, 348	40, 900	7, 634	7, 260
Copper sulfate	158, 000	174, 600	213, 872	189, 016	145, 434
2, 4–D (acid base)	(4)	14, 156	17, 671	30, 718	25, 215
2, 4–D (esters)	8, 562	9, 416	11, 592	21, 108	19, 852
2, 4–D (salts) [5]	4, 289	2, 212	5. 532	3, 933	
DDT	37, 904	78, 150	106, 139	99, 929	83, 981
Lead arsenate	16, 866	39, 434	25, 416	14, 286	14, 196
2, 4, 5–T (acid base)	(4)	(4)	2, 456	3, 398	5, 112

[1] Compiled by Production and Marketing Administration, U. S. Dept. of Agriculture
[2] Preliminary figures.
[3] Estimated from incomplete information; includes lindane.
[4] Data not available.
[5] Sodium and amine salts.

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[4] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

principal foreign source; and 6 percent from Canada. Imports of metallic arsenic totaled 71 short tons for use in hardening lead.

EXPORTS

Ordinarily, the exports of white arsenic from the United States are insignificant. Small quantities of arsenical insecticides, however, are exported. The exports of calcium arsenate in 1953 totaled 1,945 short tons. Lead arsenate exported amounted to 152 tons. Table 6 shows arsenicals imported into and exported from the United States by classes, 1944–53.

TARIFF

As under former tariff acts, white arsenic (par. 1614), arsenic sulfide (par. 1613), paris green and london purple[5] (*8*) (par. 1737), and also sheep dip, certain varieties of which contain arsenic (par. 1759), are all free of duty under the present act. Arsenic acid (par. 1) is dutiable at 3 cents a pound, lead arsenate (par. 46) is dutiable at 1½ cents a pound, and metallic arsenic is dutiable at 3 cents a pound. Compounds of arsenic not specified in the tariff act are dutiable at 12½ percent (par. 5) of their foreign market value.

WORLD PRODUCTION

Table 7 shows world production of white arsenic in 1944–53.

TABLE 6.—*Arsenicals imported into and exported from the United States by classes, 1944–53*[1]

[Short tons]

Class	1944	1945	1946	1947	1948	1949	1950	1951	1952	1953
Imports for consumption:										
White (arsenic (As_2O_3 content)	9, 965	13, 149	13, 821	13, 940	9, 336	4, 696	14, 774	14, 518	4, 483	4, 717
Metallic arsenic	11	26	46	9	18	23	69	110	30	71
Sulfide	------	1, 113	44	22	44	22	74	74	------	10
Sheep dip	80	99	1	42	19	28	39	31	51	26
Lead arsenate	------	------	------	60	------	------	------	7	81	------
Arsenic acid	------	------	------	------	------	------	1	3	------	------
Calcium arsenate	------	------	------	------	------	------	114	777	96	------
Sodium arsenate	------	------	------	------	------	------	55	90	33	40
Paris green	------	------	------	------	------	------	44	------	21	------
Exports:										
White arsenic	2, 401	858	1, 000	1, 000	------	------	------	------	------	------
Calcium arsenate	1, 206	1, 750	3, 439	2, 484	2, 285	2, 024	1, 929	2, 678	2, 803	1, 945
Lead arsenate	2, 133	3, 170	1, 398	1, 552	1, 019	430	520	313	128	152

[1] From records of the U. S. Department of Commerce.

OUTLOOK

The reserves of arsenic in the Western Hemisphere are adequate for all foreseeable requirements. Reserves of recoverable white arsenic in the United States have been estimated at 2½ million tons. Large reserves of arsenical ores are available in Mexico and Canada.

Production of white arsenic can be increased during emergencies by providing for greater recovery and treatment of metallurgical products normally wasted, by stepping up the production of operating mines, and by reactivating idle properties. The growing volume of competition from the new pesticide chemicals will continue to make substantial inroads into the market for white arsenic. However, because of insect immunity and the necessity of rotating insecticides, arsenic may have a place in the insecticide cycle.

NEW USES

Several relatively new uses of arsenic may have an influence in increasing consumption, including chemical debarking of trees for pulpwood, the control of aquatic plant life in ponds and lakes, and specialized medical applications.

PROBLEMS

Arsenic is a common impurity in many metals and appears as a byproduct in their recovery; its production is unresponsive to demand or price. Thus an important problem of the industry is the need for developing new and enlarged uses for arsenic.

A major industry problem is the growing competition of non-arsenic-containing chemicals and compounds. In recent years many new synthetic organic compounds have been prepared and tested; these are now used extensively as agricultural chemicals.

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[5] London purple is a calcium-arsenic compound used more on field crops than in gardens.

TABLE 7.—*World production of white arsenic, by countries, 1944–53*[1]

[Short tons]

(Compiled by Pauline Roberts)

Country	1944	1945	1946	1947	1948	1949	1950	1951	1952	1953
North America:										
Canada	1, 314	1, 023	373	394	581	263	397	1, 177	854	712
Mexico	16, 872	16, 549	10, 635	10, 676	8, 346	3, 942	9, 884	14, 072	3, 159	3, 253
United States	36, 094	24, 349	10, 211	18, 755	18, 639	12, 795	13, 273	16, 190	15, 673	10, 873
South America:										
Argentina	[2] 476	[2] 46	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)
Brazil	926	1, 060	914	1, 103	1, 085	1, 057	1, 176	1, 456	1, 062	(3)
Peru	7, 606	3, 527	830	670	1, 114	--------	--------	--------	17	(3)
Europe:										
Austria	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)	(3)
Belgium (exports)	--------	--------	(3)	(3)	166	581	2, 104	358	1, 106	10, 812
France	1, 878	1, 687	3, 461	2, 767	2, 487	2, 165	2, 712	6, 261	6, 934	(3)
Germany (exports)	638	(3)	(3)	(3)	41	1, 192	1, 239	3, 862	122	675
Greece	--------	--------	9	15	20	22	36	62	97	(3)
Italy	293	110	481	1, 786	1, 907	1, 587	800	1, 950	2, 363	1, 283
Spain	371	433	485	534	632	137	175	413	301	(3)
Sweden	3, 355	6, 745	11, 143	17, 734	18, 716	9, 884	15, 997	20, 427	(3)	(3)
United Kingdom[4]	155	129	162	100	(3)	(3)	(3)	(3)	(3)	(3)
Portugal	383	268	560	1, 108	1, 781	1, 075	281	618	1, 452	(3)
Asia:										
Iran[5]	(3)	(3)	(3)	(3)	(3)	88	28	(3)	(3)	(3)
Japan	1, 560	(3)	1, 204	1, 551	1, 946	2, 206	1, 463	1, 515	1, 653	(3)
Africa:										
Southern Rhodesia	945	688	238	459	312	163	126	84	568	417
Union of South Africa	--------	110	13	3	14	--------	--------	--------	--------	--------
Oceania:										
Australia	2, 581	2, 228	1, 820	1, 334	573	283	180	134	134	--------
New Zealand	18	19	20	9	9	21	11	--------	--------	(3)
Total (estimate)	74, 957	61, 288	46, 297	61, 729	59, 525	38, 581	51, 809	69, 446	52, 911	56, 218

[1] Arsenic is also believed to be produced in China, Czechoslovakia, Hungary, and U. S. S. R.; but data are not available, and there is too little information to estimate.

[2] Arsenic content of ore mined.

[3] Data not available; estimate in total.

[4] White arsenic, including arsenic soot.

[5] Year ended March 20 of year following that stated.

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ASBESTOS

By

Oliver Bowles[1]

A ROCK that may be spun into yarn and fashioned into unburnable clothing is unique among minerals. Chrysotile asbestos has that distinction, but the manufacture of heat-resistant textiles is only one of a multitude of uses that range in diversity from pot holders to automobile brakeband linings and from stove wicks to roofing shingles.

Summary

Asbestos is a mineral that has many important uses, including some of primary military importance. The United States is the largest user of asbestos in the world, but produces a mere fraction of its requirements. There are three major varieties of asbestos. Chrysotile, the principal variety of commerce, is a fibrous form of the mineral serpentine. The others—amosite and crocidolite (blue asbestos)—are classed with the amphibole minerals. The major part of United States supplies of chrysotile originates in Canada. Amosite is produced exclusively in Africa and crocidolite chiefly in Africa, with minor output in Australia and Bolivia.

Low-iron chrysotile has important military applications and during World War II was in heavy demand. Currently substitute materials appear to be gaining wider acceptance, and the supply situation has improved. Until recently the chief source of low-iron chrysotile was Southern Rhodesia, but a new enterprise in British Columbia, Canada, is producing increasing quantities of chrysotile that are equal to or better in quality than the fiber obtained in Southern Rhodesia.

Amosite is a variety of asbestos that has important military uses and is obtainable only in South Africa. The supply has been inadequate at times, but output has increased steadily since 1945, and plans are underway for further expansion that will virtually double the output.

At times there has been a threatened or actual shortage of Canadian and similar fibers of spinning grade. With increasing Canadian mill capacity and a decline in demand, there was evidence in 1954 of supply of spinning fiber that exceeded demand.

Bolivian blue asbestos has been regarded as of strategic importance; but requirements are small, imports have increased considerably, and substitute materials are finding wider uses. Accordingly, the situation is not critical at this time.

Until recently the major problems in asbestos were procurement of adequate supplies of both ordinary and low-iron chrysotile, purification of ordinary asbestos to render it acceptable as a low-iron type, substitution of other materials (such as fiber glass) for both low-iron and ordinary asbestos, and development of substitutes for amosite. The situation changed so sharply in 1954 that the problems must be rearranged in order of importance.

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[1] Commodity-industry analyst, Bureau of Mines.

The low-iron chrysotile problem is less pressing than formerly because (1) a large, new source of satisfactory low-iron fiber has been discovered and developed on the American Continent and (2) use of substitute materials has reduced the demand for low-iron fiber.

Substitution of other materials for spinning asbestos is a problem on which work is conducted even in times of adequate supply. The United States depends so much upon foreign supplies of asbestos that availability of satisfactory substitutes in an emergency is advantageous.

The amosite problem is important because there is only one source—Africa—and substitutes are costly and less satisfactory.

The need for discovery and development of new asbestos deposits in the United States is a problem of high rank because this country depends upon foreign sources for 92 to 94 percent of its requirements.

To cope with problems as they arise, maintenance of adequate technical, statistical, and economic information is needed.

BACKGROUND

CHARACTERISTICS AND SOURCES

Asbestos is a name applied to a group of naturally fibrous minerals. The principal variety is chrysotile, a hydrous magnesium silicate having the chemical composition $3MgO \cdot 2SiO_2 \cdot 2H_2O$. Other commercial varieties are amosite (a complex iron-magnesium hydrous silicate) and crocidolite (an iron-sodium hydrous silicate). Of minor importance are tremolite ($2CaO \cdot 5MgO \cdot 8SoO_2 \cdot H_2O$) and anthophyllite ($7Mg,Fe,O \cdot 8SiO_2 \cdot H_2O$) (*16*).[2] A material that has the characteristics of silk or cotton but will not burn, is unique. Because it combines these qualities with other advantageous properties, asbestos has specialized uses for which few adequate substitutes are yet available. Asbestos furnishes a major raw material for a great variety of essential products; their manufacture constitutes an important industry.

To comprehend properly the asbestos industry, it must be viewed from a world standpoint, because, although the United States is the world's leading manufacturer of asbestos products, domestic sources furnish only 6 to 8 percent of the raw asbestos needed. Most United States supplies are obtained in Canada; but important quantities, particularly of special kinds and grades, originate in Africa.

Quebec, Canada, has the largest mines and mills in the world. Soviet Russia is next in importance as a producer. Southern Rhodesia, Union of South Africa, Swaziland, Cyprus, and the United States are substantial producers, and smaller supplies originate in many countries. The variety chrysotile constitutes about 95 percent of world supplies, and unless otherwise designated, the data herein relate to it.

Total world production in 1953 is estimated at about 1,375,000 metric tons of all grades and varieties. Of this total, Canada produced 60 percent; the Soviet Union, 16 percent (estimated) ; the Union of South Africa, 6 percent; Southern Rhodesia, 6 percent; the United States, 4 percent; and Swaziland, 2 percent. The total value of world production, exclusive of Russia and its satellites, was approximately $100 million in 1953.

The one large producer in the United States operates near Hyde Park, Vt., in what is regarded as a southern extension of the Quebec deposits. Approximately a dozen operators have small chrysotile mines in an area about 60 miles long and 25 miles wide in the Salt River and Cherry Creek basins of Gila County, north of Globe, Ariz. Chrysotile deposits are known in California, Montana, Wyoming, and some other States, but production has been negligible. In addition, a sporadic output of amphibole asbestos, which has limited use, is produced by a few operators.

The largest known asbestos deposits in the world occur in an area 70 miles long by 5 or 6 miles wide between Danville and East Broughton in the Province of Quebec, Canada (*8*, *17*). Seven companies are actively mining in this area; five of them operate large asbestos-products plants in the United States. Several other companies are prospective producers. Recently one of the companies began to operate a mine and mill in Munro Township, Ontario, 325 miles northwest of Ottawa. Another mine is under development in British Columbia. The value of Canadian production in 1953 exceeded $86 million, whereas that of the United States had a sales value of only $4,857,000.

The Russian deposits—in the Bajenova district of the Urals—are extensive. The mines and mills are large, and most of the fiber is used within the country in manufacturing various asbestos products (*18*).

Southern Rhodesia—particularly the Shabáni area—is an important producer of chrysotile asbestos. The fiber from these deposits is so low in iron that it has been used extensively for electric wire and cable coverings, whereas most other chrysotile deposits of the world, except the comparatively small ones in Arizona and a new large deposit in British Columbia, produce the ordinary type. Swaziland and the Union of South Africa also produce chrysotile in large quantities. The Union also furnishes two special types of asbestos—amosite, found commercially nowhere else in the world, and crocidolite (blue asbestos) produced also in small quantities in Australia and Bolivia. Most African production is exported (*11*).

There are smaller operations in Cyprus, Italy, Finland, France, Australia, Bolivia, Brazil, Venezuela, and several other countries.

GEOLOGY

Most chrysotile asbestos deposits of the world, including those of Vermont, Canada, Southern Rhodesia, Swaziland, Union of South Africa, and the Soviet Union, consist of irregular crossfiber veins or slip-fiber zones in massive

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

serpentine. Such deposits commonly extend to unknown depths. Those of Arizona, however, consist of more or less horizontal asbestos-bearing serpentine zones in massive limestone (*21*, *23*). Such deposits are generally less extensive and less persistent than those in massive serpentine. The amosite and crocidolite of the Union of South Africa occur in schists, quartzites, and banded ironstones that are so folded and contorted that the veins are quite irregular.

MINING

In Vermont the fiber-bearing rock is removed from an open pit with a face about 100 feet high. The general practice in Arizona is to drive a 5- to 6-foot heading in barren limestone beneath the fiber zone, which is later blasted down and hand-cobbed. The Canadian mines were originally open pits, but there has been gradual evolution to underground mines using block-caving methods (*19*). The chrysotile of Southern Rhodesia, Union of South Africa, and Swaziland is obtained from underground mines. Amosite is mined chiefly from large underground workings and blue asbestos from small open pits and shallow mines. The Russian deposits are worked both in open pits and underground. Cyprus has large open-pit workings.

MILLING

Asbestos milling is a complex operation involving primarily separation of fiber from rock and classification of the fiber by length. Asbestos is not a single product but comprises a series of grades depending upon the length of individual fibers. For instance, chrysotile fibers having an average staple length of three-eighths inch or longer are classed as spinning grades suitable for making heat-resistant textiles. Shorter grades have entirely different uses. The longer the fibers the more valuable they are. Although there have been some efforts to standardize asbestos classification, as in Canada, no asbestos-classification system has been universally accepted. The asbestos of commerce, therefore, is sold under a wide variety of grades, some of which have validity only as applied to the output of a single mine. In the asbestos report prepared by the Bureau of Mines for the National Security Resources Board, Materials Survey—Asbestos, a section on grading and classification discusses in detail the methods of classification used in the principal producing countries.

In Arizona the hand-cobbed rock is reduced in jaw crushers and rolls and the fiber separated by screening into the following grades: No. 1 is ¾ inch long or longer; No. 2 is ⅜ to ¾ inch; No. 3 is ⅛ to ⅜ inch; and No. 4 is less than ⅛ inch. Some mills produce filter fiber by retreating Nos. 3 and 4 with hammer mills, screens, and air separators. Nos. 1 and 2 are classed as spinning grades if the fibers are not too harsh.

In Vermont and Canada the mills are large and complex. The fiber is classified in many grades, such as spinning, cement stocks, paper stock, etc., with several defined grades in each classification.

RESERVES

In Vermont reserves of at least 1 million tons of fiber appear to be assured. A small proportion of this quantity is of spinning length. Arizona reserves are difficult to estimate, but proved reserves are small. According to present estimates, domestic reserves are far from adequate to meet current or future demands.

The world as a whole appears to have adequate asbestos reserves for at least 25 to 30 years at current or moderately enlarged rates of output. The chief suppliers of United States markets—Canada and Africa—appear to have reserves adequate for long-range planning. The Soviet Union probably has reserves large enough to supply its domestic economy for many years.

USES

Asbestos has a great variety of uses, which depend primarily on length of fiber. The longer fibers—Canadian groups 1, 2, and 3, and their equivalents from other countries—are used for making textile products, such as rovings, yarns, tapes, and cloth. The asbestos is spun and woven in much the same way as cotton, silk, or wool. Asbestos fabrics are used extensively for lagging cloth, brakeband linings, clutch facings, safety clothing, packings, gaskets, laminated plastics, and various other commodities.

Shorter fibers, such as Canadian group 4 and to some extent groups 5 and 6, are used widely for asbestos-cement products, such as roofing shingles, flat and corrugated siding, and pipe. Rhodesian grades C & G 3 and 4 and blue fibers are used in the latter products. These products consist of about 80 percent portland cement and 20 percent asbestos. Canadian group 4, or sometimes longer grades, as well as African amosite, is used extensively for making 85-percent magnesia block and pipe insulation consisting of about 85 percent basic magnesium carbonate and 15 percent asbestos. Canadian group 5 is suitable for asbestos paper and millboard manufacture. Such products are made on standard papermaking machines. An important use of asbestos paper is for making so-called air-cell pipe coverings. The shortest fibers are used for boiler and roofing cements and as filler in asphalt floor tile and various other products.

Amosite is used for felted lagging for high-temperature insulation, as a loosely compacted covering for marine turbines, jet engines and similar applications; and as a constituent of 85-percent magnesia and lightweight, fireproof partition board.

Long-fiber crocidolite (blue asbestos) is woven into fabrics used for locomotive-boiler lagging (in Great Britain) and for acid-resistant packings and gaskets. The principal use of the shorter blue fibers is in making asbestos-cement pipe. Bolivian blue asbestos is used for making gas-mask filters.

The amphibole fibers (tremolite and anthophyllite) are used for chemical-resistant filters, as welding-rod coatings, and as fillers in various products.

PRODUCTION, CONSUMPTION, AND FOREIGN TRADE[3]

World production of asbestos, by country of origin, is given in table 1. Table 2 shows the production of leading countries, by type of asbestos.

The United States is the largest consumer of asbestos in the world. Table 3 shows consumption over a period of years. Exports are generally too small to be significant and they are not shown herein.

As the United States furnishes only a small percentage of its asbestos requirements from domestic sources, foreign trade is of the utmost importance. Table 4 shows imports and the principal country of origin.

On a tonnage basis, Canada furnishes about 95 percent of the United States imports, but most of this material is of the short grades. The comparatively small quantities received from Africa are more important than would appear on a tonnage basis, because they consist largely of special kinds and qualities unobtainable elsewhere. Under emergency conditions imports of spinning fibers from Canada are of primary importance. Such imports appear in table 4.

The so-called C & G grades 1 and 2 (spinning fibers) from Southern Rhodesia have been very important because they are suitable for making electric-cable insulation required in large quantities by the Navy. Imports of these grades are indicated in table 4. Increasing imports of low-iron chrysotile from British Columbia were noted in 1954, but statistics are not yet available. British Columbia is expected to become a major source of low-iron chrysotile.

Imports of amosite and blue asbestos appear in table 4.

Most of the blue fiber originates in Africa, but increasing quantities are derived from Australia. A small tonnage coming from Bolivia has been especially important for gas masks.

SELF-SUFFICIENCY

The United States depends upon foreign sources for 92 to 96 percent of its requirements of all grades of asbestos and for approximately 99 percent of its needs of the spinning grades that are most important in an emergency. Except for a small production in Arizona and beneficiated ordinary chrysotile, the entire United States supply of low-iron chrysotile comes from Southern Rhodesia and a newly developed deposit in British Columbia, Canada. For amosite which is also of strategic importance, the United States depends 100 percent on the Union of South Africa. For crocidolite the United States also depends entirely on foreign sources. Bolivian crocidolite is available only in limited quantities, but the demands are also small.

Asbestos kinds and grades of greatest strategic importance are being stockpiled. The three stockpile items are Rhodesian and Arizona low-iron chrysotile or their equivalents, amosite, and Bolivian blue. Canadian chrysotile of spinning grade is regarded as essential for many direct and indirect military uses, but it is not being stockpiled.

PRICES

A strong upward trend in prices began in 1945 and continued until January 1952. During 1953 and 1954 they remained virtually unchanged from 1952. Prices of asbestos vary greatly, depending upon fiber length and quality. Canadian crudes (hand-cobbed fibers of spinning length) ranged in price from $500 to $1500 a ton in 1952, but a very small tonnage of such fibers is marketed. Canadian milled fibers of spinning length ranged from $300 to $525 a ton. The shorter fibers commanded successively lower prices down to a range of $37 to $68.50 for the shortest grades sold. Vermont and Canadian prices cover approximately the same range. Arizona prices were somewhat higher, but sales were small.

TARIFF

Asbestos (unmanufactured) enters the United States duty free.

RESEARCH

Companies operating mines in Canada have conducted research on production methods. Considerable research has been done on utiliza-

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[3] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

TABLE. 1.—*World production af asbestos, unmanufactured, by principal countries, 1949–53, in metric tons*

(Compiled by Helen L. Hunt)

Country	1949	1950	1951	1952	1953
Africa:					
Southern Rhodesia	72, 246	64, 888	70, 454	76, 960	79, 595
Union of South Africa	64, 334	79, 300	97, 402	121, 416	86, 016
Swaziland	30, 814	29, 635	31, 719	31, 542	27, 309
Australia	1, 671	1, 643	2, 599	4, 124	5, 049
Canada [1] (sales)	521, 543	794, 095	882, 866	843, 078	826, 303
Cyprus [2]	12, 556	14, 989	17, 180	16, 556	14, 484
Finland	10, 486	10, 949	11, 850	6, 100	10, 929
Italy	15, 877	21, 433	22, 612	23, 938	20, 397
Japan	5, 456	5, 664	6, 139	3, 060	4, 078
U. S. S. R. (estimate)	191, 000	220, 000	220, 000	220, 000	220, 000
United States	39, 360	38, 495	46, 851	48, 864	49, 401
Other (estimate) [3]	9, 657	63, 909	15, 328	29, 362	31, 538
Total (estimate) [3]	975, 000	1, 300, 000	1, 425, 000	1, 425, 000	1, 375, 000

[1] Exclusive of sand, gravel, and stone. [2] Exports. [3] Includes small production in about 20 countries.

TABLE 2.—*Production of asbestos in leading countries, in short tons, 1949–53*

Year	Canada chrysotile	Southern Rhodesia chrysotile	Union of South Africa		
			Chrysotile	Amosite	Crocidolite
1949	574, 906	79, 638	7, 609	41, 974	21, 180
1950	875, 344	71, 527	14, 334	42, 393	30, 598
1951	973, 198	71, 663	19, 509	54, 053	33, 659
1952	929, 339	84, 834	24, 970	63, 280	44, 735
1953	911, 226	87, 739	18, 840	38, 258	37, 707

TABLE 3.—*United States consumption and imports of asbestos, 1949–53, in short tons*

Year	Apparent consumption	Total imports	Imports from—			
			Canada	Union of South Africa	Southern Rhodesia	Other
1949	532, 708	509, 366	470, 783	22, 720	13, 722	2, 141
1950	727, 002	705, 458	678, 696	14, 907	9, 464	2, 391
1951	796, 992	761, 873	726, 770	23, 583	7, 725	3, 795
1952	752, 609	709, 469	668, 900	26, 902	10, 543	3, 124
1953	754, 218	702, 838	652, 117	37, 023	9, 990	3, 708

TABLE 4.—*Special grades of asbestos imported for consumption, 1949–53, short tons*

Year	Canadian chrysotile crudes [1] and spinning or textile fiber	Rhodesian chrysotile crudes [2]	Amosite	Crocidolite
1949	14, 584	4, 175	14, 898	5, 260
1950	25, 247	3, 968	8, 220	5, 434
1951	23, 199	1, 917	15, 131	6, 076
1952	24, 667	1, 825	18, 323	7, 572
1953	20, 259	1, 853	15, 261	10, 364

[1] Classified as grades 1 and 2 and Other, by the U. S. Department of Commerce.
[2] Includes grades 1 and 2.

tion of the more plentiful shorter grades in applications where longer grades are preferred. For instance, it has been found that more complete fiberization, permitting better contact of cement and fiber, will allow groups 5 and 6 to be blended in increasing proportions with group 4 in making asbestos-cement products.

Two companies have carried on and are continuing research on removing iron from Canadian asbestos to make it suitable for electrical insulation. The Johns-Manville Co. has a plant at Tilton, N. H., where it manufactures low-iron Quinterra paper from purified short-fiber Canadian chrysotile. Several grades are now marketed for electrical insulation. Raybestos-Manhattan, Inc., has developed a deironing process applied to long-fiber Canadian chrysotile. The products—Novabestos paper and tapes—are in commercial production.

The Naval Research Laboratory has done experimental work on the possible use of a papermaker's Vortrap to remove iron from Canadian chrysotile. Separation is accomplished in a Vortrap by the swirling action that arises when rapidly flowing water is introduced into a pipe in a tangential manner. The centrifugal action tends to throw the heavy magnetite against the walls, while the asbestos fibers, freed from magnetite, pass upward in the center of the pipe by countercurrent. More comprehensive research with this and related equipment is being conducted by the Bureau of Mines.

The Department of Defense has developed fiber-glass gas-mask filters that have been accepted widely as substitutes for those of Bolivian blue asbestos. Accordingly, the demand for Bolivian blue asbestos is less urgent than formerly.

Considerable research is being done on making synthetic asbestos and other asbestiform products, but comparatively little progress has yet been made. A variety of amphibole asbestos has been synthesized experimentally by the Bureau of Mines (*14*), and very small fibers have been identified as chrysotile by several other research organizations. Chrysotile—the type most in demand—is difficult to synthesize. Details of processes and problems are contained in Materials Survey—Asbestos, mentioned in the bibliography at the end of this report.

Organic plastics may be used in an emergency for low-pressure packings and gaskets, but they are inferior to asbestos products. Silicone varnishes and silastic rubbers are being used increasingly as class H electrical insulation for service temperatures up to 500° F.

OUTLOOK

As indicated in table 4, the consumption of asbestos in the United States has increased remarkably during recent years. Asbestos is widely used in industry in such products as friction materials, pipe covering, packings, and gaskets. The building industry also provides a large market for asbestos-cement products (roofing and siding), floor tile, and insulation. Accordingly, the trend in asbestos consumption follows the trends of industrial production and building construction. Expansion in asbestos consumption for most of the major well-known uses is forecast by some manufacturers of asbestos products; because of its unburnable, fibrous structure, asbestos is constantly finding new uses where satisfactory substitutes are uncommon. Accordingly, a strong and growing demand for asbestos is anticipated by the industry.

To meet these needs, production facilities are increasing. In the Quebec asbestos area a large, new mill has recently come into production, and another large one was ready for operation early in 1955. A new asbestos-mining project involving the expenditure of many million dollars will probably reach the production stage in 4 or 5 years.

In northern British Columbia, Canada, a new mill went into regular production in July 1954. The deposit contains an exceptionally large proportion of low-iron spinning fiber. Annual production, beginning July 1, 1954, is estimated by the company at 10,000 tons of spinning fiber.

Several mines in the Union of South Africa and Southern Rhodesia are expected to reach the production stage in the near future. No significant new sources of asbestos are in sight in the United States. The single large operation in Vermont may expand moderately its output, which consists chiefly of the shorter grades.

PROBLEMS

SEARCH FOR DOMESTIC SOURCES OF SUPPLY

The dependence of the United States upon foreign sources for 92 to 94 percent of its total needs of asbestos and for at least 99 percent of its requirements of the important spinning grades presents as major problems the discovery and development of new domestic asbestos deposits. Exploration of promising asbestos areas by competent geologists, including detailed mapping, is desirable. Many serpentine areas that may contain asbestos have been inadequately explored, and a thorough study of them might reveal the presence of commercial deposits.

DEVELOPMENT OF SUBSTITUTES FOR ASBESTOS

Currently the supply of chrysotile from foreign sources is more than adequate to satisfy United States needs, but the situation may change and become less favorable. Although shortages are less likely to occur in the future than in the past, development of substitutes offers a means of making the Nation more nearly self-sufficient in an emergency.

Much work has already been done on the substitution of glass fibers for asbestos. The supply of Canadian spinning fiber was extended considerably during World War II by making fabrics containing both asbestos and fiber glass. Such fabrics are still being manufactured, and with continued improvements in glass fibers such substitution may be increased.

SYNTHESIS

Although no immediate or near-future relief from United States dependence upon foreign sources can be expected as a result of studies in asbestos synthesis, research on synthetics finds a place among the continuing long-range problems of the asbestos industry. Such investigations visualize use of the principles and techniques such as those employed in mica synthesis. Research to date indicates that fibrous products can be made. The variety of product need not be limited to varieties of asbestos now considered of commercial grade but may include asbestiform materials unknown in nature that have unique and advantageous properties.

THE PROBLEM OF LOW-IRON CHRYSOTILE

One of the most critical situations in asbestos has been the shortage of low-iron chrysotile used by the Navy for electric cable insulation. Aside from a small production of low-iron fiber in Arizona, United States needs have been supplied entirely from a source in Southern Rhodesia, and the volume of supply from that source has steadily diminished. Relief was sought by (1) promotion of the use of substitute materials and (2) beneficiation of ordinary asbestos to render it serviceable.

Conditions have recently changed greatly. The new asbestos-mining enterprise in British Columbia, Canada, will evidently produce about five times as much as was obtained annually from Southern Rhodesia during recent years.

Another factor that modifies the nonferrous asbestos problem greatly is the diminishing demand. New designs of shipboard cable, used in new naval vessels, do not require nonferrous asbestos fiber. The demand for nonferrous fiber in an emergency may be less than it was during World War II.

Because of increasing production and diminishing military essentiality, the low-iron problem therefore appears to be approaching solution.

The softer Arizona fibers are of superior quality for low-iron uses, but the mines are small, and maintenance of output is difficult.

PROBLEMS IN UTILIZATION OF CHRYSOTILE

Each asbestos mine produces fibers of several grades, according to length. Some grades are more in demand than others, and continuing research is required on blending or more precise preparation to develop wider usage of surplus grades.

Recent trends toward use of fluid transmissions and molded brake linings have reduced the market for woven friction materials, and the industry consequently has a serious textile market problem.

SUBSTITUTES FOR AMOSITE

The amosite situation is less critical than formerly because of increasing mine output; however, amosite is derived only from a limited area in South Africa, and this fact has led to a search for alternate materials. Although fiber glass is not widely used as a substitute for amosite (for price and technical reasons), this line of research offers promise, particularly for high-temperature-insulation uses. The recent manufacture of high-silica glass fibers in submicron sizes suggests the possibility of developing a fiber having the desirable characteristics of amosite.

SUBSTITUTES FOR CROCIDOLITE

The crocidolite problem was regarded as important during the early postwar period because Bolivian blue asbestos was a preferred material for making gas-mask filters. However, the Department of Defense has developed a fiber-glass filter pad that is said to be superior to those made of asbestos. The remoteness of the sources of supply of crocidolite used in asbestos-cement pipe is a fundamental supply problem. One of the newer plastics, because of its temperature stability up to 480° F., its resistance to acids and alkalies, and its low absorption is competing with crocidolite in acid-proof packings.

DEVELOPMENT OF NEW USES FOR TREMOLITE AND ANTHOPHYLLITE

The amphibole varieties of asbestos—tremolite and anthophyllite—occur in many places in the United States, but production has always been small. The uses have been confined chiefly to making chemically resistant filters, welding-rod castings, and fillers or additives in rubber, wall coatings, battery boxes, molded insulation, and other products. Amphibole asbestos, although almost always low in strength and flexibility, has higher heat and chemical resistance than chrysotile. The need for wider utilization is the basic problem of these varieties.

STATISTICAL AND OTHER INFORMATIONAL SERVICES

Producers, consumers, and Government agencies concerned with the commodity all need reliable technical, statistical, and economic information to guide management and Government activities.

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BARIUM

By

Joseph C. Arundale [1]

THE DOMESTIC barite industry, which is about a century old, is growing rapidly in importance to the national economy. Deposits are widespread in occurrence, and reserves are adequate to sustain new and increased uses. It is often wasted when other minerals are mined.

Summary

Barite (barium sulfate) is the principal barium mineral of commerce. Barite consumption has doubled in the past 10 years principally because of increased use in oil-well drilling fluids in high-pressure geologic formations, such as some of those in Texas, Louisiana, and California. It is also used in lithopone and glass and as the raw material in manufacturing various barium chemicals.

The reserve situation is reasonably satisfactory. The largest known reserves are in Missouri, California, Nevada, and Arkansas, and deposits are known in several other States and in many other countries. Barite is mined by both open-pit and underground methods and in the larger operations is concentrated by froth flotation.

The outlook for barite is for wider utilization and increasing consumption. Industry has the long-range problem of finding additional favorably located deposits. Research is needed on the properties of barium in its many forms and on the problem of recovering barite from complex ores.

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[1] Assistant chief, Branch of Construction and Chemical Materials, Bureau of Mines.

BACKGROUND

HISTORY

Barite, the principal barium mineral, has been mined in the United States since the middle of the 19th century. It is reported as first having been mined in Fauquier County, Va. A few years later it was discovered in south central Missouri and several years later in Tennessee. Production was reported from Georgia in about 1900 and a few years later from Kentucky. A deposit near El Portal, Calif., was mined in 1914, and another major deposit in this area first was worked in 1920. The deposit from which barite now is being produced in Arkansas was discovered in 1900; however, little exploratory work was done until about 1930. Sustained production was begun in 1941.

The first use of barite in the United States was as a filler in paint. Lithopone first was produced in the United States in 1892. Other barium chemicals were first produced by the Chicago Copper Refining Co. at Blue Island, Ill., in 1908. The first washers and jigs were installed in Georgia and Tennessee in 1914 and 1916. Later, in 1923 and 1924, the first washers were installed in Missouri. The first separation of barite from associated minerals by flotation was done by Magnet Cove Barium Co. at Malvern, Ark., in 1941. The use of barite as a weighting agent in well-drilling mud was begun in 1926.

PRODUCTION

The total production of primary barite in 1953 was 920,000 tons valued at approximately $9 million. The largest producers are Baroid Division, National Lead Co., and Magnet Cove Barium Co., a subsidiary of Dresser Industries. Over one-third of the total domestic output of barite in 1953 was produced in Arkansas. There are 15 to 20 companies that usually produce 10,000 to 30,000 tons a year, and numerous companies or individuals that produce less than 10,000 tons and down to a few tons a year. Most of the output of these smaller producers and also of many of the medium-size firms is sold to the larger companies or custom grinders.

Barite produced in the Far West is consumed principally in California for well drilling, lithopone, and barium chemicals. Virtually all Arkansas barite is used in well drilling on the Louisiana and Texas gulf coasts. Some Missouri barite goes to well drilling on the gulf coast; some goes to pigments, chemicals, glass, and fillers in the midwest and some to the eastern glass, chemicals, lithopone, and filler industries. Most of the southeastern barite is used in nearby chemical plants, and some goes to other Eastern States.

TABLE 1.—*Barite sold or used by producers, 1953*

	Short tons
Arkansas	380, 763
Missouri	330, 763
Georgia, Tennessee, and South Carolina	81, 846
Nevada	99, 525
Other States [1]	51, 315
Total	944, 212

[1] Arizona, California, Idaho, Montana, New Mexico, and Washington.

OCCURRENCES

Many foreign countries have deposits of barite and several produce substantial tonnages. The principal foreign producers are Nova Scotia, Mexico, Germany, Italy, Great Britain, Greece, Algeria, Japan, Brazil, Yugoslavia, France, Spain, Argentina, India, and U. S. S. R. Small tonnages are produced in Australia, Austria, Chile, Colombia, Morocco, Ireland, Peru, Portugal, Union of South Africa, Sweden, China, and Czechoslovakia.

Nova Scotia barite has been competing with domestic barite on the east coast and recently substantial tonnages have been shipped to Gulf coast ports. It also goes to South American ports. Small but increasing tonnages of Mexican barite are shipped to the gulf coast.

Principal types of occurrences are: (1) Residual deposits, in which lumps of barite remain as a result of weathering of barium-bearing rocks, chiefly dolomites in Missouri and the Eastern States; (2) total or partial replacement of limestone, shale, or sandstone, such as the deposits near Magnet Cove, Ark., in northern Nevada, in Cocke County, Tenn., at El Portal, Calif., and other areas. It also occurs as a gangue mineral or associated mineral in metalliferous ores, such as lead, zinc, silver, and in mixtures of fluorspar and barite and associated with bastnaesite in San Bernardino County, Calif. Barite may be commercially recoverable from some of these deposits.

TABLE 2.—*Primary barite in the United States, 1920–53*

[Short tons]

Year	Domestic production	Imports		Apparent new supply
		Crude	Ground	
1920	222, 173	24, 874	274	247, 321
1921	96, 287	11, 054	1, 002	108, 343
1922	133, 310	23, 239	4, 696	161, 245
1923	212, 253	15, 045	6, 022	233, 320
1924	201, 247	21, 502	2, 830	225, 579
1925	235, 515	28, 655	3, 203	267, 373
1926	237, 224	51, 016	3, 347	291, 587
1927	214, 194	70, 274	3, 259	287, 727
1928	259, 761	61, 765	3, 098	324, 624
1929	275, 945	85, 729	2, 924	364, 598
1930	237, 505	52, 111	2, 331	291, 947
1931	210, 930	73, 080	1, 851	285, 861
1932	133, 572	45, 758	1, 594	180, 924
1933	146, 402	49, 958	2, 632	198, 992
1934	178, 361	40, 031	1, 863	220, 255
1935	218, 075	47, 048	3, 354	268, 477
1936	274, 062	33, 843	2, 873	310, 778
1937	360, 877	64, 992	3, 313	429, 182
1938	335, 433	24, 845	1, 700	361, 978
1939	365, 870	11, 588	1, 590	379, 048
1940	390, 462	7, 391	314	398, 167
1941	483, 391	456	----------	483, 847
1942	449, 873	4, 680	----------	454, 553
1943	429, 298	----------	----------	429, 298
1944	515, 136	67, 888	11, 964	594, 988
1945	692, 330	56, 894	1	749, 225
1946	725, 223	44, 662	----------	769, 885
1947	884, 219	53, 222	----------	937, 441
1948	777, 841	53, 204	(1)	831, 045
1949	731, 308	26, 178	211	757, 697
1950	693, 424	58, 381	678	752, 483
1951	845, 579	52, 755	160	898, 494
1952	1, 012, 811	107, 918	6, 620	1, 127, 349
1953	920, 025	334, 788	259	1, 255, 072

[1] Less than 0.5 ton.

RESERVES

Barite deposits in the United States are numerous and widely distributed. Reserves are considered fairly large. The principal producing areas are in Washington County, Mo. and the Ouachita Mountains of Arkansas. Reserves in Washington County have been estimated to be at least 15 to 20 million tons. The two producers in Arkansas are reported to have proved 9 million tons of reserves. Bureau of Mines investigations have revealed important reserves in nearby Montgomery, Polk, and Pike Counties. Deposits at Battle Mountain, Nev., have been estimated to contain at least 2½ million tons. The total known commercial reserves in the United States have been estimated to exceed 40 million tons. Low-grade or marginal deposits and byproduct barite would enlarge this reserve considerably.

Tables 2 and 3 show production, total new supply, and consumption of barite.

TECHNOLOGY

Methods of mining and preparing barite generally depend on the type of deposit, its location, and the intended use. In Missouri, Tennessee, and Georgia, barite-bearing clay is dug with power shovels in open pits. The clay is removed in log washers. Fine barite lost in the log washer overflow may be recovered by tables and flotation cells. The two largest barite producers in the United States, which recover barite from a shale formation near Malvern, Ark., employ both open-pit and underground mining methods. This material is ground and the barite recovered by flotation. In various localities barite has been mined from veins by shrinkage-stoping methods and recovered in jigs or by open-pit methods, followed by hand picking and screening. Previously barite was mined by hand methods in Georgia, Tennessee, and Missouri, but much of this output has now been mechanized; however, some hand mining is still done in Missouri. The recovered barite is hand-cobbed and usually sold to larger firms having preparation facilities.

Barite is ground both wet and dry. For use in well drilling it is usually ground in ball mills, either wet or dry. In Missouri unbleached barite is dry-ground, and barite to be bleached is wet-ground.

USES

The largest use of barite and one that takes nearly three-quarters of the total output is as a weighting agent in rotary well-drilling fluids.

TABLE 3.—*Consumption of barite in the United States, 1943–53, by uses*

[Short tons]

Use	1943	1944	1945	1946	1947	1948	1949	1950	1951	1952	1953
Well drilling	144, 452	277, 792	407, 871	372, 610	467, 350	565, 249	494, 579	483, 519	594, 668	758, 240	824, 050
Chemicals [1]	99, 097	100, 921	99, 173	102, 439	107, 267	100, 038	80, 584	108, 350	132, 268	123, 597	163, 470
Lithopone	129, 493	134, 597	139, 288	154, 166	167, 321	153, 987	71, 710	99, 703	107, 094	61, 000	52, 308
Glass	25, 464	24, 153	25, 761	29, 181	33, 641	23, 580	21, 768	24, 638	25, 779	24, 604	24, 853
Paint	15, 000	23, 000	21, 000	26, 000	29, 000	22, 000	20, 000	28, 000	28, 000	25, 000	24, 000
Rubber	8, 000	10, 000	10, 000	20, 000	17, 000	18, 000	14, 000	19, 000	15, 000	18, 000	21, 000
Aggregate	----------	----------	----------	----------	----------	----------	(2)	15, 784	38, 143	12, 000	25, 000
Other	32, 238	25, 100	17, 810	17, 677	14, 239	11, 455	16, 902	7, 137	11, 316	1, 584	1, 181
Total	453, 744	595, 563	720, 903	722, 073	835, 818	894, 309	719, 543	786, 131	952, 268	1, 024, 025	1, 135, 862

[1] Includes some witherite in certain years.

[2] Barite used as aggregate in 1949 and previous years included in "other" but probably was very small.

Such fluids serve several purposes—lubricating and cooling the bit, plastering the walls to prevent caving, carrying the cuttings up the well to the surface, and the purpose for which barite is used, restraining abnormally high gas and oil pressures to their formation levels. There are high-pressure oil fields in the gulf coast areas of Texas and Louisiana and the San Joaquin Valley of California. This use for barite was patented in 1926 and the patent expired in 1943.

Barite crushed to about 16- to 20-mesh and added to a glass melt serves several purposes; it fluxes the heat-insulating froth that tends to form on the surface of the melt, thus saving fuel. It makes the glass more workable and increases brilliance. When crushed material is not available the industry can use ground barite.

A large tonnage of barite is used in lithopone, an intimate mixture of two precipitated salts—zinc sulfide and barium sulfate. Lithopone is used as a white pigment, principally in paints.

Barite is used as a filler or extender in many items, such as paint, inks, oilcloth, linoleum, rubber, asbestos brake linings, and other materials.

Another important use for barite is as a raw material in manufacturing various barium compounds, such as barium carbonate, precipitated barium sulfate, barium chloride, barium oxide, barium peroxide, barium hydroxide, barium nitrate, and others.

Precipitated barium sulfate, or blanc fixe as it is commonly called, is used as an extender and as a pigment in paints. Blanc fixe is used as a paper filler and in paints, printing ink, rubber, linoleum, oilcloth, and leather.

Barium chloride is used in case hardening, in leather and cloth, in making magnesium metal, in preventing scum on brick, and in water treatments. Fused barium chloride may be electrolyzed to produce barium metal.

Barium carbonate is used to prevent the formation of scum on ceramics and as a component of ceramic glazes and enamels. It is employed to diminish porosity and prevent discoloration in bricks and in making both crown and flint glasses. It is used in well-drilling muds.

Barium oxide is used in glass and in making barium peroxide. It is used in electric-furnace ferrous metallurgy to increase the life of acid furnace linings, produce a quieter and steadier arc, and reduce sulfur and slag viscosity.

Barium hydroxide prevents scumming of ceramic products. It is used to recover sugar from molasses by means of the barium saccharate process and in lubricating oils.

Barium nitrate is an ingredient in green signal flares, in primers and detonators, and in enamel frits.

Barium metal is a deoxidizer of copper. It has a high rate of electron emission when subjected to an electric potential and this property is utilized in alloys for spark plugs and electron emission elements in electronic tubes.

Minor quantities of other barium compounds have various applications.

The physical and chemical specifications for barite vary according to end uses. In well-drilling muds the material must be heavy and inert. Grinders attempt to produce a material of minimum 92 percent $BaSO_4$ free of soluble salts, such as calcium sulfate. Most grinders of well-drilling barite guarantee 90 to 95 percent passing 325-mesh and a specific gravity of not less than 4.2. Several percent of iron oxide is not objectionable. For chemical purposes, a minimum of 94 percent of $BaSO_4$ is generally specified with a maximum of 1 percent of Fe_2O_3; a maximum of 1 percent of strontium sulfate, with only a trace of fluorine permissible. Strontium sulfate in material intended for lithopone use might be somewhat higher. Mesh size is important to most consumers for chemical purposes; too fine a material results in dust loss, and too coarse a material results in poor mixing with powdered carbonaceous materials. Most consumers specify a material in the range of 4- to 20-mesh; however, some consumers grind lump to this specification. Glassmakers have a specification usually requiring a minimum of 98 percent $BaSO_4$, maximum SiO_2 1.5, Al_2O_3 0.15, and Fe_2O_3 0.15 percent. In mesh size they prefer a mixture ranging from 30-mesh down through 140. Material ground to as fine as 325-mesh is not desirable, because of its tendency to ball up in the batch, but in times of short supply even this has been used. The most objectionable impurity is iron. Specifications of barite used for filler and aggregate are not so strict, and in most instances mesh size is most important.

SUBSTITUTES

In some of its uses barite is not absolutely essential. In its largest single use—as an admix in well-drilling fluids—barite, because of its high specific gravity, low cost, softness, cleanness, and inertness, is conceded to be the most desirable weighting material; however, there are less desirable substitutes. During the period when a royalty was charged for the use of barite in well-drilling fluids, celestite was employed for this purpose; the specific gravity of the latter is somewhat lower, and it does not now compete in this field. Some iron ores would be heavier and as cheap or cheaper than barite; other than possibly being more abrasive, there apparently are no technical reasons why such material could not be used, but they are objectionable because of the heavy staining of personnel. Galena, with nearly twice the specific gravity of barite, is many times more expensive and has some technical disadvantages. Any of these materials could be used, but because of its more desirable properties, continued and increased use of barite for this purpose is anticipated.

In recent years titanium dioxide production has been greatly increased, and in some markets has been progressively displacing lithopone. Titanium is more expensive, but its covering power in paint is much greater. The price of lithopone now is maintained at about one-third that of titania to compete, and it may decline further when the supply of titania becomes adequate to meet all needs. Crude barite represents a significant fraction of the cost of finished lithopone. The future of lithopone depends as much on the price as on quality competition from titanium dioxide.

Barite as a surface flux in glass has competitors. Salt cake previously was used and is now selling at roughly the same price as barite. While a royalty of $13 per ton was being charged on the use of barite in well-drilling muds, a substantial tonnage of other material, particularly celestite, was used.

SELF-SUFFICIENCY

Insofar as reserves are concerned, the United States is potentially self-sufficient in barite. However, certain economic factors favor the importation of barite and some barium compounds. The principal source of such imports is Nova Scotia. Occurring as a high-grade, easily worked, deposit near tidewater, this material can compete successfully with domestic barite on the eastern seaboard and the Gulf coast. Several new plants have been built along the Gulf coast to grind imported barite.

Although barite and its compounds have many important uses directly or indirectly related to national defense, security problems are few. The industry is widespread throughout the United States, with the greatest production and large reserves near the center of the country. There is no barite in the National Stockpile, and adequate supplies are largely a matter of mine and plant capacity. However, a shortage developed during World War II, and similar shortages may occur in future emergencies. The shortage was due primarily to lack of labor and materials for expansion. It was relieved through domestic expansion and increased imports.

Transportation of supplies obtained from foreign sources presents some wartime problems, but these are minimized by the proximity of the principal foreign suppliers.

TARIFFS

Import duties on crude barite have been successively reduced in the series of recent trade agreements at Geneva and Torquay and now stand at $3 per long ton on crude or unmanufactured barite ore and $6.50 per long ton on ground or otherwise manufactured.

TAXES

Under the Federal tax laws, producers of barite have a percentage depletion allowance of 15 percent of gross income not to exceed 50 percent of the net income without depletion deduction. Severance taxes are levied in various producing States.

COSTS

With crude barite now selling for about $13 a ton (minimum 94 percent $BaSO_4$ f. o. b. mines), deposits must be fairly high grade to permit profitable mining and milling. The cost of transportation is an important factor. Relatively low transportation cost is the principal reason why Nova Scotia barite, which is near tidewater, can compete successfully with domestic barite delivered to either the Atlantic coast or the Gulf coast. Barite mined in the Western States is used in well drilling and in lithopone, glass, and chemical plants in the West because it cannot carry the cost of transportation to other markets.

CONSERVATION

In recent years many firms have initiated flotation recovery of barite fines lost in washer and hand-mining operations. Mines have been more mechanized, and mining methods have been improved. Recovery at the larger mines is now quite efficient. One obvious waste of barite is in some metal-mining operations, where barite occurs as a gangue or associated mineral. With few exceptions, this material has been and is being lost. There is no estimate of the quantity of such material that has been discarded to dumps or the rate at which it is now being discarded, but it is believed that a substantial quantity could be recovered.

In most of its uses barite is consumed or incorporated into the product and is not recoverable. The one exception is its use in drilling mud. In well drilling some of the barite is lost in porous formations, but the greater part is returned to mud ponds on the surface, where it usually is abandoned when the well is completed. Recovery of this barite is possible, but the practice has not been widely adopted.

OUTLOOK

Lithopone has long been used as a white pigment and filler. However, in recent years titanium dioxide production has been greatly increased, and it has progressively displaced lithopone in its principal markets. It is expected that the trend toward substitution of this material for lithopone will continue, thereby further reducing the requirement for barite in this use.

The trend in petroleum and natural-gas well drilling is toward more wells and greater depths, thereby increasing the possibilities of encountering high-pressure formations with a consequent increase in the requirements for a weighting agent, such as barite, in well-drilling muds. No other material so satisfactorily meets the requirements for a weighting agent in drilling muds as does barite. Estimates by Petroleum Administration for Defense show a total footage of rotary wells as 276 million feet by 1956 compared with 170 million feet in 1951.

Another trend is the increased use of barite as an aggregate to weight concrete around pipelines in river crossings and swampy areas. A new use that could become substantial is a mixture of finely ground barite and synthetic rubber powder. This product is added to hot-mix asphalt in the construction of highways, airports, and parking areas.

There is a considerable interest in barium titanate and its usefulness in ultrasonics.

If and when atomic power plants are constructed for commercial use, barite as an aggregate in concrete shields may become very important; it is possible that large tonnages might be used for this purpose. Another use, which at the moment is potential, is as an aggregate in concrete buildings, shelters, and military installations as protection against radiation.

Research in metallurgical and chemical utilization of barium metal, compounds, and alloys may result in their having wider use in tonnage quantities.

The short-term-reserve outlook for barite is satisfactory. Known reserves of commercial-grade material are adequate for the near future at anticipated rates of consumption.

The long-range outlook for barite supplies may not be quite so favorable. With an anticipated annual requirement of at least 1,600,000 tons by 1965, known domestic reserves do not appear so large. However, chances of finding new deposits of barite or extensions of known deposits are considered to be good. There are large deposits of ore consisting of an intimate intergrowth of barite and fluorite. Improved metallurgical methods may be devised for separating these two minerals and thereby increasing the reserves and supplies of both. There are such deposits at Tonuco Mountain, Dona Ana County, N. Mex.; Gonzales and other areas in Socorro County, N. Mex.; Sweetwater district, eastern Tennessee; and several deposits in central Tennessee, central Kentucky, and the Illinois-Kentucky fluorspar field. Low-grade barite deposits occur in numerous other localities throughout the country. The recovery of barite as a byproduct of other types of operations may also increase the supply. Such possibilities are in the rare-earth deposits near Mountain Pass, Calif., and in lead, zinc, silver, and other complex ores throughout the country. There are barite production, potential production, and large reserves in many foreign countries. The most important of these reserves, insofar as the United States is concerned, are in Nova Scotia, Mexico, and Cuba. There has been output from all of these sources in the past.

PROBLEMS

The important problems confronting the barite industry are those of reserves; conservation (including more efficient recovery, distribution, and utilization of barite); development of new uses; and transportation and other items of cost.

The problem of reserves is a long-range one. At the current rate of consumption, domestic and available foreign reserves are adequate to the needs of the industry for many years. However, consumption has doubled in the last decade. The overall requirements for barite are increasing, so it may not be long before the industry will find known reserves inadequate, particularly in deposits that are favorably situated as regards current markets. In evaluating national reserves of barite, it is not enough simply to determine high-grade commercial reserves. It also is necessary to examine the marginal and submarginal deposits that may be commercial in a few years or under emergency conditions.

The problem of tariffs is also important to the industry. Competition from foreign sources is affecting the major producing areas.

If more information on the properties and applications of barium in its many forms can be accumulated through research or other investigations, the requirements for this commodity can be met more effectively. Both technical and statistical data are needed to reveal clearly the utility and essentiality of the commodity. Such studies promise development of new applications and possible substitution of barite or barium compounds for strategic materials or others in short supply.

Further research is needed on methods of beneficiating barite, particular attention should be given those ores in which barite occurs in association with other minerals, thereby increasing not only supplies of barite but those of other and perhaps less plentiful or more essential materials, such as fluorspar, lead, and zinc.

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BERYLLIUM

By

Horace T. Reno[1]

BERYLLIUM is a metal with many unique properties; their application has resulted in its wide utilization. It is the only stable light metal with a high melting point. Its use as an alloying material has provided many superior products.

Summary

The mineral beryl, at present the only commercial source of beryllium, occurs in pegmatites, granites, and syenites but is recovered from pegmatites only. Hand cobbing is still used to recover beryl from its ores, because mechanical methods have not yet been perfected. The Bureau of Mines has achieved some success in concentrating beryl by froth flotation; practical application of the process, however, apparently depends on continued research.

The principal beryl-producing countries of the free world in 1953, in the order of their contribution to the total production, were: Brazil, Southern Rhodesia, Argentina, United States, South-West Africa, Union of South Africa, Madagascar, Portugal, and Mozambique. Beryl exports are restricted by Brazil, India, the Union of South Africa, and Argentina. United States production comes principally from the Black Hills of South Dakota, followed by New Mexico and New Hampshire.

The beryl reserves of the United States have been estimated to be 15,000 tons in deposits containing 1.0 percent or more beryl and 280,000 tons in deposits containing about 0.1 percent beryl. World reserves have been estimated to be 190,000 tons containing 1.0 percent beryl and 3,500,000 tons containing 0.1 percent beryl.

The Beryllium Corp. and Brush Beryllium Co. are the largest companies in the United States using beryl or manufacturing beryllium alloys and compounds.

Over 90 percent of the beryllium consumed is used in alloys, mostly copper alloys. Beryllium-copper alloy is utilized widely because of its hardness, great tensile strength, high electric and thermal conductivity, low creep tendency, nonmagnetic and nonsparking properties, and resistance to fatigue, corrosion, and wear. Other useful beryllium alloys are made with each of the following metals: Zinc, nickel, cobalt, aluminum, molybdenum, platinum, iron, chromium, gold, and magnesium. Pure beryllium metal is used principally in manufacturing nuclear-energy and X-ray equipment, and beryllium oxide is used principally in nuclear-energy equipment and in the production of high-temperature refractories.

Substitutes are available to replace beryllium alloys in some applications; none, however, has all the desirable characteristics of the beryllium alloys.

During World War II and the Korean War the utilization of beryllium expanded rapidly, and the supply of beryl ore did not equal the demand.

The outlook for the United States beryllium industry is for an assured ore supply and expanded utilization of beryllium and beryllium alloys at lower prices. Problems are high prices, foreign ore supply, inefficient recovery of beryl from its ores, and inherent health hazards.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

MINERALOGY

Beryllium is a constituent of over 30 known minerals. The only present commercial source of beryllium, however, is the mineral beryl, a beryllium-aluminum silicate that theoretically can contain 14 percent beryllium oxide. Beryl occurs principally in long, prismatic, hexagonal crystals that usually have a vitreous luster and are emerald green or pale green, grading into light blue, yellow, and white. It has a hardness of 7.5 to 8.0, a specific gravity of 2.63 to 2.80 (usually 2.69 to 2.70), and a white streak. Transparent crystals of beryl—the green variety called emerald and the blue variety called aquamarine—are valued as gems.

SOURCES

Beryl occurs in pegmatites and granites but, at present, is recovered economically from pegmatites only. In pegmatites beryl, if present, is usually disseminated in small grains; occasionally, however, large crystals 2 to 3 feet in diameter and many feet long are found. Although many pegmatites contain beryl, few contain enough to be mined for that mineral alone. Almost all domestic beryl is recovered as a coproduct in mining one or more of the following minerals: Feldspar, mica, columbite, tantalite, lithium minerals, or cassiterite. Beryl, to be recovered economically, must be in crystals large enough to permit cobbing (hand sorting) inasmuch as mechanical methods of recovery are not yet perfected. In the United States, cobbing is usually limited to fragments larger than 1 inch in diameter, and in the average deposit about 300 tons of rock is handled to recover 1 ton of salable beryl concentrate. In cheap-labor countries the minimum size is about one-half inch.

The Federal Bureau of Mines, in its research on methods of beneficiating beryl ore, succeeded in eliminating expensive prior conditioning and floating beryl, after finding that usual floatation and gravity-concentration methods were inapplicable (*15*).[2] Practical application of the process, however, depends on continued research in a pilot plant.

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

TABLE 1.—*Salient statistics of the beryllium industry, 1936–53*

[Short tons]

Year	Production of beryl ore[1]		United States imports[1]	United States exports		Average price per unit BeO		Consumption[1]
	World	Domestic		Ore	Metal, alloys, compounds[2]	Domestic[3]	Foreign[4]	
1936	479	28	162	(5)	(5)	$3.00	$4.12	200
1937	413	75	182	(5)	(5)	2.19	4.42	200
1938	1,153	25	146	(5)	(5)	3.08	4.10	300
1939	996	95	459	(5)	(5)	2.86	3.18	500
1940	2,388	121	805	(5)	(5)	3.08	2.97	600
1941	4,507	158	2,666	(5)	(5)	4.62	5.40	1,200
1942	3,312	269	2,050	0.8	12.6	8.99	6.71	2,352
1943	6,008	356	4,840	2.5	27.5	12.47	7.80	3,058
1944	3,261	388	3,115	9.5	42.1	14.47	9.18	2,176
1945	1,085	39	1,201	.1	51.6	15.73	10.98	1,738
1946	1,873	100	1,188	1.4	69.9	17.79	8.90	1,013
1947	1,430	145	767	.2	140.7	17.39	14.95	1,735
1948	2,470	99	1,720	.1	13.0	26.87	17.41	1,970
1949	4,587	475	3,811	.3	94.0	32.10	22.52	1,029
1950	6,651	559	4,860	.1	110.5	30.51	25.43	3,007
1951	5,720	484	4,316	.3	94.8	33.34	31.67	3,388
1952	6,651	515	5,978	1.9	[6] 196.6	38.55	38.55	3,476
1953	4,587	751	8,245	0	103.7	47.00	47.00	2,661

[1] Average 10 percent BeO content.
[2] Beryl equivalent.
[3] F. o. b. mine.
[4] C. i. f. U. S. ports.
[5] U. S. exports were not classified separately before 1942.
[6] Does not include an undisclosed quantity of secondary material exported to United Kingdom.

Beryl-producing countries of the free world in 1953, with their output expressed in percentage of the total produced, were: Brazil, 25.4 percent; Southern Rhodesia, 18.9 percent; Argentina, 16.2 percent; United States, 8.0 percent; South-West Africa, 6.9 percent; Union of South Africa, 5.7 percent; Madagascar, 5.5 percent; Portugal, 4.4 percent; Mozambique, 2.9 percent; and others, 5.1 percent.

The States of Páraiba, Rio Grande del Norte, Ceará, and Alagoas, in northern Brazil, supply most of the Brazilian production, usually as a coproduct of columbite-tantalite mining. In the Union of South Africa, beryl is mined in a large area comprising the districts of Little Namaqualand, Kenhardt, Gordonia, and Cape of Good Hope. The commercial beryl deposits of Southern Rhodesia are at Bikita. Pegmatites in the Alto Molocue district of Quelimane Province yield most of the Mozambique production. The principal deposits in French Morocco are at Tazenakht, in the Anti-Atlas mountains. Madagascar is believed to have large deposits of beryl, especially at Fianarantsoa.

In the United States deposits of beryl occur in South Dakota, Colorado, New Hampshire, Maine, Connecticut, Massachusetts, New Mexico, North Carolina, Idaho, Nevada, Wyoming, Arizona, Utah, Georgia, Alabama, and California. Beryl has been produced principally from the Black Hills area in South Dakota as a coproduct in the mining of columbite-tantalite, spodumene, feldspar, or mica. This area has furnished about 39.4 percent of domestic production since 1949. New Mexico produced about 19.4 percent of the total during the same period; Colorado about 16.8 percent; New Hampshire about 14.1 percent; and Arizona, Connecticut, North Carolina, Georgia, Maine, and Massachusetts the remainder—10.3 percent.

RESERVES

United States reserves in deposits containing 1.0 percent or more beryl have been estimated at 15,000 tons, and reserves in deposits containing about 0.1 percent beryl have been estimated at 280,000 tons, 243,800 tons of which is believed to be in the pegmatites of the Kings Mountain district of the North Carolina tin-spodumene belt (*3*). It has been stated that the entire tin-spodumene belt extending from Gaffney, S. C., to Lincolnton, N. C., contains 823,000 tons of beryl (*6*).

Beryl reserves in foreign countries are not known with any degree of certainty. An estimate of world reserves has been calculated, however, assuming that the ratio between the United States reserves and world reserves is the same as the ratio that exists between United States production and world production. Using this factor, a world total of 190,000 tons of 1.0-percent ore and 3.5 million tons of 0.1-percent ore is obtained. Following the same reasoning, the reserves in 1.0-percent-beryl deposits are distributed as follows: Brazil, 97,000 tons; Argentina, 37,000 tons; India, 14,000 tons; Union of South Africa, 8,000 tons; and Australia, 6,000 tons. Those in 0.1-percent-beryl deposits are distributed as follows: Brazil, 1.8 million tons; Argentina, 680,000 tons; India, 260,000 tons; Union of South Africa, 150,000 tons; and Australia, 110,000 tons (*3*).

INDUSTRY

The Beryllium Corp. and Brush Beryllium Co. are the two largest consumers of beryl and manufacturers of beryllium alloys and compounds in the world.

The Beryllium Corp. of America was organized in Cleveland, Ohio, in 1927 and made the first significant quantity of beryllium-copper alloy in the United States at its Marysville, Mich., plant. The corporate name of the company was changed to The Beryllium Corp. of Pennsylvania in 1929 and to The Beryllium Corp. in 1945. The corporation produces beryllium-copper alloy (strip, rod, wire, bar, forgings, and safety tools), beryllium-copper master alloy (4 percent Be), beryllium-copper alloy (0.25 to 2.85 percent Be), beryllium-nickel alloy, beryllium-aluminum master alloy, beryllium oxide, and pure beryllium.

The Brush Beryllium Co., Cleveland, Ohio, was incorporated in 1931 and started to produce beryllium-copper alloy commercially at Lorain, Ohio, in 1935, continuing until the plant was destroyed by fire in 1948. Between 1949 and 1953 the company operated, for the Atomic Energy Commission, a surplus wartime magnesium-reduction plant converted for beryllium production. A new beryllium-copper master-alloy plant at Elmore, Ohio, was completed by the company in 1953, increasing United States production capacity 54 percent. The Brush Beryllium Co. produces pure beryllium and beryllium parts, beryllium-copper master alloy, beryllium-copper (0.25 to 3.00 percent Be), beryllium-copper rolling billets, beryllium oxide, beryllium ceramics, and beryllium-aluminum and beryllium-aluminum-magnesium master alloy.

Other beryllium companies and their products are:

Producer and plant location:	*Products*
Beryl Ores Co., Arvada, Colo.	Ground beryl; beryllium oxide; ceramic frit.
Ampco Metal Co., Milwaukee, Wis.	Beryllium-copper foundry products.
P. R. Mallory & Co., Indianapolis, Ind.	Do.

Producer and plant location—Continued	Products
Wilber B. Driver Co., Newark, N. J.	Beryllium-copper primary rolling-mill products.
Riverside Metal Co., Riverside, N. J.	Do.
American Silver Co., New York, N. Y.	Beryllium-copper reroll-mill products.
General Plate Division, Attleboro, Mass.	Do.
Penn Precision Products Co., Inc., Reading, Pa.	Do.
Little Falls Alloys, Inc., Patterson, N. J.	Beryllium-copper wire.

TABLE 2.—*Prices*[1] *of beryllium metal, alloys, oxide, and nitrate, November 1954*

Material	Unit	Price	Remarks
Beryllium-copper master alloy.	Pound of contained Be.	$40.00	
Beryllium-magnesium-aluminum (5 percent Be).	do	58.00	
Beryllium-aluminum (5 percent Be).	do	72.75	
Beryllium-nickel (2.7 percent Be and 1 percent Cr).	Pound of alloy	2.70	
Beryllium metal powder, standard mesh, premium grade	Pound	103.00	
Beryllium metal, lump or pebble:			
Technical grade	do	56.96–71.50	
Premium grade	do	85.00	
Beryllium oxide, high-fired, refractory-grade.	do	15.00 18.00	Depending on quantity.
Beryllium nitrate	do	4.95–5.95	Do.

[1] Prices quoted by Brush Beryllium Co.

TECHNOLOGY

Beryllium was discovered by a Frenchman, Vanquelin, in 1797 while making a chemical analysis of the mineral beryl. No useful application for the metal was known, however, before American and German scientists began investigating the properties of beryllium alloys after World War I. Since then the use of beryllium alloys has expanded constantly, being limited only by the high cost of beryllium and the supply of beryl.

Beryllium has atomic number 4; atomic weight, 9.02; specific gravity, 1.847; melting point, 1,284° C.; and boiling point 2,970° C. It is a grayish metal normally having large crystals and a bright, metallic luster (4). The physical properties of beryllium as listed in the literature differ considerably, largely because of the difficulties involved in preparing pure beryllium metal and making sound test specimens. Extreme hardness, often attributed to beryllium, is localized in a thin surface film of beryllium oxide that is hard enough to scratch glass. At ordinary temperatures the metal is quite brittle, possibly because small quantities of impurities are concentrated along the grain boundaries. Progress is being made, however, in preparing beryllium that is somewhat ductile at room temperature. At temperatures above 800° C. the metal is ductile enough to be swaged or rolled. It is claimed that small additions of titanium and zirconium increase its ductility at temperatures higher than 600° C.

Polished surfaces of beryllium metal in air keep their brilliance for years at normal temperatures. At higher temperatures, however, the solid metal oxidizes—noticeably at 700° C. and rapidly at 1,000° C. It is soluble in mineral acids and combines with the halogens. Beryllium is chemically related closely to aluminum. Complete separation of the two, therefore, is difficult.

The Copaux-Kawecki process (known as the fluoride process) and the Sawyer-Kjellgren process (known as the sulfate process) are generally used to convert beryl to beryllium oxide.

The fluoride process involves sintering briquets of a beryl-sodium ferric fluoride mixture; leaching the sintered briquets with water; precipitating beryllium hydroxide with caustic soda; filtering; and then igniting the precipitate to form beryllium oxide.

The sulfate process involves melting beryl at about 1,625° C. and then quenching the melt in cold water to obtain a glass; drying, grinding, and then leaching the glass with strong sulfuric acid; extracting beryllium and aluminum sulfates with water; and then precipitating most of the aluminum with ammonium sulfate. Beryllium hydroxide is precipitated from the remaining sulfate solution as a hydroxide and ignited to form beryllium oxide.

Preparation of the first metallic beryllium was accomplished with difficulty. Sir Humphrey Davy attempted unsuccessfully to reduce beryllium oxide with potassium vapor; later he used iron filings and obtained a metallic product, probably an iron-beryllium alloy. In 1828 Wöhler and Bussy independently prepared the first metallic beryllium by reducing the oxide with potassium. Pure metallic beryllium was prepared years later by Lebeau; he electrolyzed sodium-beryllium fluoride, using a nickel crucible as the cathode and graphite as the anode. Hugh S. Cooper produced the first sizable ingot of beryllium in the United States in 1916 and 2 years later patented the beryllium-aluminum basic alloy. In 1926 a metallurgist of the Electro Metallurgical Corp. discovered beryllium's ability to age-harden copper. Beryllium metal is now largely obtained by reducing beryllium fluoride with a stoichiometric deficiency of magnesium, a process patented in 1945 by Bengt R. F. Kjellegren. In this process, impure beryllium oxide is dissolved in an aqueous solution of acid ammonium fluoride, and the ammonium-beryllium fluoride formed is heated to expel the ammonia, leaving a residue of beryllium fluoride, which is reduced with a stoichiometric deficiency of magnesium in a graphite-lined furnace. A stoichiometric deficiency is the crux of the process, because the excess beryllium fluoride acts as a flux that permits reaction at lower temperatures. At 1,300 ° C. the beryllium floats on the fluid molten slag, from which is is easily separated after solidification. The excess beryllium fluoride is returned to the cycle.

HEALTH HAZARDS

Mists, dusts, and fumes of beryllium compounds may cause dermatitis on contact or acute pneumonitis if inhaled. Individual sensitivity varies greatly, and no way has been discovered to establish sensitivity before exposure. Safety equipment and procedures have been developed, however, which effectively reduce the hazard. For example, the Atomic Energy Commission[3] has found in its own operations, that when rigid industrial hygiene control methods are used that maintain a maximum average concentration in the atmosphere of 2 micrograms per cubic meter (maximum concentration at any 1 time of 25 micrograms per cubic meter), beryllium and beryllium compounds can be handled with apparent safety; and the Pennsylvania Health Department's Bureau of Industrial Hygiene 7-year survey of the Beryllium Corp. plant at Reading, Pa., has not disclosed one case of disease attributable to beryllium.

USES

Over 90 percent of the beryllium consumed is used as a hardening agent in alloys, mostly those of copper (*1*). Beryllium-copper alloys have commercial and strategic value because of their hardness, tensile strength, high electrical and thermal conductivity, low creep tendency, low-sparking and nonmagnetic properties, and resistance to fatigue, corrosion, and wear. The addition of 2.5 percent beryllium to copper produces an alloy which, after heat treatment, is 6 times as strong as pure copper. Beryllium-copper alloys, therefore, are used in parts subject to abnormal wear, such as tappet bushings, counterweight bushings, heavily loaded bushings where the action is eccentric or reciprocating, oil pressure-release-valve guides, spacing shims in airplane engines, and parts that mate with steel and are subjected to extreme vibrations or shock loading. Current-carrying springs of beryllium-copper, because they retain their elasticity, are used in aircraft instruments, pressure gages, electrical contacts and switches, and radio and radar devices. Beryllium-copper is used also in the manufacture of nonsparking tools, parachute-harness fasteners, and release springs.

Beryllium added to zinc with other elements improves its creep and corrosion resistance and increases its tensile strength to about that of cold-rolled 70-30 brass. Ticonium, a commercial alloy of beryllium, nickel, cobalt, chromium, and molybdenum (1–6 percent Be), is used to make surgical and dental instruments. Beryllium-nickel has a much higher tensile strength, improved grain refinement and ductility, and greater toughness than the copper-base alloy. It also has marked age-hardening characteristics. Beryllium-nickel is used for diamond-drill-bit matrix metal, watch-balance wheels, and certain airplane parts. Beryllium also is alloyed with platinum, iron, aluminum, gold, and magnesium.

Pure beryllium is used principally in X-ray-tube windows and with radium as a source of neutrons. It has numerous uses in the atomic-energy field as a moderator and reflector of neutrons. Small quantities of beryllium are used in melting and casting aluminum-magnesium alloys. Metallurgical research from 1939 to 1953 made possible the production of high-purity beryllium and the fabrication of large and intricate shapes. Beryllium has other uses that cannot be disclosed because of national-security regulations.

Beryllium oxide is outstanding as a high-temperature refractory because it is an excellent dielectric at both normal and high temperatures and is resistant to thermal shock (*12*). It is used in small, high-temperature, electric furnaces, in the fabrication of laboratory ware, and in certain glazes to reduce crazing. High-quality porcelains containing beryllium oxide are used for aircraft sparkplugs and ultra-high-frequency insulators. Other uses include its utilization as a component of special glasses, as a dehydrogenation catalyst, and in many applications that have not been made public because of trade-secret restrictions. Beryllium oxide has, and probably will have many more, uses in nuclear power plants because, in addition to its good refractory characteristics, it has a low neutron-absorption factor. Beryllium nitrate is used in combination with thorium nitrate in fabricating gas mantles. Beryllium chloride has proved useful in diagnosis of tuberculosis. Sodium-beryllium fluoride is used in manufacturing glass having high ultraviolet permeability, as a flux in certain porcelain enamels, and in coating special welding rods.

Beryl sometimes is used directly in the production of high-grade dielectrics, such as those used in airplane sparkplugs, and in ceramic bodies and glazes where the alumina and silica are not injurious.

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[3] According to Dr. Roy E. Albert, Division of Biology and Medicine.

SUBSTITUTES

The demand for beryllium alloys during World War II and the Korean War prompted a search for substitutes for beryllium and ways of conserving the metal. Substitutes were found that could replace most of the beryllium alloys; none, however, has all the desirable characteristics of the beryllium alloy that it might replace.

Beryllium-copper alloy was developed to replace phosphor bronze, aluminum bronze, and certain alloy steels; these materials, therefore, can in turn be used to some extent to substitute for beryllium-copper.

Phosphor bronze is a copper-base alloy containing as much as 10 percent tin; however, it is not age-hardening, its strength and electrical conductivity are only about half those of beryllium-copper alloys, and it has a lower yield strength. Phosphor bronze has been used to replace beryllium-copper alloy in current-carrying springs, diaphrams, and Bourdon tubes. The tin supply might limit use of phosphor bronze as a beryllium-copper alloy substitute.

Aluminum bronze is a copper alloy containing 10 percent aluminum and a trace of tin. It is a possible substitute for the beryllium-copper alloy used in manufacturing nonsparking safety tools and low-load bearings. The addition of nickel to aluminum bronze gives it age-hardening properties, thus improving its substitution possibilities for beryllium-copper alloys where strength is required.

Alloys of copper, nickel, aluminum, and silicon have been developed that might be substituted for the 1 percent beryllium-copper used in some electricity-conducting springs. However, if these alloys were to completely replace beryllium-copper in electricity-conducting springs, 250 tons of nickel annually would be required to make them.

WAR-ECONOMY SUPPLY-DEMAND PATTERNS

Beryllium was not used in large quantities until World War II, when the demand for beryllium products, particularly beryllium-copper alloys in airplane parts and instruments and many other war machines resulted in expansion of existing production facilities and greatly increased consumption of beryl ore.

During 1942–44, the Beryllium Corp. and Brush Beryllium Co. expanded production facilities under sponsorship of the War Production Board and the Defense Plant Corporation, and the Government bought foreign and domestic beryl ore. Supply did not equal the combined demand of industry and Government.

After the war, consumption decreased from 2,176 tons in 1944 to 1,738 in 1945; by the end of 1945 supply exceeded demand, and purchase for the National Stockpile was discontinued.

The situation changed in 1946, when it again became difficult to obtain enough ore to sustain the beryllium industry. Controls were placed on the export of beryllium November 27, 1946, and consumers relied for their principal source of beryl upon stock retained by the Office of Metals Reserve. Throughout 1947 and 1948 the demand continued to be greater than the supply, and in 1949 the stockpile had become so low that considerable concern was expressed over the possibility of its being exhausted by the end of the year if consumption remained at the 1948 level. The demand for beryllium-copper decreased during 1949, and late in the year all beryl stocks in the World War II stockpile were transferred to the Bureau of Federal Supply for inclusion in the Natioual Stockpile.

With the advent of the Korean War, consumption again rose, and the free-world supply of beryl and United States consumption and stockpiling were in delicate balance through 1950, 1951, and 1952.

The Defense Materials Procurement Agency proposed an interim Beryllium Ore Program on August 9, 1951, which was approved in final form May 19, 1952. It provided for: (1) A purchase plan with payment as high as $50 per unit for domestically produced concentrates, (2) loans, (3) geological investigations and process development, and (4) procurement from abroad.

To implement item 3, the DMPA entered an agreement with the Bureau of Mines in September 1952 that provided for the Bureau to investigate the possibility of recovering beryl concentrates of commercial grade from the pegmatites of the Kings Mountain area of North Carolina.

On October 7, 1952, the General Services Administration announced a program for purchasing beryl from small domestic producers at prices higher than those prevailing in the market. Purchase depots, at Spruce Pine (N. C.), Custer (S. Dak.), and Franklin (N. H.) were authorized to accept up to 25 tons of beryl a year from an individual producer. It was stated under the program that sales of larger quantities by single producers must be negotiated with DMPA, and producers wishing to participate in the beryl program must notify GSA not later than June 30, 1955. The program, under an order issued October 7, 1952, will terminate on June 30, 1957, or when 1,500 dry short tons of ore has been received.

To be accepted by the Government, ore must contain not less than 8 percent beryllium oxide by weight and must be in the form of clean crystals, cobbed free from waste. Shipments up to 500 pounds each are accepted or rejected on the basis of visual inspection. The Government will pay a flat price of $0.20 a pound or $400 a ton for ore accepted on the basis of visual inspection. The price of the ore subjected to chemical analysis will be predicated on the number of short-ton units of beryllium oxide contained in the ore. For ore containing 8.0–8.9 percent BeO, the price will be $40 per unit; for 9.0–9.9 percent BeO, $45 per unit; and for ore containing 10 percent or more BeO, $50 per unit.

EXPORT CONTROLS

In 1946 India imposed an export embargo on beryl; some shipments, however, have been sold to the United States Atomic Energy Commission as a result of Government-to-Government negotiations. Brazil prohibited beryl exports early in January 1951 but relaxed this restriction July 16, 1951, to allow limited exports; 4,000 tons was the quota set for 1953 and 1954.

France formerly required Madagascar and French Morocco to ship all of their production to the mother country to support its industry and to provide a stockpile for future atomic needs but now permits exports to the United States. Argentina has lifted the embargo it placed on beryl in 1948 and now allows restricted export. The Union of South Africa restricts export of beryl to 75 tons per month. This production, however, has not been attained.

OUTLOOK

Designers of precision machines have hesitated to specify beryllium alloys because of the uncertain supply, although their properties make them the best and most economical materials for a wide variety of uses. However, as a result of increased production facilities and reasonable assurance that the feast-famine pattern of supply will not be repeated, the outlook of the beryllium industry is for expanded utilization of beryllium and beryllium alloys.

Research is being conducted by the Federal Bureau of Mines to recover beryl from pegmatites. An inexpensive batch method of floating beryl has been developed in the laboratory, and investigation of continuous beryl flotation is progressing in a pilot plant at Rapid City, S. Dak. Results to date are encouraging. Industry and the Bureau are cooperating to find a process for recovering beryl as a byproduct from North Carolina spodumene ores. If this research is successful, commercial beryl reserves will be increased greatly.

The favorable outlook for an assured ore supply and the additional processing facilities constructed during the Korean War will result in keener competition and probably lower prices for beryllium products. Lower prices will put beryllium in more direct competition with other light metals for many applications where its favorable strength-weight ratio and high melting point make it preferable.

PROBLEMS

The major problems of the beryllium industry in the United States are: (1) To find ways to decrease the cost of its products, (2) to lessen dependence on foreign sources for ore supplies, (3) to develop a cheap and efficient method of recovering beryl from its ores, (4) to provide an adequate stockpile for national defense without disrupting the supply of raw materials to domestic industry, and (5) to prevent exposure to the toxic effects of beryllium compounds.

The relatively high price that the beryllium industry must now charge for its products hinders expansion of the use of beryllium and prevents its use in some applications.

More than 90 percent of the beryl-ore supply is imported, and dependence on foreign sources for ore subjects United States consumers to fluctuating raw-material supply resulting from conditions over which he has little influence.

Development of a cheap, efficient, mechanical process for concentrating beryl ore would lower the cost of beryl, prevent waste of reserves now incurred by hand cobbing, and make possible exploitation of the large reserve in low-grade deposits.

An adequate stockpile for national defense is needed to assure a supply for essential operations during wartime. It should be accumulated, if possible, without disrupting the flow of raw materials to domestic industry.

Prevention of exposure to the toxic effects of beryllium compounds is an ever-present problem of the industry.

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BISMUTH

By

Abbott Renick[1]

DURING the past 20 years bismuth, formerly used chiefly in pharmaceutical salts and in low-melting solders, has become an important industrial material through development of its alloys; today there are over 70 ways in which the alloys are usefully employed.

Summary

The United States is not self-sufficient in bismuth; much of the metal it consumes is imported or derived from imported raw materials. Peru and Mexico are its chief suppliers. The Western Hemisphere, as a whole, produces more bismuth than it consumes. Essential requirements for bismuth are now well provided for.

During 1949–53 the estimated annual world production of bismuth averaged about 1,800 short tons. United States annual consumption averaged about 775 short tons during the same period. United States imports of bismuth metal, excluding base bullion from Mexico, during 1949–53 averaged 314 short tons annually, while its exports, predominantly as metals and alloys shipped to Great Britain, averaged about 90 tons a year during the same period. The New York price of high-quality bismuth metal has varied between $1.80 a pound in 1946 and $2.25 a pound in 1950–53. The price has remained steady at $2.25 a pound since 1950.

In the United States native bismuth or bismuthinite ores have never been found of major importance. Minor occurrences of these ores have contributed relatively small tonnages of bismuth. The major domestic sources are the lead-silver ores of Utah and Colorado and the copper and lead ores of Montana, Utah, Nevada, Arizona, Idaho, and New Mexico. It is estimated that current indicated and inferred reserves of these ores would yield approximately 50 million pounds of bismuth. Based on present technological developments and economic conditions, the commercial ore reserve is considered to be about 30 to 50 million pounds.

The future level of domestic production will depend essentially upon the output of lead and, to a minor extent, of copper. The overall demand for bismuth in the United States may increase because new uses for metallic bismuth are rising and the long-range demand for bismuth at present prices may increase as much as 75 percent by 1975 to about 3.5 million pounds (*14*).[2]

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[1] Commodity-industry analyst, Bureau of Mines.

[2] Italicized numbers in parentheses refer to items in the bibliography at the end of this chapter.

BACKGROUND

HISTORY

Bismuth, as a metal, has been known since the Middle Ages. The origin of the name is uncertain, but the word probably came from the German Wismut. Agricola refers to it as a form of lead and describes a method of separation from its associated minerals by liquation. The impure bismuth of the early writers was often confused with antimony, tin, or zinc. During the 18th century, bismuth was identified as a metallic element.

OCCURRENCE AND SOURCES

Bismuth is an uncommon element. Clarke and Washington (*6*) estimate that it represents less than one-millionth part of the earth's crust, and Vogt places the ratio at less than 1 part in 10 million. Bismuth frequently occurs native; it and the sulfide, bismuthinite (Bi_2S_3), are its chief ore minerals. Two silicates, several sulfosalts, and the telluride, oxide, carbonate, molybdate, vanadate, and arsenate exist as relatively rare mineral species (*5*). Bismuth ochre or bismite ($Bi_2O_3 \cdot 3H_2O$) is the commonest oxidation product in deposits.

Bismuth minerals occur in veins in gneiss and other igneous rocks and also in sedimentary rocks, notably shales. They rarely occur except associated with other ores. Mellor groups bismuth deposits into three classes: (1) Bismuth-tin deposits, in which the bismuth is associated with tin and copper, with tungsten and molybdenum as accessories (for example, the Korean and Bolivian deposits, and those of Canada, Chile, and Peru). (2) The bismuth-cobalt deposits, in which the bismuth accompanies cobalt and uranium (for example, the deposits of Saxony). (3) The bismuth-gold deposits (for example, those in Queensland, New South Wales, Norway, and the United States). Bismuth occurs as an accessory constituent of many ores and minerals, especially with lead minerals.

Bismuth is obtained from two principal sources: (1) Metallurgical byproducts containing bismuth that are obtained from ores mined and treated primarily for the recovery of other metals; and (2) ores mined and treated chiefly for their content of bismuth and 1 or 2 other commonly associated metals, such as tin and tungsten. Only a few small deposits are mined for bismuth alone. The domestic supply is obtained as a refinery byproduct in the treatment of domestic and foreign lead bullion and nonferrous ores, mainly lead, silver, copper, and gold. The bismuth content of these ores is concentrated in certain metallurgical products, chiefly bismuth-lead bullion, from which bismuth is recovered in the refining process. The two major domestic producers are the American Smelting & Refining Co., at Omaha, Nebr., and Perth Amboy, N. J., and the USS Lead Refinery, Inc., at East Chicago, Ind., the refining subsidiary of the United States Smelting, Refining & Mining Co. The bismuth recovered at the Anaconda operations, Butte, Mont., is refined on a toll basis at Omaha, Nebr., by the American Smelting & Refining Co.

The principal foreign sources, in order of their importance, are the copper, lead, and silver deposits of the district in Peru operated by the Cerro de Pasco Corp., the lead deposits of Mexico, and the tin and tungsten deposits of Bolivia. A large tungsten mine in the Province of Kangwon, southern Korea, in operation the last 1½ years of World War II, had a byproduct yield of bismuth intermediate between that of Peru and Mexico. Minor supplies come from Argentina, Chile, Brazil, and Canada. Long-accumulated Canadian stocks of bismuth-bearing residues from refineries were treated during World War II to help meet the needs of the Allies.

The principal domestic sources are the lead-silver ores of Utah and Colorado and the copper, lead, and zinc ores of Montana, Utah, Nevada, Arizona, Idaho, and New Mexico. Some gold ores of Idaho, Montana, Nevada, Alaska, Colorado, and other States contain bismuth, but only a part of it is recovered. Bismuth also occurs in minor quantities in many different kinds of mineral deposits from which it is not ordinarily recovered. Some deposits containing from a few hundredths of 1 percent to several percent of recoverable bismuth do not constitute economic reserves of bismuth minerals, either because of sporadic distribution of the bismuth minerals and the low grade of the deposits or because of lack of valuable associate minerals.

RESERVES

The bismuth reserves in the United States available under present economic and technologic conditions can be roughly estimated from the average production life of the principal classes of deposits from which it is now recovered. Improvements in efficiency of recovery would increase the quantity of available bismuth somewhat but not the life of the reserves, since the output of bismuth is controlled by demands for its companion metals. Based upon present recovery and estimates of commercial base-metal reserves, the reserves of available bismuth metal are considered to be about 30 to 50 million pounds. The material available under improved economic conditions probably exceeds those quantities. The principal sources of such material are those gold veins from which bismuth is not recovered at present and submarginal base-metal deposits containing bismuth (*21*).

PRODUCTION AND REFINING[3]

In many respects, bismuth acts similarly to lead in the usual smelting operations employed in treating copper and lead ores. During the roasting of copper ores to eliminate sulfur, most of the lead and bismuth remain in the calcine, and only minor amounts go into the flue dust and fume. When the calcine is smelted to copper matte, the matte collects the major portion of the bismuth and lead. A small portion of bismuth and lead, however, is volatilized and goes into the dust and fume, and some is lost in the waste slags. During converter treatment of the copper matte to blister copper, the major portion of the lead and bismuth carried by the matte is fumed off during the iron-slagging stage of the converter operation. A minor portion is carried in the converter slag; but, since it is too rich in copper to be discarded, this slag is usually re-treated in the smelting furnaces, and some of the lead and bismuth in the converter slag are again collected in the copper matte. Copper smelters that treat ores containing lead and bismuth are usually equipped with baghouses or Cottrell precipitators to collect the dusts and fumes from various operations. These produces are, in general, sent to the lead-smelting plants for treatment. The bismuth in lead ores, concentrates, and flue dusts is collected in the lead bullion during lead-smelting operations. Bismuth is considered to be a detrimental impurity in refined lead and, if present in excessive amounts, must be removed before the lead can be used for many applications.

A number of methods have been employed for "debismuthizing" lead bullions. These are: Crystallization; electrolytic (Betts) refining; and the Betterton-Kroll process. The Sperry process for producing white lead uses lead containing bismuth, which is removed during the process.

The crystallization process takes advantage of the formation of lead-bismuth alloys of lower melting temperature (eutectic, 124° C.) than lead (melting point 327° C.). When lead bullion containing bismuth is slowly cooled, usually with gentle agitation, lead of low bismuth content tends to crystallize first, leaving a liquid portion of higher bismuth content. The crystallization is generally stopped when the proportion of crystals to liquid is about 2 : 1. The liquid is separated from the crystals by drainage. Sometimes the crystals are removed, as they are formed, by skimming. The remelted crystals will normally have about one-half the assay of bismuth, gold, and silver of the original lead, and the liquid portion will have about twice the assay of bismuth, gold, and silver of the original lead. A large number of crystallizations are required to produce a low-bismuth lead on the one hand and a very high bismuth lead on the other hand. The bismuth-rich lead (work lead) is sent to the silver cupel furnaces, where the lead and bismuth are slagged as litharge. The first litharge from the cupels carries very little bismuth; but, as cupellation proceeds, the litharge becomes progressively richer in bismuth. The crystallization process is seldom used in the Western Hemisphere for debismuthizing lead, but the method is sometimes employed as a step in concentrating or purifying crude bismuth to refined bismuth.

The electrolytic (Betts) process starts with lead bullion, which may carry tin, silver, gold, bismuth, copper, antimony, arsenic, selenium, tellurium, and other impurities but should contain 90 percent or more lead. If the bullion contains more than about 0.01 percent tin, the latter usually is removed from the bullion first by means of a tin-drossing operation. The lead bullion is cast as plates or anodes, and numerous anodes are set in parallel in each electrolytic cell. Between the anodes thin sheets of pure lead are hung from conductor bars to form the cathodes. Several cells are connected in series.

The electrolyte employed is a solution of lead fluosilicate and fluosilicic acid containing a small amount of glue or other suitable colloid or addition agent. Direct current, in passing through the cells, dissolves lead from the anodes and deposits it on the cathodes. The impurities in the lead anodes are insoluble under the conditions of normal cell operations and remain on the face of the anodes as a porous slime blanket. The finished cathodes are withdrawn from the cells, washed, and melted to refined lead. The scrap anodes are withdrawn, and the slime is washed free of soluble matter, either while still on the anode or after it has been removed by cleaning. The cleaned anode scrap is returned to the anode-casting kettle for recasting.

The washed slime is dried and melted to produce slag and metal. The slag is usually purified by selective reduction and smelted to

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[3] W. C. Smith, Cerro de Pasco Corp.

produce antimonial lead. The metal is treated in the molten state by selective oxidation to remove arsenic, antimony, and some of the lead. When free of arsenic and antimony, it is transferred to a cupel furnace, where the oxidation is continued until only the silver-gold alloy, known as "doré", remains. This doré is cast and treated to produce fine silver and fine gold. The bismuth-rich cupel slags are crushed, mixed with a small amount of sulfur, and reduced with carbon to a copper matte and impure bismuth metal, which is transferred to the bismuth-refining plant.

All gases from the various furnaces treating the slimes carry bismuth, silver, gold, and other values and pass through Cottrell precipitators, baghouses, or scrubbers for the recovery of these metals.

The Betterton-Kroll process starts with softened and desilverized lead that contains bismuth. The lead is heated in large steel kettles, and metallic calcium or calcium and magnesium gently stirred into the molten lead. The alkaline-earth metals form high-melting intermetallic compounds with bismuth, which separate as a floating dross. When the lead in the kettle is cooled to near its melting point, the dross is skimmed off. The residual calcium and magnesium in the lead are removed by treating the molten lead with chlorine gas or a mixture of molten chlorides, such as those of lead and zinc. It can then be cast as refined lead. Several additions of calcium and magnesium and drossings are usually necessary to obtain refined lead of low bismuth content equivalent to that produced by the electrolytic process.

The calcium-magnesium dross is liquated in a kettle to melt as much entrained lead as possible; the liquation residue is then melted in a reverberatory furnace, treated with chlorine or molten mixed chlorides to remove the calcium and magnesium, cast as a lead-bismuth bullion, and sent to the bismuth refinery for further treatment.

The Sperry process for manufacturing white lead begins with softened and desilverized lead, preferably containing some bismuth, cast in the form of anodes. These anodes are placed in the Sperry white-lead cells. Direct current is used to dissolve the lead from the anodes, with diaphragms separating the anolyte and the catholyte. Carbon dioxide is passed into the solution, white lead (basic lead carbonate) is produced, and no cathode deposit is formed.

The impurities in the anodes remain on the anodes as a "slime blanket." If the slime blanket falls off, the impurities will contaminate the white lead; on the other hand, if the slime blanket is allowed to become too thick, the electromagnetic force across the blanket will increase to a value high enough to force solution to metals other than lead from the face of the blanket. Thus it is desirable to have a slime blanket that adheres to the anodes and to remove it at short intervals. A bismuth-containing lead is preferred for the anodes, since the bismuth hardens the slime blanket and makes it adhere. The Sperry-process slime is washed, dried, and melted to an impure bismuth bullion, which goes to the bismuth refinery.

Recovery of bismuth from tin concentrates starts with leaching from roasted tin concentrates and other bismuth-bearing materials by means of hydrochloric acid. The acid leach liquor is clarified by settling or filtration, and the bismuth is precipitated as bismuth oxychloride when the liquors are diluted with large volumes of water. The impure bismuth oxychloride is usually redissolved in hydrochloric acid and reprecipitated by dilution several times. It is then dried, mixed with soda ash and carbon, and reduced to metal; or the wet bismuth oxychloride may be reduced to metal by means of iron or zinc in the presence of hydrochloric acid. The metallic bismuth produced by the oxychloride method requires additional refining.

The bismuth bullion or impure bismuth produced from lead by the various methods, as outlined in the preceding pages, is melted in kettles and treated with repeated additions of molten caustic soda to remove tellurium and arsenic. After cooling to the freezing temperature of the metal, any dross that forms is skimmed to remove copper. The metal is heated to a high temperature, and a calculated amount of metallic zinc is added. The charge is stirred until all the zinc has dissolved; it is then allowed to cool to about its freezing temperature, when the zinc dross or skims are removed. If the crude bismuth carries high silver and gold values, two additions of zinc and subsequent skimmings are employed. In this instance the second zinc skimming from a preceding lot plus new zinc is used for the first zincing. The first skimming, which is removed while the metal is fairly hot, contains most of the gold and silver and is treated for its values. The metal is reheated, and new zinc is added. The second skimming is done at a lower temperature, and the second product is held for the first zincing of the new lot.

The desilverized bismuth should now contain only bismuth, lead, and zinc. It is heated to about 500° C. and treated with chlorine first combined with zinc to form zinc chloride. After the zinc has been removed, the chlorine forms lead chloride. The chlorine treatment is continued until the lead assay has been reduced to about 0.03 percent. This point is indicated by a change in the appearance of the crystalline structure of a fractured test bar from fine-grain to coarse radiating crystals. At this point the chlorine treatment is stopped, the lead chloride slags are removed, and the metal is pumped into a clean kettle. Air is blown through the molten metal until all fuming has stopped, dross begins to form, and the surface of the metal shows a highly colored oxide film. The metal is again pumped to a clean kettle, covered with molten caustic soda, sampled, and cast as refined bismuth. The method of refining just described produces commercial bismuth 99.999 percent pure, with all impurities of the order of 0.001 percent or less.

Electrolytic refining processes for bismuth employing chloride electrolyte and fluosilicate electrolyte have been used but have not been able to produce as pure bismuth or to operate as cheaply as the pyrometallurgical process outlined above.

PROPERTIES

METAL

Bismuth is a metal of high luster having a distinct pinkish tinge. It is coarsely crystalline and very brittle. Its atomic number is 83 and atomic weight 209. Pure bismuth metal melts at 271° C. The specific gravity at 20° C. is 9.80, but in the molten state at 271° C. it is 10.067. It expands 3.32 percent during solidification. The Brinell-hardness number is 7.3.

The thermal conductivity of solid bismuth is less than that of any other metal except mercury, being 0.018 calorie per second per cubic centimeter at 100° C. The mean specific heat of the solid metal is 0.0319 calorie per gram, and the latent heat of fusion is 14.1 calorie per gram. The vapor pressure is 10.3 mm. of mercury at 540° C., 100 mm. at 1,200° C., and 760 mm. at 1,460° to 1,480° C. The boiling point rises with increased pressure, being 2,080° C. at 17 atmospheres. The coefficient of expansion of bismuth at 0° to 100° C. is 0.00000731. Its surface tension is lower than that of tin, cadmium, lead, and antimony.

The specific resistance of solid bismuth is 106.5 microhmcentimeters at 0° C. and 267 at 270° C. but becomes 147 microhms for liquid bismuth at 271° C. The electrical resistance increases with pressure and decreases with tension.

USES

METALLURGICAL

Bismuth improves the casting properties of tin and lead by lowering the surface tension, thereby producing sharply defined castings even when the percentage of bismuth (or antimony) is insufficient to cause the metal to expand and fill the mould at the moment of solidification. Bismuth also acts as a hardening agent, and its use has been recommended for this purpose in lead plates for storage batteries.

Extremely small amounts of metallic bismuth are added to molten iron, both gray and white, to produce cleaner castings. The addition of bismuth produces greater fluidity of the liquid. The amount used is about 0.1 to 0.2 percent of the weight of the molten castings.

Addition of bismuth to certain aluminum alloys is an established procedure, and the Aluminum Co. of America and others produce alloys that contain 0.5 percent bismuth and 0.5 percent lead for free-cutting screw machine stock. Small amounts of bismuth added to corrosion-resistant steels produce castings of greatly improved machinability.

ALLOYS

Bismuth is a component metal of a number of low-melting eutectic alloys that have melting temperatures lower than that of bismuth. Table 1 lists the eutectic alloys having melting temperatures below 200° C. (392° F.).

In addition to the eutectic alloys, hundreds of noneutectic alloys have been made, and many of them are employed to meet conditions that cannot be served by the eutectic alloys. The noneutectic alloys all exhibit melting or freezing ranges, and their "let-go" or yield temperatures vary with the method of testing, the shape and mass of the alloy, their location in the apparatus, the loading applied to the alloys, the rate of heat input, and other factors.

They are usually rated according to their "yield" temperature as determined by the method of testing established by the Compressed Gas Manufacturers Association (C. G. M. A. Safety Device Schedule). A number of those alloys have been given names, which, however, are not always used to specify the same composition. Table 2 lists some noneutectic alloys having yield temperatures below 300° C.

In 1929 Cerro de Pasco Corp. entered the field as a major producer of bismuth when supply greatly exceeded demand. The chief use for bismuth then was in pharmaceutical salts. The company initiated a program for developing new uses for the alloys that previously had been used in relatively small quantities for such devices as sprinkler links and vents on compressed-gas tanks, for dental castings, and in a few other applications. Today, after 20 years of research and development, with the aid of customers and foreign associates, over 70 useful industrial uses have been developed.

In general, the low-melting alloys that contain less than about 48 percent bismuth shrink during solidification, those containing 48 to 55

TABLE 1.—*Eutectic alloys* [1] [2]

Melting temperature		Composition, percent				
° F.	° C.	Bi	Pb	Sn	Cd	Others
390	198.8	----------	----------	91.00	----------	9.00 Zn.
362	183.3	----------	38.14	61.86	----------	----------
351	177.2	----------	----------	67.75	32.25	----------
291	143.9	60.00	----------	----------	40.00	----------
288	142.2	----------	30.60	51.20	18.20	----------
281	138.3	58.00	----------	42.00	----------	----------
255	123.9	55.50	44.50	----------	----------	----------
216.5	102.5	53.90	----------	25.90	20.20	----------
203	95	52.00	32.00	16.00	----------	----------
196.7	91.5	51.65	40.20	----------	8.15	----------
174	78.9	57.50	17.30	----------	----------	25.20 In.
158	70	50.00	26.70	13.30	10.00	----------
136	57.8	49.40	18.00	11.60	----------	21.00 In.
117	47.2	44.70	22.60	8.30	5.30	19.10 In.

[1] Mainly from The Encyclopedia of Chemical Technology, vol. 2, published by The Interscience Encyclopedia, Inc., New York (*9*).

[2] For any alloy, except alloys of eutectic composition, melting occurs over a range of temperatures. A eutectic alloy, like a pure metal, melts completely at a single temperature.

TABLE 2.—*Some noneutectic alloys* [1]

Yield temperature, ° C.	Freezing range, ° C.	Composition, percent				
		Bi	Pb	Sn	Cd	Others
271	349–261	56.00	----------	40.00	----------	4.00 Zn.
159	176–145	12.60	47.50	39.90	----------	----------
154	163–143	14.00	43.00	43.00	----------	----------
145	173–130	20.00	50.00	30.00	----------	----------
142	152–120	21.00	42.00	37.00	----------	----------
135	139–132	5.00	32.00	45.00	18.00	----------
127	130–124	56.00	2.00	40.70	.70	.40 In.
116	227–103	48.00	28.50	14.50	----------	9.00 Sb.
111	143–95	33.33	33.34	33.33	----------	----------
100	114–95	59.40	14.80	25.80	----------	----------
96	104–95	56.00	22.00	22.00	----------	----------
89.0	92–83	52.00	31.70	15.30	1.00	----------
72.5	90–70	42.50	37.70	11.30	8.50	----------
64	65–61	48.00	25.63	12.77	9.60	4.00 In.

[1] Mainly from The Encyclopedia of Chemical Technology, vol. 2, published by The Interscience Encyclopedia, Inc., New York (*9*).

percent bismuth exhibit little change of volume during solidification, and those containing over 55 percent bismuth expand during solidification. Alloys that contain bismuth and lead in ratios between 1 : 2 and 2 : 1, even when the alloys contain other metals, usually exhibit growth after solidification and on aging. Many low-melting alloys exhibit the usual thermal contraction during cooling from solidification to room temperature. Some shrink for the first few minutes after solidification, followed by rapid growth. Others shrink on cooling to room temperature, and the growth does not begin for 30 minutes to 3 hours after cooling to room temperature, and still others exhibit no shrinkage and grow rapidly while still warm. Approximately 90 percent of the growth occurs during the first 12 to 24 hours after casting, but some of the alloys continue to grow at reduced rates for 500 to 1,000 hours after casting. Rapid cooling or quenching of the alloys that exhibit growth tends to increase the rates of growth but does not affect final growth. The total growth varies from 0.0001 inch per inch to as much as 0.008 inch per inch and is governed by the composition of the alloys.

The fusible alloys range in hardness from about 5 to as high as 22 Brinell. Their tensile strength ranges from 3,000 to 13,000 p. s. i., and they have elongations of 0 to 300 percent, depending on the alloy composition. All the fusible alloys creep at room temperature under relatively light continuous loads, as do some lead alloys.

Fusible alloys	Percent				Melting point, °C.
	Bismuth	Lead	Tin	Cadmium	
Newton's metal	50	31.25	18.75		95
Rose's alloy	50	28	22		100
Darcet's alloy	50	25	25		93
Wood's alloy	50	24	14	12	71
Wood's metal	50	25	12.5	12.5	71
Lipowitz's alloy	50	27	13	10	70

In addition to use in fusible safety devices, some of the more important applications of bismuth alloys are:

Cerromatrix:[4] Anchoring punches in dies for stamping, blanking, etc., of sheet metal; securing bearings, bushings, and nonmoving parts in machinery; anchoring bushings in drill fixtures; and shimming locator pads in aircraft assembly fixtures.

Cerrobase: Master patterns for foundry matchplates and several practical foundry and pattern shop uses; liquid-metal heat-transfer medium in tempering baths and autoclaves; liquid seal in heat-treat and nitriding furnaces; fusible cores in electroforming, metal spinning, and forming fiber glass plastic shapes; glass-to-metal hermetic seal.

Cerrobend: Filler to support tubing during bending to prevent wrinkles; melts out in hot water; drop-hammer dies to form sheet metals; accurate spotting, checking, and inspection tools; heat-transfer medium, etc.

Cerrocast, Cerrotru: Soft-metal dies for patterns in "lost-wax" process; molds for casting and forming certain plastics; tracer models in duplicating machines.

Cerrosafe: Proof casting gun chambers and other cavities as blind holes, forging dies, etc.; supporting work pieces during machining and grinding operations (jet-engine blades), etc.

Bismuth alloys are sprayed from specially designed guns to protectively coat wood patterns and core boxes and thus prevent wear and shrinkage; to coat finished tools and metal parts as protection against rust or damage in transit (melts off in hot water); and for many other uses which limited space does not permit enumerating.

COMPOUNDS

One of the major uses of bismuth has been in the form of its compounds, which are used extensively in medicinal and cosmetic preparations, including various indigestion remedies and toilet powders. A number of these compounds are employed to treat wounds, and the metal therefore has some importance in time of war. Insoluble salts of bismuth are given to patients before X-ray examination of the digestive tract.

Bismuth compounds have also been used for other purposes than pharmaceutical and medicinal preparations. Organo-metallic compounds of bismuth have decided "antiknock" properties, although they are only about one-third as efficient as lead compounds in this respect. Bismuth trichloride is claimed to render some waterproofing materials, such as tar, creosote, and nondrying oils, less flammable. Bismuth trioxide has been used to a limited extent as a pigment and in vitreous enamels, procelain, and glass and chiefly as a flux to increase the fusibility of coloring oxides or compounds. In these uses it resembles lead oxide but has the advantage of being nonpoisonous.

The best known bismuth compounds contain trivalent bismuth. As a metallic element, bismuth forms a basic trioxide (Bi_2O_3) and a pentoxide (Bi_2O_5) that is considerably less acidic than antimony pentoxide. Bismuth trioxide forms salts with acids; the oxide is not strongly basic, hence the normal salts hydrolyze readily to form difficult soluble "basic" (hydroxy and oxy) salts. The addition of an excess of the corresponding acid dissolves the precipitates and reverses the reactions. Bismuth nitrate ($Bi(NO_3)_3$) forms a whole series of basic salts of varying basicity. A number of basic carbonates and sulfates are also known. Bismuth compounds, particularly the basic (or sub-) carbonate and nitrate and certain organic bismuth derivatives, are used in medicine, although they may cause toxic reactions in the human body. Other uses for some inorganic bismuth compounds are in cosmetic preparations and in ceramics.

Most organic compounds containing bismuth are bismuth salts or coordination complexes. Only a few true organobismuth compounds, with carbon to bismuth linkages, are known. Most of these are derivatives of the unstable trihydride bismuthine (BiH_3), analogous to stibine, arsine, and phosphine; the trisubstituted bismuthines (R_3Bi) are the most stable. A very few bismuthonic acids ($RBiO(OH)_2$) have been reported. Other types of bismuth compounds corresponding to the many organo types of the analogous element arsenic are virtually unknown.

Some important bismuth compounds are bismuth trioxide, bismuth hydroxide, bismuth trichloride, bismuth subcarbonate, bismuth subchloride, bismuth subiodide, bismuth nitrate, bismuth citrate, bismuth subsalicylate, bismuth subgallate, and bismuth sulfocarbonate.

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[4] *Cerromatrix* and other names in italics are registered trade names of Cerro de Pasco Corp.

Commercial products vary in carbon dioxide content. They yield on ignition not less than 90 percent Bi_2O_3, as specified by the U. S. Pharmacopoeia. There is a demand both for a bulky and a dense grade. Freshly precipitated bismuth subnitrate is the starting material for commercial preparation of subcarbonate. The subnitrate is suspended in distilled water and treated with sodium carbonate or a mixture of ammonia water and sodium bicarbonate. The conditions of operation, such as the volume and temperature of the water, as well as the composition and crystalline structure of the subnitrate, determine the bulk and appearance of the subcarbonate. The product is collected on large suction filters or separated in a centrifuge. It is washed thoroughly with cold distilled water and then dried at about 40° C. either at atmospheric pressure or under vacuum. When dry, it is milled and packaged in paper-lined barrels, paper cartons, or tin cans.

Prices of bismuth chemicals per pound, as quoted by the Oil, Paint and Drug Reporter, on September 6, 1954, were as follows:

Bismuth chloride__________jars__	$5. 11
Hydroxide__________dm__	4. 65
Metal, boxes__________ton lots__	2. 25
Nitrate, crystalline__________dm__	2. 10–2. 17
Oxide, anhydrous__________do____	4. 47–5. 05
Oxychloride__________do____	4. 37–4. 42
Phenolsulphonate, fibrous______do____	5. 22
Subcarbonate, U. S. P__________do____	3. 20
Subgallate, NF, fibrous__________do____	3. 15
Subiodide, fibrous__________do____	5. 37
Subnitrate, NF__________do____	2. 65
Subsalicylate, U. S. P__________do____	3. 50
Bismuth-ammonium citrate, U. S. P., powdered__________jars__	4. 22

SUPPLY—DISTRIBUTION

WORLD PRODUCTION AND CONSUMPTION

Estimated world production of bismuth in 1953 (2,100 short tons) was about 100 tons higher than in 1952 and exceeded the 1944–48 (average) (1,400 short tons) by 46 percent. Of the total output, North America supplied 46 percent, Europe 18 percent, South America 19 percent, Asia 16 percent, and other sources 1 percent. Table 3 shows the estimated world production of bismuth for 1948–53, by countries.

TABLE 3.—*Estimated world production of bismuth, 1948-53, by countries* [1]

[Pounds]

(Compiled by Pauline Roberts)

Country	1948	1949	1950	1951	1952	1953
North America:						
Canada (metal)[2]	239, 736	102, 696	191, 217	229, 814	162, 032	98, 613
Mexico (impure bars)	338, 800	680, 557	578, 600	743, 600	893, 352	803, 807
United States	(3)	(3)	(3)	(3)	(3)	(3)
South America:						
Argentina:						
Metal	(4)	(4)	(4)	(4)	1, 100	(4)
Ore	(4)	(4)	(4)	(4)	1, 100	(4)
Bolivia (in ore and bullion exported)	77, 312	18, 088	53, 775	151, 978	35, 046	138, 442
Peru:						
Metal	452, 894	474, 555	499, 072	577, 841	706, 416	630, 672
Lead-bismuth alloys	103, 895	5, 276	----------	----------	6, 921	
Europe:						
France (in ore)	123, 200	129, 800	171, 600	129, 800	127, 600	(4)
Spain (metal)	53, 392	43, 679	24, 957	33, 396	26, 987	(4)
Sweden	----------	----------	----------	----------	(4)	(4)
Yugoslavia (metal)	112, 420	83, 820	133, 168	193, 072	217, 140	216, 594
Asia:						
China (in ore)	(4)	11, 000	(4)	(4)	(4)	(4)
Japan (metal)	51, 319	57, 081	72, 708	92, 422	96, 800	110, 000
Korea, Republic of	228, 800	381, 524	(4)	27, 500	242, 000	528, 000
Africa:						
Belgian Congo (in ore)	1, 003	1, 188	1, 470	495	1, 034	(4)
South-West Africa (in ore)	----------	1, 100	15, 840	220	----------	----------
Uganda	8, 719	16, 542	8, 048	6, 371	6, 160	(4)
Union of South Africa (in ore)	961	11, 099	16, 828	7, 005	3, 384	2, 200
Oceania: Australia (in ore)	8, 941	1, 452	2, 011	2, 570	3, 128	1, 100
World total	3, 300, 000	3, 300, 000	3, 080, 000	3, 740, 000	3, 960, 000	4, 180, 000

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[1] Bismuth is believed to be produced also in Brazil, Germany, Rumania, U. S. S. R., and the United Kingdom. Production figures are not available for these countries, but estimates are included in total.

[2] Refined metal plus bismuth content of bullion exported.

[3] Production included in total. Bureau of Mines not at liberty to publish separately.

[4] Data not available. Estimate included in total.

Australia.—Bismuth ores are smelted and refined by Bismuth Products Pty., Ltd., Sydney.

Bolivia.—In 1953 bismuth exports totaled 63 metric tons of bismuth contained in concentrates compared with 16 tons in 1952.

Canada.—The Consolidated Mining & Smelting Co. of Canada, Ltd., Trail, B. C., continued during 1953 in its position as Canada's largest producer. Some shipments of bismuth oxychloride were made by the Molybdenite Corp. of Canada, Ltd., from its operations at La Corne, Quebec.

Korea.—In 1953 bismuth production at the San Dong mine totaled 240 metric tons of bismuth contained in concentrates.

Mexico.—Production of bismuth (metal content) in Mexico totaled 365 metric tons in 1953 and exports 261 tons. The United States was shipped 113 tons and the United Kingdom received 146 tons. The principal Mexican producers of bismuth are—

American Smelting & Refining Co., Apartado Postal 101, Monterrey, Nuevo Leon.
Compania Metalugia Penoles, S. A. (subsidiary of The American Metal Co., Ltd.), Apartado 251, Monterrey, Nuevo Leon.

The official value for calculating ad valorem export taxes, as of May 15, 1952, was 42.7534 pesos per kilogram (US$2.247 per pound) and for bismuth in impure bars 32.2195 pesos per kilogram (US$1.693 per pound).

Peru.—Peru is a substantial producer of bismuth. In 1953, Cerro de Pasco Corp. (home office, 40 Wall Street, New York, N. Y.) produced 287 metric tons of contained bismuth as byproducts of its copper and lead smelting operations at Oroya.

United Kingdom.—Bismuth ores and residues are smelted and refined by Capper Pass & Sons, Ltd., Bristol; Derby & Co., Ltd., London; and Mining & Chemical Products, Ltd., London.

United States.—Most of the bismuth produced is obtained as a byproduct from smelting domestic and foreign lead ores and by refining imported bismuth bars containing lead as a major impurity. The Bureau of Mines is not at liberty to divulge the quantities produced, but the 1953 output declined 3 percent compared with 1952.

Yugoslavia.—Bismuth metal output in Yugoslavia was 98 metric tons in 1953, virtually unchanged from 1952.

Free World Consumption.—The consumption of bismuth outside the United States is little known—better figures are needed. Table 4, though largely conjectural, sets forth the best estimates currently available.

TABLE 4.—*Estimated free world consumption in 1953*

	Thousand Pounds
Far East	100
Africa	100
Latin America	150
North America	1,750
Western Europe	1,500
Total	3,600

In 1953 domestic consumption of bismuth metal was 12 percent less than in 1952 and totaled 1,568,000 pounds. The bismuth content of pharmaceuticals was 414,200 pounds in 1953, virtually unchanged from the 406,800 pounds consumed in 1952, while the total quantity used by all other industries decreased 16 percent from the previous year. Table 5 gives consumption of bismuth in the United States.

PRICES

Bismuth is sold in 10- and 22- to 25-pound bars, the wholesale lot being 2,000 pounds. Bismuth bars are packed in cartons or wooden boxes protected with steel strappings. Wedges are often placed between the bars to prevent injury to the soft, brittle metal. There are no standard specifications, but the pharmaceutical and medicinal trade demands a purity of 99.99 percent, absolutely free from arsenic. The following is a typical analysis:

Silver	oz. per ton	½.
Lead	percent	0.001.
Copper	do	0.0009.
Iron	do	0.008.
Calcium		Trace.
Bismuth	percent	99.990.

The price of bismuth (pharmaceutical grade), formerly erratic, has been relatively stable in recent years, as shown in table 6.

TARIFF

The tariff on refined bismuth metal was reduced from 3¾ percent to 1⅞, effective October 1951. Paragraph 22, Tariff Act of 1930, provides that on chemical compounds and salts of bismuth the duty is 35 percent ad valorem. Bismuth alloys are dutiable at 22½ percent ad valorem. Bismuth ores are not dutiable.

OUTLOOK

CURRENT TRENDS

The immediate outlook is for the world output of bismuth to decline slightly below the 1953 level—4.2 million pounds.

There should be no difficulty in meeting domestic requirements during the next 3 or 4 years. During this period it is believed that the United States supply of bismuth metal will average about 1.8 million pounds a year. Im-

TABLE 5.—*Bismuth metal consumed in the United States, 1951–53, by uses*

Uses	1951 [1]		1952		1953	
	Pounds	Percent of total	Pounds	Percent of total	Pounds	Percent of total
Fusible alloys	186, 800	12	261, 700	15	191, 200	12
Solder	100, 200	6	145, 900	8	221, 000	14
Other alloys	513, 400	32	865, 800	49	613, 800	39
Selenium rectifiers	50, 400	3	25, 500	1	47, 500	3
Pharmaceuticals	569, 600	36	406, 800	23	414, 200	27
Other uses	171, 600	11	69, 300	4	80, 300	5
Total	1, 592, 000	100	1, 775, 000	100	1, 568, 000	100

[1] February-December only; compiled by National Production Authority, U. S. Department of Commerce.

TABLE 6.—*High and low prices of bismuth wholesale, pharmaceutical grade, New York, 1910–53*

[Dollars per pound]

Year	High	Low	Year	High	Low
1910	$2. 15	$1. 70	1932	$0. 85	
1911	2. 15	2. 10	1933	1. 30	0. 85
1912	2. 25	1. 80	1934	1. 30	1. 10
1913	2. 25	1. 75	1935	1. 10	1. 00
1914	3. 00	2. 75	1936	1. 00	
1915	3. 00	2. 75	1937	1. 00	
1916	4. 00	3. 25	1938	1. 05	1. 00
1917	3. 50	3. 35	1939	1. 25	1. 05
1918	3. 50	3. 35	1940	1. 25	
1919	3. 50	2. 65	1941	1. 25	
1920	2. 70	2. 40	1942	1. 25	
1921	2. 40	1. 50	1943	1. 25	
1922	2. 45	1. 50	1944	1. 25	
1923	2. 55	2. 45	1945	1. 25	
1924	2. 55	1. 50	1946	1. 80	1. 25
1925	2. 70	1. 30	1947	2. 00	1. 80
1926	3. 35	2. 70	1948	2. 00	
1927	2. 75	1. 85	1949	2. 00	
1928	2. 25	1. 70	1950	2. 25	2. 00
1929	1. 70		1951	2. 25	
1930	1. 70	1. 00	1952	2. 25	
1931	1. 50	1. 00	1953	2. 25	

ports of bismuth metal will continue to average about 700,000 pounds, and domestic consumption is estimated at 1.5 to 1.6 million pounds.

NEW USES

A recent potential development for bismuth is in nuclear power. A survey of solutions of uranium and thorium liquid metals and dispersions of uranium-thorium compounds in liquid metals has led to the discovery of several fuel and breeder systems useful in reactor design. A solution of uranium in bismuth and a dispersion of a uranium-tin compound (USn_3) in lead-bismuth-tin alloys have been proposed for two reactor designs, externally and internally cooled, respectively. A thorium-bismuth compound (Th_3Bi_5) dispersed in bismuth and lead-bismuth has been suggested for a breeder blanket in both reactor designs (*19*).

The United States Atomic Energy Commission now holds patents for 5 alloys running 35 to 40 percent bismuth, 15 to 40 lead, 15 to 35 tin, and 3 to 10 gallium. They have been patented because of their low melting points and low neutron capture qualities, which makes them suitable for coolants for atomic piles. The AEC also holds patents to 3 alloys containing 55 to 65 percent bismuth, 19 to 22 percent lead, 17 to 18 percent tin, and 3 to 5 percent thallium.

A new metallic alloy from which the world's "most-permanent" magnets can be made has been developed by scientists of the Naval Ordnance Laboratory, Washington, D. C. This material—a mixture of bismuth and manganese—has been found to have the highest coercive force of any known substance—a force of 3,000 oersteds compared with 2,600 oersteds for its closest rival; it also has high resistance to demagnetization. It is formed as a powder that can be molded into any shape desired. Its properties are such that no particular geometric form is required, such as a bar or horseshoe. The outlook for industrial application of this alloy depends largely on reducing production costs (*15*).

Another relatively new use for bismuth is in the dyeing industry. The Standfast molten-metal continuous dyeing process is a recent development that is expected to expand the use of bismuth. The Standfast process is a complete departure from the traditional methods in dye techniques. In this process, heat and pressure are applied by molten metal (bismuth) in which dye-impregnated fabric is immersed.

PROBLEMS

The bismuth industry has certain economic problems that spring from the byproduct relationship of bismuth to other metals, chiefly lead. As the raw-material supply cannot be greatly expanded (or readily controlled) in response to changes in demand, producers are concerned to maintain a reasonable degree of market stability in spite of large fluctuations in demand and stocks. Having developed a widely diversified industry that now consumes about the entire normal and available supply, no large new uses can be contemplated except at the expense of some established ones.

There appear to be no major and serious industry problems of the type that might be solved by scientific research. Cost-cutting improvements in production, refining, and methods of utilization are, of course, desirable.

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BITUMINOUS COAL

By

Staff, Division of Solid Fuels

BECAUSE of its importance as a source of energy and as a raw material, coal reaches directly or indirectly into almost every phase of daily life and national well-being. It has helped to develop the great iron and steel and electric power industries. Upon its production, distribution, and utilization have rested the employment, health, and prosperity of many millions of people. Coal still is the top supplier of energy for industrial purposes; it is by far the greatest reserve of energy, except possibly nuclear, and it is the backbone of industrial expansion in national emergencies.

Summary

In creating wealth, establishing high standards of living, and assuring continuity of industrial progress, energy has played a leading role. Of all fossil-fuel sources of energy, coal is the most abundant.

Starting as plant material in swampy areas, coal is formed gradually over millions of years while the plants decay into peat, which is buried and changed under the influence of pressure to the various ranks of coal. These ranks are not sharply divided but overlap because the starting material is similar for all coals. In order of increasing rank, coals are classified as lignitic, subbituminous, bituminous, and anthracitic. Coal, like wood and peat, contains carbon, hydrogen, oxygen, nitrogen, sulfur, and other elements in small quantities.

The United States has one-third of the earth's reserves of coal, according to geological estimates, and is the greatest producer of coal. Nearly 80 percent of the reserves of coal, on the basis of heating value per pound, is bituminous and subbituminous. These ranks of coal are distributed over all of the United States—about one-third in the East, one-third in the interior, and one-third in the Northern Great Plains, Rocky Mountains, and Gulf and Pacific areas.

Coal is extracted from the earth by underground and strip mining. Modern, mechanized-mining technology has resulted in higher productivity, greater safety for miners, and the minimizing of hard labor. After mining, coal is prepared to remove impurities and crushed to sizes suitable for its ultimate use. It may be burned domestically or to generate power, gasified for industrial or home uses, or coked for metallurgical purposes. The byproducts from coking—tar and light oils—can be used as such or processed further to serve as raw materials for the chemical industry.

The coal industry, widely dispersed over the Nation, reached its peak production of over 630 million tons in 1947. Between that time and 1953 production dropped 173 million tons, the number of mines from 9,400 to 6,600, and the number of men employed from more than 440,000 (in 1948) to fewer than 300,000. Part of this decline has been due to increased mechanization and coal technology, and much of it was caused by a change from coal to fluid fuels (gas and oil) by major fuel consumers. With continued growth of population and industry as well as development of new uses for coal, such as its conversion to liquid fuels, the demand for coal is expected to increase in the near future and substantially improve coal's position.

BACKGROUND

ORIGIN AND CLASSIFICATION OF COAL

Classifications of peat and coal are based on their nature and chemical composition, and these in turn depend on the kind of original plant material, its partial decay in peat swamps, and subsequent metamorphism by geological processes throughout the ages. The following description of the origin and nature of peat and coal is adapted from a Bureau of Mines publication entitled, "Facts About Coal".[1]

FORMATION OF PEAT AND COAL

Although coal is commonly thought of as a mineral, it differs from the usual minerals, such as stone and iron ore, because it is of organic origin, meaning that coal was formed from the remains of living things—trees, shrubs, herbs, and vines—that grew millions upon millions of years ago, during periods of mild, moist climate. During those periods there was heavy growth of trees, ferns, and other plants in swamps and bogs. Century after century the vegetation died and accumulated. Buried to a gradually increasing depth each year by new accumulations, the remains of roots, trunks, branches, and leaves changed gradually to peat, just as decaying vegetable matter is doing today in the Dismal Swamp of Virginia and North Carolina and in smaller swamps and bogs in many areas.

Peat is the first product when organic matter begins to change to coal. In a block of peat one can often see, with the naked eye, woody fragments of stems, roots, and bark. As peat is buried, it is cut off from the oxygen in air, and rapid decay of its organic matter is prevented by slowing bacterial action. The weight of more vegetation falling on the peat helps to compress and solidify it, as does the weight of water when the deposit sinks below a lake or sea, as has often happened. Sometimes mineral sediments have settled from muddy flood waters while vegetable matter was accumulating and formed "partings" or layers of shale in the coal vein. At the end of coal-forming periods, swamps remain flooded for a long time, and earthy sediments are deposited in thick beds over the peat, further compressing it and starting "coalification," the coal-making process.

Coalification is extremely slow when it depends mainly upon pressure generated by overlying rock. For this reason, many coal deposits are still of "low rank," that is, in an early stage of coalification, although they have been buried millions of years. A few deposits, however, are situated where profound movements of the earth's crust occurred during periods of mountain building. This rock movement generated much additional heat and pressure, producing "high-rank" coals, such as medium- and low-volatile bituminous coal and anthracite. Sometimes additional heat and pressure came from the movement of molten rock oozing from deep, hot regions of the earth into its outer crust.

RANKS OF COAL

The rank or degree of coalification of the higher rank coals is determined by "proximate analysis," a chemical analysis in which coal is decomposed into four constituents: (1) Water, called moisture; (2) mineral impurity, called ash, left when the coal is completely burned; (3) volatile matter, consisting of gases driven out when coal is heated to certain temperatures; and (4) fixed carbon, the cokelike residue that burns at higher temperatures after volatile matter has been driven off. For the lower rank coals heating value and caking and weathering properties determine rank. The rank of coal increases as the amount of fixed carbon increases and the amounts of moisture and volatile matter decrease. Moisture and volatile matter were squeezed and distilled from coal during its formation by pressure and heat, raising the proportion of fixed carbon. That this change is great will be seen from the following analyses, on an ash-free basis, of a typical lignite, the lowest rank of coal, and an anthracite, the highest:

	Lignite	Anthracite
Fixed carbon________percent__	33	92
Volatile matter_________do____	26	5
Moisture______________do____	41	3
Total___________do____	100	100

Table 1 lists the ranks of coal; the following description of coals by rank is given to show the overall picture. *Lignite* (including brown coal and lignite) ranges in appearance from

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[1] A selected bibliography appears at the end of this chapter (p. 134).

TABLE 1.—*Classification of coals by rank*[1]

[F. C.=fixed carbon; V. M.=volatile matter; B. t. u.=British thermal units]

Class	Group	Limits of fixed carbon or B. t. u., mineral-matter-free basis	Requisite physical properties
I. Anthracitic	1. Meta-anthracite	Dry F. C., 98 percent or more (dry V. M., 2 percent of less).	
	2. Anthracite	Dry F. C., 92 percent or more and less than 98 percent (dry V. M., 8 percent or less and more than 2 percent).	
	3. Semianthracite	Dry F. C., 86 percent or more and less than 92 percent (dry V. M., 14 percent or less and more than 8 percent).	Nonagglomerating.[2]
II. Bituminous[3]	1. Low-volatile bituminous coal.	Dry F. C., 78 percent or more and less than 86 percent (dry V. M., 22 percent or less and more than 14 percent).	
	2. Medium-volatile bituminous coal.	Dry F. C., 69 percent or more and less than 78 percent (dry V. M., 31 percent or less and more than 22 percent).	
	3. High-volatile A bituminous coal.	Dry F. C., less than 69 percent (dry V. M., more than 31 percent); and moist B. t. u.,[4] 14,000[5] or more.	
	4. High-volatile B bituminous coal.	Moist B. t. u.,[4] 13,000 or more and less than 14,000.[5]	
	5. High-volatile C bituminous coal.	Moist B. t. u.,[4] 11,000 or more and less than 13,000.[5]	Either agglomerating or nonweathering.[6]
III. Subbituminous	1. Subbituminous A coal.	do	Both weathering and nonagglomerating.
	2. Subbituminous B coal.	Moist B. t. u.,[4] 9,500 or more and less than 11,000.[5]	
	3. Subbituminous C coal.	Moist B. t. u.,[4] 8,300 or more and less than 9,500.[5]	
IV. Lignitic	1. Lignite	Moist B. t. u.,[4] less than 8,300[5]	Consolidated.
	2. Brown coal	do	Unconsolidated.

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[1] This classification does not include a few coals that have unusual physical and chemical properties and that come within the limits of fixed carbon or B. t. u. of the high-volatile bituminous and subbituminous ranks. All of these coals either contain less than 48 percent dry, mineral-matter-free fixed carbon or have more than 15,500 moist, mineral-matter-free B. t. u.

[2] If agglomerating, classify in low-volatile group of the bituminous class.

[3] It is recognized that there may be noncaking varieties in each group of the bituminous class.

[4] Moist B. t. u. refers to coal containing its natural bed moisture but not including visible water on the surface of the coal.

[5] Coals having 69 percent or more fixed carbon on the dry, mineral-matter-free basis shall be classified according to fixed carbon, regardless of B. t. u.

[6] There are three varieties of coal in the high-volatile C bituminous group, namely (1) agglomerating and nonweathering; (2) agglomerating and weathering; (3) nonagglomerating and nonweathering.

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brown to black. It yields a brown powder when ground or when rubbed over a rough, white surface. American lignite as mined contains 30 to 40 percent moisture. When exposed to air, it soon slacks or falls to pieces because of loss of moisture. Although there are large deposits of lignite in the United States, its production is not yet important nationally, partly because it is far from major markets. It is relatively difficult to store, and its heating value is low, making unprocessed lignite uneconomical to ship very far.

Subbituminous coal is black; as mined, it looks much like the bituminous coal commonly seen in coal yards. It contains 15 to 30 percent moisture. The range of heating values of subbituminous A coal is the same as that of high-volatile C bituminous coal. Subbituminous coal is entirely noncoking. Like lignite, it weathers upon exposure to air and is subject to spontaneous combustion if not stored properly. Like lignite, it is used principally in the areas where it is mined.

Bituminous coal is the most abundant and widespread rank of coal in the United States. It is the coal used most commonly for industrial, power, railroad, and heating purposes. Bituminous coal may be either coking or noncoking. This property is not based on the rank of the coal but rather on whether it will produce a coke when processed in a coke oven. Nearly all eastern bituminous coals have coking properties, but many of the western bituminous coals are noncoking or free burning.

A coking or caking coal is one that softens and flows when it becomes almost hot enough to take fire. As it is heated further, some degradation occurs, and volatile matter escapes as a gas. When the coal is heated red hot in a sealed oven where there is little or no air, a dull, gray, porous mass—coke—remains after most volatile liquids and gases have been driven off. Because coke consists largely of fixed carbon, coking is often called carbonization.

Noncoking bituminous coal may look like coking coal, and its composition may be similar, but it burns freely without pronounced swelling. Instead of a porous mass, it leaves a char or powdery residue. Noncoking and coking bituminous coals can be used interchangeably for many purposes, such as fuel for power plants, railroads, and heating and cooking. For some purposes, however, coking coals are required, the principal one being the manufacture of coke for smelting iron ore in blast furnaces. Noncoking coals are preferred for cement and tile burning, for which a high heating value and a high proportion of volatile matter are desirable.

Most bituminous coals appear black and lustrous in the pile; but close inspection shows a banded structure with alternate layers of bright, glossy coal and dull, grayish black coal. In some coals this dull material predominates.

Anthracite, sometimes called hard coal, has a brilliant luster and uniform texture. It will not soil objects as does coal of lower ranks. Anthracite has a higher percentage of fixed carbon and a lower percentage of volatile matter than the lower rank coals. It burns slowly, with a pale-blue flame free from smoke. Most anthracite has a somewhat lower heating value than the highest grade bituminous coals, but its lack of soot and the fact that it will burn longer without attention make anthracite an ideal domestic fuel. Most of it is now used for heating and cooking.

VARIETIES OF COAL

There are four varieties of bituminous and subbituminous coal—common-banded, splint, cannel, and boghead.

Common-banded coal, sometimes called bright coal, is the variety that includes most of the coals. It is composed of both anthraxylon and attritus in varying proportions and has a bright luster.

Anthraxylon, meaning "coal from wood," is the name given to the glossy, jet-black bands in a lump of coal. These bands were formed from the woody parts of plants—their stems or trunks, limbs, and branches.

Attritus refers to the finely divided matter of which the dull bands of coal are largely composed. This material is a mixture of plant debris consisting of the more resistant residues of various parts and organs of plants, such as cell walls, leaf tissue, spore coatings, pollen, and resin particles.

Splint coals are composed almost wholly of attritus, with only a little anthraxylon. The coal is dull, grayish black, hard, and tough. When mined, it breaks into block lumps; because of this characteristic, it is called "block coal" in some mining regions. Splint coal is good fuel for locomotives and steam power plants, and it makes a hot, cheerful fire in the fireplace. When heated, it tends to splinter, and it makes better coke when used in blends with banded coals.

Cannel coals are made up almost entirely of attritus. They are often rich in spores, the reproductive organs of the lower plants that do not produce seeds. In the original coal swamps, spores and seed cases of plants and particles of wood floated upon water until they became waterlogged, sank to the bottom, and finally changed into cannel coal. This coal usually contains a high percentage of volatile matter and burns with a long, yellow flame. It used to be called "candle" coal because of the appearance of the flame and because it can often be lighted with a match or a piece of burning paper. Cannel coal is used chiefly as a fireplace fuel; it was a desirable coal for making illuminating gas in an earlier era. It usually occurs in small lenses or deposits in beds of other coals and may be separated from them during mining.

Boghead coal is a variety of bituminous or subbituminous coal that looks and burns much like cannel coal. It is made up largely of the remains of algae—water plants that form the greenish or brownish scum often seen on fresh-water ponds and reservoirs in summer and fall. It is rich in volatile matter. When distilled, it yields a great deal of gas, tar, and oil. Like cannel coal, it is usually found in small deposits or lenses.

CHEMISTRY OF COAL

Coal, like wood and peat, contains carbon, hydrogen, oxygen, nitrogen, sulfur, and other elements in small quantities. The proportions in which the major elements are present differ greatly in different kinds of coal, and the chemistry of coal is quite complicated. For practical purposes, however, coal is analyzed by proximate analysis (see above) or sometimes by a more complete test—ultimate analysis—to determine the relative amounts of chemical elements.

Moisture, sulfur, and ash are the undesirable ingredients of coal. Volatile matter and fixed carbon are the most important contributors to the energy produced when coal is burned; and producing energy in the form of heat is the purpose of most uses of coal.

When a scientist wishes to find the heating value of a sample of coal, he weighs an exact quantity of it. He burns this completely in a bomb filled with compressed oxygen and immersed in a carefully measured quantity of water. He takes the temperature of the water at the start of the test and again after the burning coal has warmed it. He can then calculate the heating value of the coal, which is commonly expressed in British thermal units a pound. One British thermal unit (abbreviated B. t. u.) is the amount of heat needed to raise the temperature of 1 pound of water 1° F.

FUEL RESERVES

The United States is fortunate in being richly endowed with mineral-fuel resources. It has one-third of the geologically estimated coal and lignite reserves of the world, one-fourth of the proved petroleum reserves, and perhaps one-third of the world's proved natural-gas reserves. The United States also is the greatest producer of solid, liquid, and gaseous mineral fuels. In 1953 there were produced 23 percent of the world production of coal, 50 percent of all petroleum, and probably 90 percent of the natural gas.

To satisfy this large and still increasing demand for liquid fuel, more petroleum deposits must be discovered, more petroleum imported, or, from the standpoint of self-sufficiency, synthetic liquid fuels made from natural gas, coal, or oil shale. Each of these procedures probably will be employed to some degree, depending on economic and national defense conditions.

Additional domestic reserves of petroleum are being found at a rate almost adequate to supply the increasing domestic demand, although the cost of discovery is increasing and importation of oil has grown to 13 percent of consumption in 1953. Development of oil deposits in South America, Canada, and the Middle East is making more oil available for importation to the United States under peacetime conditions and for supplying the growing demand of other countries. The first industrial plant in the United States is beginning to manufacture chemicals and oils from natural gas at Brownsville, Tex. Bureau of Mines research at Rifle, Colo., has demonstrated that liquid fuels can be made from oil shale at comparatively small increases over prices prevailing in recent years for corresponding fuels from petroleum. Large reserves of coal and lignite can be converted to liquid fuels as prices of petroleum rise still further.

COMPARISON OF RESERVES OF MAJOR FUEL SOURCES

Ultimately, coal must be the primary source, not only of solid fuels, but of liquid and gaseous fuels as well, as will be seen by comparing coal reserves with those of petroleum, shale, and natural gas.

The Federal Geological Survey estimated the remaining coal reserves of the United States, not including Alaska, at 1,900 billion short tons as of January 1, 1953. These include beds down to 14 inches thick for anthracite and bituminous coal and down to 2.5 feet thick for subbituminous coal and lignite and at depths not exceeding 3,000 feet. The estimates are largely geologically inferred and do not allow for mining losses or consider that beds less than 2 feet thick are not economically minable by present underground methods. Investigations by the Bureau of Mines and others have shown that, on the average, about 50 percent of the coal is recovered in underground mining. Losses include coal left in pillars, in areas around oil and gas wells, under towns, railroads, roads, and streams, in river beds, in thin and impure beds, and in isolated areas not readily accessible for mining. For these reasons recoverable coal is considered to be one-half of the total estimated reserve.

As of December 31, 1953, committees of the American Petroleum Institute and of the American Gas Association estimated the proved reserves of crude oil at 28.94 billion barrels and natural-gas liquids at 5.44 billion barrels, making a total of 34.38 billion barrels of liquid hydrocarbons. The American Gas Association committee estimated the proved reserves of natural gas at 211 trillion cubic feet; the Federal Geological Survey estimated the recoverable oil from oil shales, averaging not less than 30 gallons oil content per ton, at 25 billion barrels, and shale deposits having an average content down to 15 gallons of oil per ton at 500 billion barrels.

These figures cannot be directly compared, because they are not calculated on the same basis. Only proved reserves are given for liquid and gaseous fuels. These are increased each year by new discoveries and extensions of fields and decreased by the year's production. More oil and gas reserves will be proved in the United States; in this respect, these fuels differ from coal and oil shale.

The solid-fuel reserves are based largely on geological observations after limited exploratory drilling, especially in the Western States. No infallibility is claimed for these estimates, but future exploration is not likely to add much to the solid-fuel reserves. Revisions will probably be downward as more data are obtained in the various coalfields.

Therefore, to compare reserves of fuels, one should express the figures not only in terms of energy content—British thermal units (B. t. u.)—but also indicate whether they refer to proved, measured, or indicated amounts of material recoverable by current methods. Proved reserves of oil and gas meet the last qualification. On the other hand, estimates of recoverable coal are long-term estimates, including considerable reserves inferred and not proved, as well as beds only 14 to 28 inches thick and those 2,000 to 3,000 feet below the surface and thus too expensive to mine at present. The Federal Geological Survey studied its recent reestimation of reserves in 10 States and concluded that one-fourth of the minable reserves would fairly represent the measured and indicated reserves of coal commercially recoverable by current production methods. This amount—237 billion tons—is one-eighth of the total reserves of 1,900 billion tons.

The Geological Survey also has cited estimates of authorities on the total inferred reserves of petroleum and natural gas for present and future use and has compared these with the minable reserves of coal (including measured, indicated, and inferred) without reference to the cost of mining. These two types of estimates are given in table 2 in terms of British thermal units.

Column 1 of this table, showing 5,669 quadrillion B. t. u. for all mineral fuels, gives the measured and indicated reserves of coal now commercially minable, proved reserves of petroleum and natural gas, and estimated reserves of oil from measured and indicated oil shale deposits averaging not less than 30 gallons of oil per ton.

Column 2, showing 24,377 quadrillion B. t. u., gives the minable (one-half of the total), measured, indicated, and *inferred* reserves of coal without any deduction for thin and deep beds, the unproved reserves of oil and gas estimated on the basis of favorable geological formations, and the estimated reserves of oil from indicated, measured, and *inferred* oil-shale deposits averaging not less than 15 gallons of oil per ton.

The long-term estimate of coal reserves (including inferred and difficulty accessible coal) is 4 times the short-term estimate, the long-term oil and gas estimates are about twice the short-term estimates, and the long-term estimate of oil for oil shale is 20 times the short-term estimate, principally because leaner and much more abundant shales are included. Obviously, coal is the dominant fuel in both estimates. It contributes 90 percent of the total energy in the short-term and 84 percent in the long-term estimate. If coal had been used to satisfy the demand for liquid and gaseous fuels in 1953, over 4 times as much coal would have been used as the 488 million tons produced that year. This example illustrates the tremendous demands that will be made on the reserves of coal, oil shale, fissionable minerals, and other sources of energy as gas and petroleum deposits become depleted. The life of these fuel reserves cannot be estimated because of the many variable factors involved. No one can accurately predict demands, new discoveries, and the effect of further technological developments. Such "guestimates" as have been made by competent scientists who have studied the question have ranged from 75 to 300 years for fossil-fuel reserves.

TABLE 2.—*Estimated recoverable mineral-fuel reserves of the United States as of Jan. 1, 1953*

Kind of fuel	Reserves, quadrillion B. t. u.		Percent of total fuel reserves	
	Proved and commercially recoverable [1] (1)	Total potential reserve (2)	Proved and commercially recoverable (3)	Total potential reserve (4)
Coal	5,117	20,469	90.3	84.0
Petroleum, including natural-gas liquids	198	423	3.5	1.7
Natural gas	200	487	3.5	2.0
Bitumen from bituminous sandstone	4	8		
Oil from oil shale	150	3,000	2.7	12.3
Total	5,669	24,377	100.0	100.0

[1] Measured and indicated coal reserves are considered proved.

COAL RESERVES

Table 3 lists the recoverable reserves of coal as of January 1, 1953. They total 950 billion tons and include bituminous 525, lignitic 232, subbituminous 186, and anthracitic coal 7; however, the average heating values of the several ranks of coal decrease from 14,000 B. t. u. for low-volatile bituminous coal to 6,700 B. t. u. for lignite. Therefore, these coals have also been compared, in terms of energy obtainable from them, by converting the tonnages given in column 1 to equivalent tons of bituminous coal of 13,000 B. t. u. per pound. Thus, a reserve of 232 billion tons of lignite equals 119 billion tons of 13,000-B. t. u. bituminous coal in heating value.

Tables 4 and 5 show the distribution, by rank and geographical location, of coal reserves on the basis of equivalent heating value. According to this method of evaluation, about 54 percent of the minable reserves of coal and lignite occur in the Eastern and Interior provinces and 46 percent in the Northern Great Plains, Rocky Mountain, Gulf, and Pacific provinces.

Eastern and western coals differ sharply in rank. Virtually all of the Eastern and Interior province coals, except anthracite, are caking bituminous coals. Coals of the Appalachian region are medium to strongly coking and in 1947 supplied 95 percent of the coal used for making coke. On the other hand, 70 percent of the coals of the Northern Great Plains, Rocky Mountain, Gulf, and Pacific provinces

TABLE 3.—*Potential recoverable reserves of various ranks of coal as of Jan. 1, 1953*

Rank of coal	Billion net tons (1)	Bituminous coal of 13,000 B. t. u. per pound calorific value, equivalent billion net tons (2)	Total coal reserves on B. t. u. basis, percent (3)	Heating values used in conversion, B. t. u (4)
Anthracite	7	7	1	12,700
Bituminous	525	545	64	13,500
Subbituminous	186	136	15	9,500
Lignitic	232	119	20	6,700
Total	950	807	100	

TABLE 4.—*Distribution of recoverable coal reserves in terms of B. t. u. value, by ranks and provinces*

[Weight basis]

Rank o fcoal	Equivalent 13,000 B. t. u. coal, billion net tons				
	Anthracite	Bituminous	Subbituminous	Lignitic	Total
Eastern	6.3	223			229.3
Interior	.1	206			206.1
Northern Great Plains, Rocky Mountain, Gulf, and Pacific	.4	116	136	119	372.4
Total	7	545	136	119	807.8

TABLE 5.—*Distribution of recoverable coal reserves in terms of B. t. u. value, by ranks and provinces*

[Percentage basis]

Rank of coal	Anthracite	Bituminous	Subbituminous	Lignitic	Total
Eastern	1	27			28
Interior		26			26
Northern Great Plains, Rocky Mountain, Gulf and Pacific		14	17	15	46
Total	1	67	17	15	100

are subbituminous and lignitic. These coals do not cake on heating.

WESTERN COKING COALS

Most bituminous coals in the Western States are free burning and noncaking. Some of them are slightly caking but do not become plastic enough to form coke strong enough for blast-furnace use.

Relatively small reserves of good coking coal occur in the Trinidad-Raton field of southern Colorado and northern New Mexico. This coal has been used for some years by the Colorado Fuel & Iron Co. at Pueblo, Colo., for making metallurgical coke. Bureau of Mines engineers estimate the inferred reserves of coking coal in beds 3 feet and more thick as 850 million tons for the Trinidad field and 600 million tons for the Raton field. These estimates are made on the basis of Federal Geological Survey and other data.

Another coking-coal reserve of probably greater extent but poorer coking properties occurs in the Book Cliffs field of eastern Utah and western Colorado. The Sunnyside beds in Utah are supplying the coke ovens at Provo and Geneva, Utah, and Fontana, Calif. This coal has a high oxygen content and approaches the limit of suitability for making metallurgical coke with adequate physical properties for blast-furnace use.

Some strongly coking, low-sulfur coals occur in Pierce and Kittitas Counties, Wash.; but, unfortunately, they are high in ash and difficult to mine owing to pitching and faulting of the beds.

TECHNOLOGY OF COAL INDUSTRY

Before coal reaches consumers, it has been mined and prepared; much of it will also have been converted to energy that may be used by individual consumers directly (for example, electricity) or for industrial purposes. The technology of mining has been mechanized to a large extent during the last 10 to 20 years, and preparation techniques had to be modified accordingly, as well as to meet changing patterns of utilization, as will be discussed below.

MINING

The mining of coal involves its disengagement, in suitable-size pieces, from the solid mass of a coal bed and transportation to the mine mouth or portal. This procedure involves preparing an access to the deposit and installing a transporting system for men, supplies, and mined coal.

Initially, in America, coal was recovered from outcrops of beds on hill or mountain slopes or on level lands. The operation was manual, the tools were picks and shovels, and conveyances were simple. As coal winning progressed into the earth, mine transportation and roof control entered coal-mining practice. Thus, the three primary objectives of modern mining technology originated: (1) High productivity in terms of tons per man-day or man-shift, (2) maximum safety for the miners, and (3) minimum need for hard labor.

Application of engineering, geologic, and physical sciences to attain these objectives has resulted in numerous technologic developments in mining. The contributions have not been exclusively American; many miners and mining techniques have come from Europe throughout the life of the American coal industry.

Methods for obtaining safety in mining have undergone much study and development. In some mines compressed carbon dioxide or air is used instead of explosives. To prevent a mine roof from caving in, props (wood, steel, or aluminum) may be used for its support; however, in the last 10 years the practice of roof bolting has risen phenomenally. Expansion bolts are wedged into holes drilled in the roof to bond the overlying rock and slate strata together to form a single, strong span. Ventilation in underground mining must be provided, not only to supply fresh air but particularly to remove explosive firedamp (methane) stored in the coal bed and released during mining. Its concentration must be checked at frequent intervals and, when necessary, ventilation must be increased or—in severe instances—men withdrawn from the mine until the danger has passed. Another hazard is introduced by coal dust, which may form an explosive mixture with air. Frequent removal of such dust, washing it down with water sprays, and diluting it with nonflammable rock dust help to solve that problem.

Constant vigilance and inspection, testing of equipment for use underground, and adoption of improved practices have greatly lowered the number of accidents in mines. The Bureau of Mines program of training mine inspectors and certifying explosives, shot-firing units, blasting devices, safety lamps, and diesel and electrical mobile equipment has contributed much to increasing safety in underground mines.

STRIP MINING

When coal occurs under a shallow layer of ground, strip or open-pit mining may be used. Where the coal deposit is thick, as at Kemmerer, Wyo., the open pit may be terraced down, as in the mining of massive copper ore bodies. In flat country, where the deposit is nearly horizontal and of normal thickness, the earthy cover (overburden) is moved to one side like a windrow with huge shovels. The coal is then scooped up by power shovels into cars or trucks. The next strip of overburden is moved into the trenchlike excavation and the sequence repeated. Depending on conditions, operations may follow the dip of the bed until the ratio of overburden to coal makes production uneconomic. In other instances, the initial cut is down the dip, and further mining proceeds sideways. A third form of strip mining—contour stripping—is used in hilly or rugged terrain where the outcrop runs around the hill or mountain. Again, economic factors determine the depth of cut and width of the strip.

The phenomenal growth of strip mining was made possible by development of bigger, more effective equipment. Some shovels now are over 40 cubic yards in bucket capacity. The earlier types of shovels were replaced by full-revolving shovels able to dig and dump in any direction. In turn came crawler or caterpillar traction and development of large dragline excavators and carryall scrapers. Steam and gasoline shovels have been decreasing while electric- and diesel-powered shovels have been increasing. Improvements have been made in power drills for shooting holes in overburden and in trucks for hauling the coal from strip pits to tipples or railroad ramps.

UNDERGROUND MINING

Underground mining is used when strip mining becomes uneconomical. Underground coal mines may be directly accessible from an outcrop, slope, or drift or may have to be reached through shafts—these are so-called shaft mines. These terms derive from the manner of entry and of moving coal to the surface. Shaft mining involves vertical entry for hoisting men, supplies, and coal. Mining methods in this case include room-and-pillar, longwall, and auger mining, and the "machine" method, in which all operators and machine controls remain outside. The last method is normally applied at a hillside outcrop.

The prevailing mining system in the United States is the traditional room-and-pillar, which has been frequently described and illustrated. In general, the entries into the coal measure serve as haulageways and fan out into the coal bed with side or cross entries from which the coal is removed to form rooms. Up to about 50 percent of the coal is left unbroken and uncut to serve as support for the roof—usually slate or rock that overlies the coal bed. To conserve resources and obtain the maximum coal output, pillars are sometimes removed or "robbed" on the retreat. In earlier times coal was drilled, blasted, and then manually loaded into cars for transportation to the tipple at the surface. Later, cutting machines were introduced for undercutting, overcutting, or making vertical cuts. Lateral cuts were often made for bench mining or keeping partings of slate separate.

In recent years coal mining has been highly mechanized, and loading and conveying are no longer manual. In fact, universal cutting machines are now mounted and have been developed into continuous mining machines with attached or integral conveyors that deliver cut coal to mine cars or belt conveyors that take it to the mine mouth. With the development of mechanized cutting and transportation, the use of explosives in coal mines has declined, and productivity has increased. Likewise, manual labor has declined, and skilled labor has increased.

Mining Machines.—The Joy continuous miner, mounted on rubber tires or caterpillar treads, has multiple cutters on a horizontal boom that make vertical cuts from floor to roof. Coal is mechanically swept to a chain conveyor and passed to the rear of the machine onto a shuttle car or a continuous conveyor.

The Colmol is another mounted cutting machine whose face holds several circular cutters that revolve vertically. The cuttings are thrown, by appropriate opposed directions of rotation (clockwise and counterclockwise) of the cutters, to the center of the machine front and are carried back on an integral conveyor.

Other mining machines currently in use in the United States are the Marietta and the Coal Master.

The Konnerth miner differs somewhat from these machines and basically consists of 2 horizontal undercutters, 2 shearing or vertical cutters, and coal-breaking hammers operable within the cuts. The hammers deliver 1,800 fifteen-ton blows per minute and are maneuverable vertically and laterally. The cutters move the broken coal into a hopper, and an integral chain conveyor transfers the coal to shuttle cars.

The above represent typical, fully mechanized machines used in room-and-pillar, modified room-and-pillar, and shortwall mining in the United States.

Longwall Mining.—In Nova Scotia, Canada, another machine, the Dosco continuous miner, is used for longwall mining. It is mounted, and the cutting head employs seven ripping-type cutter chains on a boom which sweeps from floor to roof in making a cut. The coal is carried back to a belt conveyor running transversely to the machine's direction of movement and discharging at the side into shuttles or other conveyors.

In the United States longwall mining was started with a coal planer or plow; several are now in service. This relatively simple machine consists of a heavy base extending the full length of the working face (328 to 500 feet long) and containing a coal conveyor; the plow is on the side of the base that faces the coal; it holds several cutter bits in a guide that point in opposite directions parallel to the coal face. In the guide is the pulling chain powered by electric motors at the extreme ends of the base. When power is applied, the planer rips off a thickness of about 4 inches of coal at the rate of 75 feet a minute. Coal falls onto the conveyor and is spilled continuously at the end of the machine onto other transporting systems. The long base is automatically kept tight against the face by air-operated piston jacks. Roof control in this case is temporary and effected by yielding props (50 tons per square inch) and collapsible cribs. The props are manually leap-frogged toward the receding working face. Behind the props the roof is allowed to break and fall. Due to voids, the broken roof occupies almost the same volume that it had originally plus that which the coal occupied, so that disturbance at the surface is small. The production of the coal planer is high.

Carbide Miner.—The carbide miner is a combination of multiple rotating cutters with a unitized conveying system in removing the coal. It is employed at and controlled from an outcrop portal. With the aid of an oscillograph (the "viewing" device), the cutting head can be controlled both laterally and vertically from outside the portal. Pillars of uncut coal support the cover. Coal recovery is about the same as with the auger.

A much simplified picture of coal-mine mechanization has been presented. Developments along that line, particularly rapid in the last decade, are still proceeding. Even today, although many mines are only partly mechanized, about five-sixths or more of all coal mined is mechanically loaded.

AUGER MINING

One of the recent developments in soft-coal recovery is auger mining. Many strip mines, particularly in the rough terrain of the Appalachian region, gradually encounter overburden so thick that continued stripping becomes economically impracticable. These strip mines have vast areas of coal seams exposed along what is known as the "high wall." Auger mining employs a large auger as a cutting head and a tube and screw for transporting coal. The mechanism is unitized so that the coal-removing system is extensible as the cutter penetrates the coal measure. The auger is generally operated from an outcrop location. The labor force is small, and recoveries of about 60 to 65 percent of the coal are reported. After much experimentation, huge augers were designed to bore horizontally into the exposed solid seam of coal much as a carpenter's auger bores into wood. These augers, 16 to 52 inches in diameter, are mounted on movable frames and bore holes 100 to over 200 feet deep. The coal falls from the auger into a conveyor and is elevated into a truck. This type of mining is generally known as high-wall auger mining.

MECHANIZATION

UNDERGROUND MINING

The first major task in underground mining—undercutting of the coal seam—was the first function to become mechanized, 25 percent having been cut by machines as early as 1900.

The percentage in 1953 was over 92. Similarly, hand drills have been replaced by power drills, the latter being used in 84 percent of total underground production by 1953. Mechanical loading has already been mentioned. Although some animals, mostly mules, still are used for hauling in small mines, they have been generally replaced by electric locomotives, rope haulage, belt conveyors, or shuttle cars.

STRIP AND AUGER MINING

Strip mining has a substantial advantage over underground mining: The output per man-day in strip mines has been approximately three times as large as in underground mines, resulting in a lower cost and price per ton of strip coal. For the past 10 years the f. o. b. mine value of strip coal was lower by 16 to 29 percent. In 1953, the average value f. o. b. mine for underground coal was $5.27 per ton and for strip coal $3.75.

Auger mining at soft-coal mines in the United States averaged 25 tons per man-day in 1953 compared with 18 at strip mines and 7 at underground mines. Under favorable operating conditions exceptionally high production was achieved, several auger mines in West Virginia averaging over 40 tons per man-day in 1953.

As a result of all these developments in mechanization, the quantity of bituminous coal mechanically cleaned in the United States increased from 28 million tons (5 percent of total production) in 1927 to 242 million tons (53 percent of total production) in 1953.

PREPARATION

In the early days of coal production, the preparation of coal constituted screening and hand picking the slate and rock, at the working face, from the coal. As time went on, crushing became an important factor in fitting coal to the needs of consumers. More recently, greater emphasis has been placed on upgrading coal quality by removal of extraneous material. To meet this requirement and sales competition, coal washing or cleaning grew in importance, and the operation became largely mechanized. At present, more than half of all coal mined in the United States is mechanically cleaned.

Cleaning consists of removing rock, other high-ash material, and sulfur-bearing components of coal from the carbonaceous matter. Sulfur occurs generally in two forms—inorganic matter such as pyrites (iron sulfide), and organic sulfur compounds that are an inherent part of the basic coal substance. The latter is not separable by mechanical cleaning, whereas part of the inorganic sulfur is separable.

Separation of impurities is based primarily on differences in density, the separable impurities being heavier than an equal volume of coal. To effect the separation, the natural force of gravity is employed in addition to other forces such as centrifugal and those introduced by agitation, jigging, flow of water, and the like. Various mediums are used to effect the separation, namely, water, chemical solutions, suspended solids in water, air flotation with special wetting agents, and even air alone.

COAL-CLEANING SYSTEMS

Many processes and a variety of equipment are used for cleaning coal, depending on the properties of the coal to be treated and the product desired. Classifiers employ water for washing and sizing. Many ingenious devices have been invented to draw off relatively uniformly sized and cleaned coals. A variant group of cleaners includes the launder types, in which flowing water attains the same objective. Another group is known as the jig type, in which a coal-water mixture is agitated by impulses, either piston-induced or mechanically applied. Further variation is found in the combination of water flow and a differential-motion table, a process also used for concentrating metallic ores.

In so-called dense-medium processes, the separation between coal and refuse is usually effected in a solution of calcium chloride or in liquids obtained by suspending solids in water. The suspended solids may be magnetite (fine iron oxide ore), sand, clay and barite, loess, or pulverized refuse from coal washing. A dry separation of coal from refuse can be achieved by substituting an air-sand mixture for the more conventional liquids.

For the very fine sizes of coal that have a large proportion of impurities and would otherwise be lost, flotation procedures are employed. In these, the mixture of coal and impurities is fed into a water bath, flotation agents are added, and the bath is aerated with small bubbles of air. The flotation agent (usually kerosine plus pine oil) wets only the coal particles, which are attached to air bubbles, lifted to the surface of the water, and skimmed off mechanically.

In the Humphreys spiral, the coal to be cleaned and water are passed down a spiral pipe, and the coal is separated from refuse by a combination of centrifugal and gravitational forces. This combination of forces has taken another form in the cyclone separator and thickener, in which the principle of the cyclone is employed as in dust, sawdust, and grain collectors with pneumatic systems.

In other systems air flows counter to the pull of gravity and separates coal and impurities. Again, suitable takeoffs are provided. This process is known as elutriation when applied to dusts and other fine materials for classification or segregation purposes; when applied to coal it is known as pneumatic cleaning.

Thermal and mechanical processes have been developed to dry the coal after washing. Calcium chloride either in a water solution or flake form may be applied to the coal to prevent freezing in transit.

MECHANICAL CRUSHING

Mechanical crushing is the process of running the coal through a heavy machine that crushes large lumps into smaller ones for easier handling and cleaning and for meeting market requirements. Before about 1930 the larger sizes were at a premium, and mechanical cleaning was little used. The only reason for crushing, therefore, was to reduce large lumps so that they would pass through the tipple. In subsequent years, however, with emphasis in customer demand on smaller sizes for use in mechanical stokers, crushing became essential for this reason as well as for mechanical cleaning.

TREATMENT FOR ALLAYING DUST

Although dust on coal may be negligible in comparison with the total weight, it creates a nuisance or even a hazard from explosion. Much experimentation has been carried on to develop surface processes for allaying coal dust (as distinguished from dust problems in underground coal mines). Such treatment also prevents freezing of wet coal, reduces windage loss of fines in transit, improves burning performance at times, reduces the tendency toward spontaneous heating during storage, and reduces air slacking and disintegration.

The materials used for such surface treatment are (1) water in conjunction with wetting agents (organic chemical compounds that cause water to spread into a thin film and wet the coal surface readily), (2) calcium chloride with or without various admixtures, and (3) oil or other petroleum products. The predominant treating agent has been oil, except during World War II, when its use was restricted.

TRANSPORTATION

Almost 80 percent of all coal leaves bituminous mines via railroads, at rates that currently add nearly 70 percent to the coal cost f. o. b. mines. Oil and gas have an advantage over coal, because the use of pipelines reduces the transportation cost.

Railroads have been the principal means for shipping soft coal since their development. In recent years this method of transporting coal has declined 10 percent owing largely to increased movement by water and truck from the mines to final destinations. Furthermore, rail shipment is not necessarily uninterrupted. Approximately 40 to 50 million tons of coal is dumped annually at Atlantic coast tidewater piers and transported by water to ports in the United States or exported overseas. In addition, 40 to 50 million tons is dumped at lower Lake Erie and Lake Ontario ports and transported by water to the Northwestern States and Canada. Some of this coal is transhipped by rail or truck to final destination.

Motor trucks for transporting coal appeared immediately after World War I but were not an outstanding factor until the late 1920's. By the middle 1930's trucks had become an important means of transporting coal from mines not only to final destination but also to tipples, railroad sidings, or waterways for further shipment. The development of bigger and better trucks and improved highways has stimulated the growth of truck shipments to final destinations. As railroad freight rates for coal are not strictly proportional to distance and as freight for short hauls is higher than for long ones, trucks are used increasingly for short-haul traffic.

Although some of the earliest shipments of soft coal were made by water, the major movement by river awaited the growth of large consumers on water, particularly the steel industry on the Monongahela River near Pittsburgh. Although in 1953 soft coal was shipped from mines by way of nine rivers, the largest movement was on the Monongahela River. The proportion shipped from mines by water, including the tonnage trucked to water, has been growing from 4 percent of total production in 1933 to 8 percent in 1953. Part of this growth represents increased shipments on rivers other than the Monongahela. Like rail shipment, water shipment is not necessarily to a final destination. A considerable tonnage, particularly in the Cincinnati area, is transhipped from the Ohio River to final destination by railroad or truck.

The most recent development in shipping soft coal is trucking of coal from mines to railroad sidings or to waterways for further shipment; such movement has increased from 1 percent in 1940 to 6 percent of the total production in 1953. During the same period the average distance of haul doubled, increasing from 4 miles to 8. This relatively new use of trucks made possible the opening of many small mines and even of larger strip mines.

Because of the importance of the cost of transportation, experiments are in progress on pipeline transmission of coal in water.

UTILIZATION

In this short account only limited and general statements relating to the technology of coal usage can be made. The basic technology of the past and present includes combustion, gasification, and cokemaking, all of which overlap.

COMBUSTION

Coal is burned for space heating, whether heat distribution is by warm air, hot water, steam, or electricity. It is burned to supply hot water and steam for many purposes. One of the most important of these numerous uses is the generation of electric power for lighting, the motive force for manufacture and transportation, communication systems, air conditioning, domestic appliances, and the like. The combustion of coal and its control and application for technical purposes are being intensively studied and developed. In fact, burning of coal has, to a large degree, become a science. Pulverized coal can now be moved and handled like a fluid (analogous to oil and gas). Combustion has developed from simple burning in rudimentary grates, stoves, and furnaces to a highly complex operation, instrumented for efficiency and automatically controlled. Progress in efficiency is well illustrated by the fact that not too many years ago (1920) about 3 pounds of coal was consumed in the United States for 1 kilowatt-hour of electricity, whereas in 1953 the average in the United States was slightly more than 1 pound, and some of the most efficient plants need only 0.7 pound or less.

GASIFICATION

Gasification of coal, which has had an important role in American industry, is likewise undergoing many changes and developments and has taken several forms. Limited and controlled quantities of air and steam passed through ignited coal yield producer gas (whose combustible constituents consist mainly of carbon monoxide and hydrogen with limited heating value), which finds extensive use in industry for operations requiring heat, such as coking and ceramic firing.

Coal, when coked, produces a valuable gas (about 10 to 11,000 cubic feet per ton of coal) of good heating quality (about 550 B. t. u. per cubic foot) with a composition that makes its utilization as raw material for synthetic chemical production feasible. Such production is now beginning in the United States. This gas has been used over the years as a domestic gas and for industrial purposes.

Coal, usually as coke, has been the source of much water gas, a mixture of mainly hydrogen and carbon monoxide. This gas is produced by the action of steam on incandescent coke and has a heating value of about 300 B. t. u. per cubic foot. For many years, enriched in heating value to 500 to 550 B. t. u. by gases from cracked oil, carburetted water gas has served as domestic gas. As the ratio of hydrogen to carbon monoxide is adjustable by chemical reaction, water gas can provide hydrogen for ammonia synthesis, for hydrogenation of oils to make margarine and cooking fat, and to convert coal to chemicals and liquid fuels. Water gas also serves as raw material for the synthesis of motor fuels, methanol, higher alcohols, and other chemicals.

The advent of natural gas (mainly methane) and its distribution in volume in recent years have caused changes in the production and use of gas from coal in almost all fields, whether its use be as a fuel or as a chemical raw material. Methods are now being studied (and used to some extent abroad) for converting coal to methane.

COKING

Large amounts of coal are coked for metallurgical use in blast furnaces and foundry cupolas for the production of pig iron and castings of iron, steel, and other metals. Coal tar and light oils, produced during such carbonization, serve (1) as raw materials for organic chemicals like synthetic rubber, detergents, plastics, photographic materials, flavor and perfume chemicals, explosives, and medicinals, and (2) after comparatively slight processing, as wood preservatives, disinfectants, animal dips, and road and building materials.

LOW-TEMPERATURE CARBONIZATION

Low-temperature carbonization (at 900° to 1,000° F. compared with about 2,000° F. for high-temperature or coke-oven carbonization) is a little-practiced but promising use of coal for producing smokeless fuel, industrial carbons, and substantial quantities of low-temperature tar and oils. The key to widespread practice of this type is development of markets for these tars and oils. When economical operation of low-temperature carbonization is achieved, it should result in cleaner urban atmospheres, as well as expansion of coal consumption.

STATISTICAL DATA ON THE COAL INDUSTRY

GEOGRAPHICAL DISTRIBUTION

The coal industry is widely dispersed over the Nation. Commercial quantities of coal are produced in 25 States, from Pennsylvania to Washington, from Illinois to Alabama, and in Alaska. The volume of production, however, is not proportional to the volume of reserves. The reasons for these variations include differences (1) in rank or quality and (2) in costs of production (due principally to geologic conditions), (3) distances from principal markets

TABLE 6.—*Maximum and total production of bituminous coal and lignite in the United States, by States*

[Thousand net tons]

State	Maximum production		Total production from earliest record to end of 1953
	Year	Quantity	
Alabama	1926	21,001	874,212
Arkansas	1907	2,670	95,300
Colorado	1917	12,483	486,681
Georgia	1903	416	[1]
Illinois	1918	89,291	3,333,574
Indiana	1918	30,679	1,056,858
Iowa	1917	8,966	344,023
Kansas	1918	7,562	273,881
Kentucky	1947	84,251	2,240,415
Maryland	1907	5,533	262,180
Michigan	1907	2,036	46,391
Missouri	1917	5,671	269,048
Montana (bituminous and lignite)	1944	4,844	166,071
New Mexico	1918	4,023	124,224
North Carolina	1922	79	[1]
North Dakota (lignite)	1950	3,261	79,912
Ohio	1920	45,878	1,844,338
Oklahoma	1920	4,849	168,038
Pennsylvania	1918	178,551	7,762,337
Tennessee	1942	8,158	345,515
Texas (bituminous and lignite)	1913	2,429	60,929
Utah	1947	7,429	223,356
Virginia	1952	21,579	633,888
Washington	1918	4,082	145,714
West Virginia	1947	176,157	5,544,987
Wyoming	1945	9,847	387,572
Other States [2]			68,668
Total	1947	630,624	26,838,112

[1] Included under "Other States."

[2] Includes Alaska.

that affect transportation costs, and (4) competition with other sources of energy.

Over the years there have been wide variations in production in the several States. For instance, total production in the United States reached its peak in 1947 (see table 6). Maryland, Arkansas, and Michigan all reached their maximum production in 1907. Nine States, including Illinois and Pennsylvania, reached their peak during World War I. In contrast, Kentucky, West Virginia, and Utah attained their maximum output in 1947. Ranked in terms of production in 1953, West Virginia was first, followed by Pennsylvania, Kentucky, and Illinois, in the order listed.

There are wide variations among regions in production, number of mines, uses of coal, average value per ton, men working, days worked, and output per man-day. Most of these variations are explainable in such terms as extent of mechanization, quality of product, transportation, and market demands.

Only 6 States have produced over 1 billion tons from earliest records to the end of 1953: Pennsylvania (7.8 billion), West Virginia (5.5), Illinois (3.3), Kentucky (2.2), Ohio (1.8), and Indiana (1.1).

PRODUCTION

Through the turn of the century, the bituminous-coal industry grew rapidly, almost doubling its production and capacity every 10 years (see table 7). From 1890 to 1910 production increased from 111 million to 417 million tons; it reached a war-time peak of 579 million tons in 1918. During these heydays the number of mines and employees increased heavily—from 2,500 mines and 240,000 employees in 1895 to 9,300 mines and 705,000 employees in 1923. The value of production, f. o. b. mines, increased from 116 million dollars in 1895 to over 2 billion dollars in 1920.

From 1920 to the early 1930's, however, the general trend of production was downward—to a low of approximately 310 million tons in 1932. The number of mines and employees dropped to 5,400 and 406,000, respectively, and value f. o. b. mines dropped to 407 million dollars. The steady, normal growth of the industry had come to a halt and given way to excess capacity, which resulted in a severe depression for coal.

Not until 1940 did the industry as a whole come out of the depression, following the passage of the Bituminous Coal Act of 1937 and the beginning of heavy war requirements. Influenced by the latter, production increased to a wartime peak of 620 million tons. To meet the heavy demands for energy during the postwar rehabilitation period, including the highest demands in history for exportation of United States coal, production climbed to its alltime peak of more than 630 million tons in 1947. Value f. o. b. mines in 1948 reached almost 3 billion dollars, and the number of mines in 1950 exceeded 9,400. From these pinnacles, however, the drop has been very severe, even disastrous for many mining companies and miners. Output in 1953 fell to 457 million tons, the lowest since 1939; the number of mines to 6,600; and value f. o. b. mines to 2.2 billion dollars. Employment declined to 293,000. Furthermore, production in 1954 is estimated to have been only 390 to 395 million tons; the number of mines under 6,000; value f. o. b. mines around 2 billion dollars; and employment approximately 237,000, which would be the lowest since 1893.

Although the nature of the industry is such that a certain degree of overcapacity is both desired and essential, present troubles stem from a combination of too much capacity and growing competition from other sources of energy—namely, oil and natural gas.

Although an excess of coal-producing capacity is a bulwark of safety in emergencies, at other times it is a depressing factor in operations of the industry. On the basis of a 280-day work-year, capacity in 1923 was 885 million tons (565 million produced); in 1932, 594 million (310 million); in 1944, 624 million (620 million); and in 1953, 670 million (457 million). In 1944

TABLE 7.—*Growth of the bituminous-coal- and lignite-mining industry in the United States and Alaska, 1915–53*

Year	Production (net tons)	Average value per ton	Number of mines	Men employed	Net tons per man-day	Percentage of underground production mechanically loaded	Percentage of total production—	
							Mechanically cleaned	Mined by stripping
1915	442, 624, 426	$1. 13	5, 502	557, 456	3. 91	----------	4. 7	0. 6
1916	502, 519, 682	1. 32	5, 726	561, 102	3. 90	----------	4. 6	. 8
1917	551, 790, 563	2. 26	6, 939	603, 143	3. 77	----------	4. 6	1. 0
1918	579, 385, 820	2. 58	8, 319	615, 305	3. 78	----------	3. 8	1. 4
1919	465, 860, 058	2. 49	8, 994	621, 998	3. 84	----------	3. 6	1. 2
1920	568, 666, 683	3. 75	8, 921	639, 547	4. 00	----------	3. 3	1. 5
1921	415, 921, 950	2. 89	8, 038	663, 754	4. 20	----------	3. 4	1. 2
1922	422, 268, 099	3. 02	9, 299	687, 958	4. 28	----------	----------	2. 4
1923	564, 564, 662	2. 68	9, 331	704, 793	4. 47	0. 3	3. 8	2. 1
1924	483, 686, 538	2. 20	7, 586	619, 604	4. 56	. 7	----------	2. 8
1925	520, 052, 741	2. 04	7, 144	588, 493	4. 52	1. 2	----------	3. 2
1926	573, 366, 985	2. 06	7, 177	593, 647	4. 50	1. 9	----------	3. 0
1927	517, 763, 352	1. 99	7, 011	593, 918	4. 55	3. 3	5. 3	3. 6
1928	500, 744, 970	1. 86	6, 450	522, 150	4. 73	4. 5	5. 7	4. 0
1929	534, 988, 593	1. 78	6, 057	502, 993	4. 85	7. 4	6. 9	3. 8
1930	467, 526, 299	1. 70	5, 891	493, 202	5. 06	10. 5	8. 3	4. 3
1931	382, 089, 396	1. 54	5, 642	450, 213	5. 30	13. 1	9. 5	5. 0
1932	309, 709, 872	1. 31	5, 427	406, 380	5. 22	12. 3	9. 8	6. 3
1933	333, 630, 533	1. 34	5, 555	418, 703	4. 78	12. 0	10. 4	5. 5
1934	359, 368, 022	1. 75	6, 258	458, 011	4. 40	12. 2	11. 1	5. 8
1935	372, 373, 122	1. 77	6, 315	462, 403	4. 50	13. 5	12. 2	6. 4
1936	439, 087, 903	1. 76	6, 875	477, 204	4. 62	16. 3	13. 9	6. 4
1937	445, 531, 449	1. 94	6, 548	491, 864	4. 69	20. 2	14. 6	7. 1
1938	348, 544, 764	1. 95	5, 777	441, 333	4. 89	26. 7	18. 2	8. 7
1939	394, 855, 325	1. 84	5, 820	421, 788	5. 25	31. 0	20. 1	9. 6
1940	460, 771, 500	1. 91	6, 324	439, 075	5. 19	35. 4	22. 2	9. 2
1941	514, 149, 245	2. 19	6, 822	456, 981	5. 20	40. 7	22. 9	10. 7
1942	582, 692, 937	2. 36	6, 972	461, 991	5. 12	45. 2	24. 4	11. 5
1943	590, 177, 069	2. 69	6, 620	416, 007	5. 38	48. 9	24. 7	13. 5
1944	619, 576, 240	2. 92	6, 928	393, 347	5. 67	52. 9	25. 6	16. 3
1945	577, 617, 327	3. 06	7, 033	383, 100	5. 78	56. 1	25. 6	19. 0
1946	533, 922, 068	3. 44	7, 333	396, 434	6. 30	58. 4	26. 0	21. 1
1947	630, 623, 722	4. 16	8, 700	419, 182	6. 42	60. 7	27. 7	22. 1
1948	599, 518, 229	4. 99	9, 079	441, 631	6. 26	64. 3	30. 2	23. 3
1949	437, 868, 036	4. 88	8, 559	433, 698	6. 43	67. 0	35. 1	24. 2
1950	516, 311, 053	4. 84	9, 429	415, 582	6. 77	69. 4	38. 5	23. 9
1951	533, 664, 732	4. 92	8, 009	372, 897	7. 04	73. 1	45. 0	22. 0
1952	466, 840, 782	4. 90	7, 275	335, 217	7. 47	75. 6	48. 7	23. 3
1953	457, 290, 449	4. 92	6, 671	293, 106	8. 17	79. 6	52. 9	23. 1

most of the factors essential for capacity production were present, including a maximum supply of railroad cars and 6 to 7 days of operation per week. Because of the intangibles and complexities involved, attainable production is something less than calculated capacity.

A coal mine is not like a factory, in that it cannot be closed and reopened in part or in full to meet fluctuating demands. Because of natural conditions of water and gas accumulations, roof falls, and similar problems, particularly in underground mines, maintenance of idle or only partly used mines is costly. Coal must therefore be sold at almost any price to keep the plants going, benefit from the advantages of steady output, and hold out as long as possible for a change to the better. Only when prices are not commensurate with costs and when capital investment has been drained do many mines close.

Factors that have recently tended to reduce the number of mines as well as to increase efficiency and reduce costs of production, include the accelerated growth in mechanization. As a result, bituminous-coal and lignite mines have been increasing in size. In 1924, when less than 1 percent of underground production was loaded mechanically, only 18 percent came from large mines (those having an annual output of 500,000 tons and over). By 1953 small mines (those producing less than 10,000 tons per year), constituted 54 percent of the total number of mines and accounted for only 3 percent of total production, whereas large mines constituted only 3 percent of the total number of mines but furnished 44 percent of the total production.

Because of geographic and other factors, the industry has not lent itself well in the past to combined ownership among comparatively few large concerns, as is common in many other major industries. There are some large coal companies, however. Further major combinations of this kind might establish leadership, methods, and concepts that would aid in overcoming some serious problems of the industry.

Besides commercial mines which sell their product in the competitive market, the industry includes many so-called "captive" mines owned and operated by parent companies primarily to supply their own coal requirements. These mines are said to furnish approximately 20 percent of total United States production. The principal captive owners are steel companies, electric power utilities, railroads, coke plants, and clay-product companies.

The rapid increase in mechanization in a comparatively short period, loss of markets to competing fuels, and increased efficiency in burning coal caused reductions in the number of men employed by the coal industry. Although employment dropped from a peak of 705,000 employees in 1923 to 293,000 in 1953, productivity increased from 4.47 to 8.17 tons per man per day. The latter figure is an average of 7.01 tons for underground, 25.30 for auger, and 17.62 for strip mining. Correspondingly, mechanical loading of coal underground increased from less than 1 percent in 1923 to almost 80 percent in 1953 and coal stripping from 2 to over 23 percent.

COST AND PRICE OF COAL

Two controlling forces operate to keep coal prices at levels that are not always commensurate with costs of production or with the attainment of reasonable profits—intraindustry competition and competition with oil and natural gas. Whereas under pressure from one or the other of these forces coal prices might rise in keeping with increases in costs or with required profit margins, pressure from both makes rises difficult under all but exceptional market conditions.

Current statistics on soft-coal prices at the consumer level are very sketchy, although they do throw some light on the market situation. Except for the annual average cost of bituminous coal at merchant coke ovens, the Bureau of Mines does not collect data on market prices of coal. The principal Government activities in this respect are: (1) Average wholesale and retail prices of soft coal, based on sampled coverage, as published by the Bureau of Labor Statistics; (2) monthly publication by the Interstate Commerce Commission of the average cost of railroad fuel purchased f. o. b. mines; and (3) Federal Power Commission information pertaining to the cost of coal at electric power utilities. The coal-trade journals publish some posted current wholesale prices, f. o. b. mine, usually in terms of high and low quotations.

The most valuable index of the trend of prices at the mines is the annual figures of average value per ton published by the Bureau of Mines. These data are on a national, State, and county basis and are classified according to the method of mining. The national average value f. o. b. mines has fluctuated widely from from year to year, although it remained remarkably constant from 1948 through 1954, ranging from $4.99 per ton in 1948 through $4.85 in 1950 to an estimated $4.82 in 1954. The main reason for this constancy, even though labor and other costs have increased during the period, is competition with other fuels.

There are wide variations in the average value f. o. b. mines among States and counties. Mining conditions, the quality of coal, the distance to markets, the method of transportation, and the competitive situation in the energy market largely explain these variations. The same factors also cause wide differences in costs and prices among producing companies. For example, the principal price factor above f. o. b. mine values is cost of transportation. For rail shipments at present it amounts to some 70 percent of the f. o. b. mine price. Those producers who are favored by circumstances often have a stabilizing influence on the market.

Conversely, such factors as excess capacity and production strongly influence market prices. Both encourage strong intraindustry competition, certain price instability, and lower profits than otherwise might be available. As little coal is stored at producers' facilities, particularly at rail-shipping mines, and as producers generally must dispose of all their sizes, both prepared and resultant, if they are to stay in operation, coal must move into the market within a reasonable time after production. Mines sometimes run long on resultant sizes and dispose of them at prices that disregard production costs. These surpluses can have a depressing effect on prices quite out of proportion to the amount of such coal available. The same coal being offered by several distributors or agents can assume the significance in the market of many times its actual quantity.

The fact that the demand for coal is relatively inelastic, except in a rising market, has much to do with the instability of prices and margins in the industry. Although consumers may switch orders from one supplier to another because of price advantages, a price reduction does not generally encourage consumers to buy appreciably greater amounts at any given time.

During periods of severe downward pressures on prices, high maintenance costs of idle facilities and efforts to avoid loss of investments force mining companies to continue operating while hoping for a change to better conditions, even though realization from sales may be below the cost of production; under such circumstances operations are maintained only by using up capital.

The shift in emphasis from domestic to industrial uses for coal has brought about some fundamental changes in coal sizes which, in turn, have affected the economics of the industry. Whereas in past years the large market for domestic sizes of coal carried premium prices, and therefore the major share of financial return to the industry, the present and future market will be preponderantly for industrial sizes. Because of competitive pressures, the coal industry has not been able fully to recast its pricing system to match the heavy shift to industrial sizes and thus to bolster its tenuous margin of profitability.

STOCKS

Stocks of coal at consumer facilities and retail yards and coal at terminal facilities and in transit are the principal balancing factors between production and consumption; there is little or no stocking of soft coal at mines. Approximately 7 to 9 million tons of coal is usually in railroad cars in transit. This backlog is very valuable in emergencies. Traditionally the largest industrial stocks (over 100 days' supply) are held by electric utilities and the smallest stocks (approximately 30 days' supply) by railroads. Retail yards have seasonal variation in the size of their stockpiles, building up in the summer and reducing during the winter.

CONSUMPTION AND USES OF COAL

Increased availability and consumption of petroleum and natural gas have wrought drastic changes in coal utilization. Bituminous-coal and lignite consumption dropped from a peak of 594 million tons in 1943 to 427 million tons in 1953, a net decline of 167 million tons (see table 8). Consumption during the first 11 months of 1954 was 15 percent below that of 1953, an additional drop of 60 million tons.

From 1943 to 1953 the coal industry lost 218 million tons of annual consumption in the following markets: 103 million tons used by railroads; 62 million sold by retail dealers; 48 million used for general industrial purposes, excluding electric power utilities; and 5 million used by steel and rolling mills, excluding coking coal. Over the past decade, however, 3 groups increased their annual consumption of bituminous coal: Electric power utilities (38 million tons); coke plants (10 million tons); and cement mills (3 million tons).

When utilization statistics are available for 1954, the net annual loss in the past 10 years will probably exceed 227 million tons. Nearly half of the total drop results from conversion of locomotives to diesel power. From 1943 to 1954 the number of steam locomotives in the United States declined 80 percent, while the number of diesel locomotives increased nearly 1,000 percent—from 1,500 to 16,400. Experimentation with a gas turbine using coal as fuel is now in process in an attempt to recover the lost railroad market.

In the 5 years ended with 1953, sales of mechanical coal stokers fell 80 percent, while shipments of oil burners increased 115 percent. Shipments of solid-fuel units for warm-air furnaces fell from nearly 400,000 to 72,000, whereas the sale of oil furnaces rose from 190,000 to 421,000 and gas-furnace shipments climbed from 188,000 to 504,000. In 1948 one solid-fuels unit was sold for each 1.6 units of oil and gas, but in 1953 the ratio was 21 : 1 in favor of oil and gas over solid fuels.

In the generation of electric energy, consumption of oil and gas increased 358 and 242 percent, respectively, from 1943 to 1953 compared with only 52 percent for coal.

The declining importance of coal in the total energy market is indicated further by the fact that, at the end of World War I, coal contributed over 80 percent of our total energy supply, whereas by 1953 it contributed only 32 percent of the annual new supply (see fig. 1). During 1943–53 petroleum and natural gas increased their contribution from 41 to 64 percent of the total supply. Waterpower remained constant at approximately 4 percent. Even though these percentages are for all types of energy (including automotive equipment and aviation, in which coal is not competitive), its contribution has been declining on an absolute basis as well, having dropped from almost 18 quadrillion B.t.u. in 1947 to about 12 quadrillion in 1953 (see fig. 2).

TABLE 8.—*Consumption of bituminous coal and lignite, by consumer classes, with retail deliveries in the United States, 1933–53*

[Thousand net tons]

Year	Electric-power utilities[1]	Bunker foreign trade[2]	Railroads[3] (class I)	Coke plants		Steel and rolling mills	Cement mills[4]	Other industrials[5]	Retail deliveries[6]	Total of classes shown[7]
				Beehive	Ovens					
1933	27, 088	1, 316	72, 548	1, 408	38, 681	10, 009	2, 832	83, 321	80, 482	317, 685
1934	29, 707	1, 321	76, 037	1, 635	44, 343	10, 898	3, 500	89, 448	86, 925	343, 814
1935	30, 936	1, 576	77, 109	1, 469	49, 046	11, 747	3, 516	96, 937	83, 990	356, 326
1936	38, 104	1, 622	86, 391	2, 698	63, 244	13, 471	4, 771	113, 792	84, 200	408, 293
1937	41, 045	1, 832	88, 080	4, 927	69, 575	12, 853	5, 247	127, 142	80, 076	430, 777
1938	36, 440	1, 352	73, 921	1, 360	45, 266	8, 412	4, 483	96, 527	68, 520	336, 281
1939	42, 304	1, 477	79, 072	2, 298	61, 216	9, 808	5, 274	103, 079	71, 570	376, 098
1940	49, 126	1, 426	85, 130	4, 803	76, 583	10, 040	5, 633	110, 469	87, 700	430, 910
1941	59, 888	1, 643	97, 384	10, 529	82, 609	10, 902	6, 832	124, 868	97, 460	492, 115
1942	63, 472	1, 585	115, 410	12, 876	87, 974	10, 434	7, 570	135, 979	104, 750	540, 050
1943	74, 036	1, 647	130, 283	12, 441	90, 019	11, 238	5, 851	145, 518	122, 764	593, 797
1944	76, 656	1, 559	132, 049	10, 858	94, 438	10, 734	3, 789	134, 610	124, 906	589, 599
1945	71, 603	1, 785	125, 120	8, 135	87, 214	10, 084	4, 215	129, 606	121, 805	559, 567
1946	68, 743	1, 381	110, 166	7, 167	76, 121	8, 603	7, 009	120, 610	100, 586	500, 386
1947	86, 009	1, 689	109, 296	10, 475	94, 325	10, 048	7, 938	126, 948	99, 163	545, 891
1948	95, 620	1, 057	94, 838	10, 322	96, 984	10, 046	8, 554	112, 741	89, 747	519, 909
1949	80, 610	874	68, 123	5, 354	85, 882	7, 451	7, 988	98, 957	90, 299	445, 538
1950	88, 262	717	60, 969	9, 088	94, 757	7, 698	7, 943	98, 164	86, 604	454, 202
1951	101, 898	890	54, 005	11, 418	102, 030	7, 973	8, 525	105, 634	76, 531	468, 904
1952	103, 309	723	37, 962	6, 912	90, 702	6, 820	8, 073	95, 863	68, 393	418, 757
1953	112, 283	605	27, 735	8, 226	104, 648	6, 207	8, 362	97, 437	61, 295	426, 798

[1] Federal Power Commission figures for public-utility power plants, ncluding a small quantity of coke, approximately 100,000 tons annually.

[2] Bureau of Census, U. S. Department of Commerce.

[3] Association of American Railroads.

[4] Includes a small amount of anthracite.

[5] Estimates based upon reports collected from a selected list of representative manufacturing plants.

[6] Estimates based upon reports collected from a selected list of representative retailers. Include some coal shipped by truck from mine to final destination.

[7] Approximate total consumption.

DEFENSE PROGRAMS

Coal is of vital importance in national emergencies, both as a source of energy and as a raw material for military and civilian requirements. The availability of adequate quantities of proper sizes and kinds of coal and their correct distribution aid in the timing and success of military programs. For the production of iron, steel, and electric power in the huge quantities required for national defense, coal is indispensable. Similarly, it is essential for manufacturing expansion generally, for space heating, and for the production of chemical raw materials such as toluol (a prime ingredient of explosives), benzol (vital in the manufacture of aviation gasoline and synthetic rubber), tar derivatives (used in the manufacture of a variety of important chemical products such as dyes, drugs, creosote oil, and fertilizers).

Almost 1.4 tons of coal is needed to produce 1 ton of coke, nearly nine-tenths of which is required for making 1 ton of pig iron. Coal now, in peacetime, provides over 50 percent of the total electric power in the United States. In manufacturing, 6 pounds of coal is used to make 1 pound of smokeless powder, 2 tons for a 1-ton bomb, and 220 tons for a modern tank. In addition to expanded requirements for coal for the kinds of uses to which it is normally put, coal must replace fuel oil when petroleum is diverted to more strategic uses and when oil supplies are limited because of transportation difficulties, including curtailment of offshore shipments because of submarines.

Coal is a bulk commodity consumed by the hundreds of millions of tons; in turn, it requires other materials and services that are scarce in national emergencies. Thus, the best preparation for defense (with regard to any basic commodity) is reasonably healthy operation during peacetime. To produce and distribute coal in wartime, adequate manpower must be available at mines, as well as transportation facilities, particularly open-top railroad cars. Stockpiling and distribution must be modified to allow for diversions to war industries, defense-production centers, and military bases. These other problems require planning and coordination of industry and Government, including the forecasting of requirements for coal, productive capacity, manpower, equipment, supplies, and services so that the huge quantities required for defense will be available when and where needed.

FOREIGN TRADE

The United States is both an importer and an exporter of coal. Although it has a large sur-

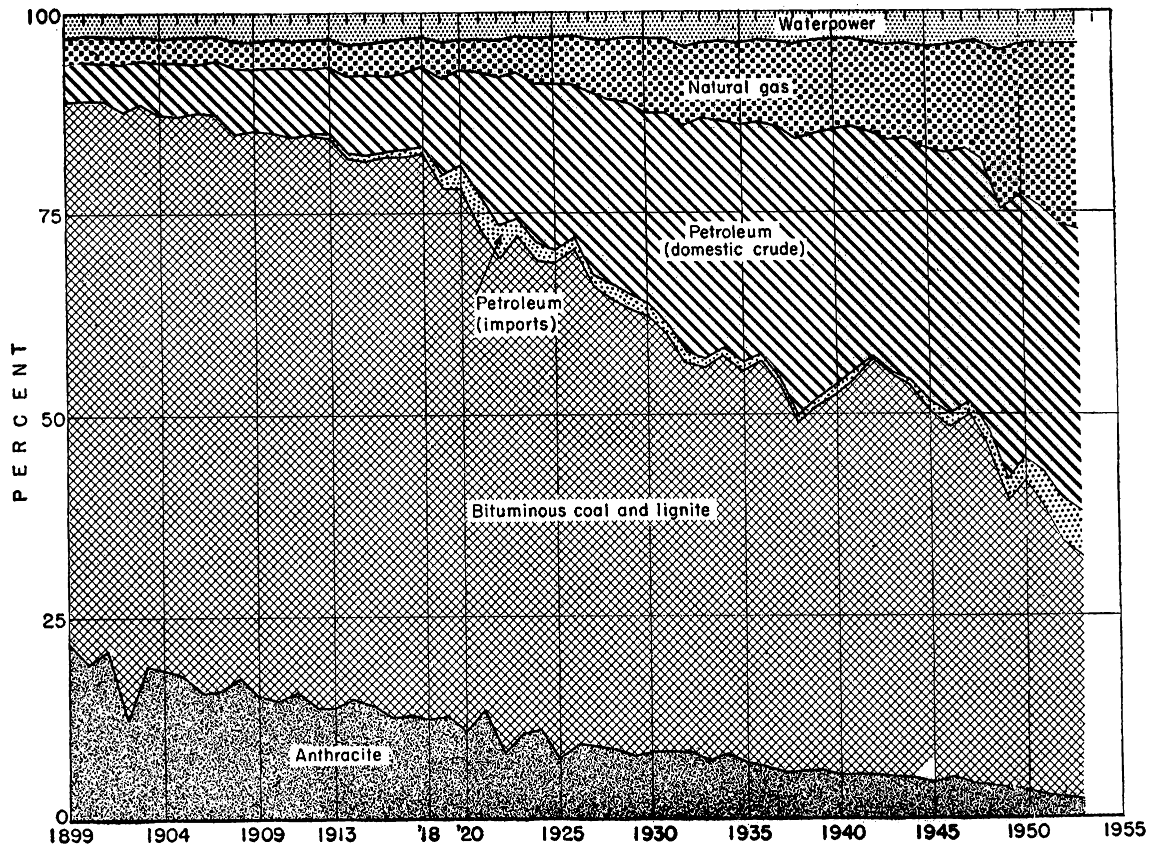

FIGURE 1.—Percentage of Total British Thermal Units Equivalent Contributed by the Several Sources of Energy in the United States, Counting Waterpower at the Prevailing Fuel Equivalent of Central Stations in Each Year, 1899–1953.

plus, Canadian coal goes to a few States—from British Columbia to Washington, Montana, and Idaho and from Nova Scotia to Maine and New Hampshire. The volume is not great, however, compared with exports.

For many years Canada has been the principal customer for United States coal, averaging around 20 million tons a year. Overseas exports, chiefly to Europe, have fluctuated widely, influenced primarily by war, the aftereffects of war, and prolonged strikes. Substantial shipments were made to Europe during and after World War I, during the British coal-mine strike in 1926, and from 1947 (36 million tons) to 1953 (8 million tons). Recently, total exports ranged from 69 million tons in 1947 to a low of 25 million in 1950. Following the outbreak of the Korean War in 1951, the demand increased, and exports in that year totaled 57 million tons. With expansion of foreign coal-producing facilities, exports fell to 34 million tons in 1953. In recent years a considerable amount of bituminous coal has been exported to South America and to Asia.

Coal production increased in most European countries during 1953; but, as requirements exceeded available supplies, production from the United States made up a large part of the deficit. Because of increasing demands for energy throughout the world, an appreciable export market should be available to the United States for some years.

OUTLOOK

Although production of bituminous coal and lignite has dropped seriously until in 1954 it fell to the level of the midthirties, employment in the industry was the lowest since before the turn of the century, and coal's share of the total energy market has been declining drastically, coal still is the top supplier of industrial energy. It is the greatest reserve of energy except possibly nuclear sources, and it is indispensable to industrial expansion in national emergencies.

The principal reason for its declining use is the shift to other sources of energy. Although future coal production will depend on the avail-

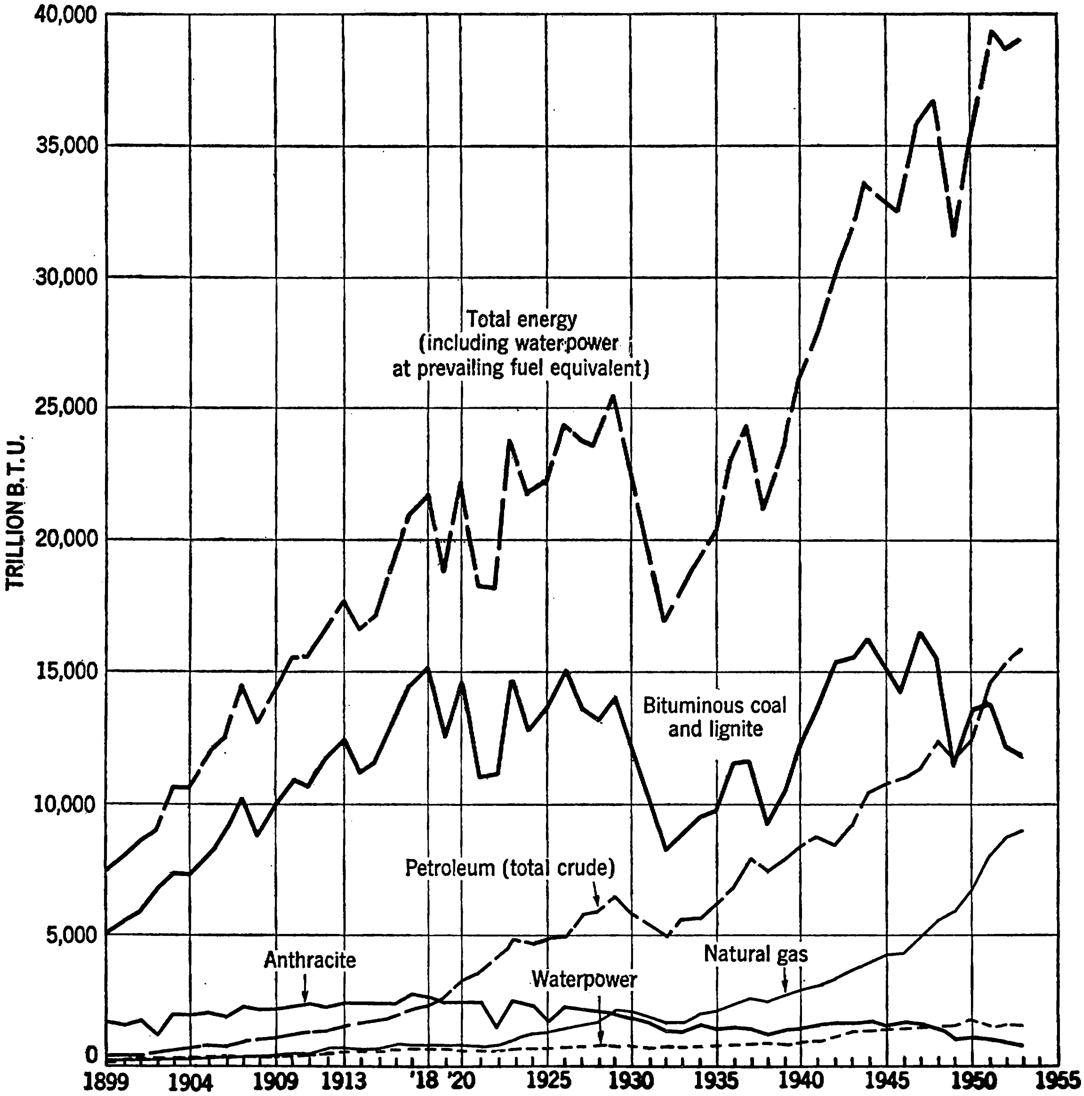

FIGURE 2.—Annual Supply of Energy From Mineral Fuels and Waterpower in the United States, 1899–1953.

ability and utilization of oil and natural gas, it is believed that the consumption of coal has reached its lowest point and that the economy already has entered a transitional period in which the demand for energy of all kinds will increase heavily. Population is growing rapidly, and business and industry should expand accordingly. The gross national product (the total national output of goods and services) is increasing steadily, and the tremendously accelerated growth of electric power has been demonstrated. All such factors indicate a substantial growth in the steel industry, which will be reflected in heavily increased demand for coke. Advances in coal technology are being made, which will result in lowered production costs and increased efficiency in consumption. There are certain limitations in the sources of competing fossil fuels to supply the expanding energy demand. New uses for coal will be found in hydrogenation, gasification, gas syntheses, and carbonization processes and in the production of nuclear fuels. In view of these and other factors, experts predict that the demand for energy will be so greatly expanded that the annual coal output will reach 750 million tons or more within the next 20 years.

PROBLEMS

The principal present problems in coal utilization include the development of means to enable coal to compete more favorably with oil and natural gas. The basic challenge here is advancement of technology to reduce costs of coal production and to develop methods of handling and burning coal that will give increased efficiency and greater convenience in the use of coal and disposal of its waste products.

Another means of increasing consumption is the development of new, large-scale uses of coal. Among these are the following processes, all requiring further research:

1. Production of organic chemicals by mild hydrogenation of coal.
2. Production of liquid fuels and chemicals by more rigorous and fuller hydrogenation of coal.
3. Production of liquid fuels and chemicals from water gas.
4. Complete gasification of coal by causing it to react with oxygen and steam for making high-B. t. u. gas, liquid fuels, and chemicals.

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BORON

By

Joseph C. Arundale [1]

AFTER its difficult early history in the Far West nearly a century ago, the domestic boron industry has evolved as the world's principal source of boron minerals.

Everyone has washed his hands in borax cleansing powders, and boric acid has a place on every medicine shelf, but few are familiar with the multitude of other uses for boron in its many forms. It is consumed in almost every type of industry and the future promises many new uses.

Summary

In the United States boron minerals now are produced exclusively from the arid regions of California. This area produces most of the world's supply of boron.

The common boron compounds of commerce are inexpensive, and their prices have declined to a point where transportation to important consuming areas usually is the largest element of cost. They are plentiful, and known reserves equal to anticipated requirements for scores of years. Production and consumption have increased fourfold in 20 years, trebled in 10 years. Continued expansion of the industry is expected.

The problems of the industry are principally those of more efficient production, distribution and utilization, development of new uses, the long-range problem of reserves, and maintenance of necessary fact-finding activities to serve as a basis for optimum development of the industry.

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[1] Assistant chief, Branch of Construction and Chemical Materials, Bureau of Mines.

BACKGROUND

The principal boron minerals are: Borax (tincal), $Na_2B_4O_7 \cdot 10H_2O$; kernite (rasorite), $Na_2B_4O_7 \cdot 4H_2O$; colemanite (borocalcite), $Ca_2B_6O_{11} \cdot 5H_2O$; ulexite (boronatrocalcite), $CaNaB_5O_9 \cdot 8H_2O$; priceite (pandermite), $5CaO \cdot 6B_2O_3 \cdot 9H_2O$; boracite (stassfurtite), $Mg_7Cl_2B_{16}O_{30}$; and sassolite (natural boric acid), $B(OH)_3$.

Many other minerals contain boron, but they are either limited in quantity available or low in boron content.

HISTORY

Borax has been known and used for many centuries. Tibet is said to have been the first important source, and later Italy was the world's chief supplier. Then South America became the principal producer. By the middle of the 1920's, production in California had made the United States the world's foremost producer.

About 1864 borax crystals were recovered on a commercial scale from small lakes in Lake County, Calif. These crystals were separated from the mud of the lake bottom by washing and purified by solution and recrystallization. Several years later borax and ulexite were being recovered from saline marshes in Nevada and California. Borax next was produced from Searles Lake (then known as Slate Range Lake or Borax Lake) until about 1895. The scene of activity then shifted to Death Valley, where Pacific Coast Borax Co. began producing colemanite. Production from Searles Lake began again in 1908 and since 1916 has been continuous; various materials, such as potash, salt cake, bromine, and lithium phosphate, are recovered as coproducts.

Discovery of a large deposit of kernite and borax in the Kramer district of California was reported in 1926; since that time, the deposit has been developed into the world's largest source of boron compounds.

PRODUCTION

Four firms currently produce natural boron minerals and compounds—Pacific Coast Borax Co., American Potash & Chemical Corp., West End Chemical Co., and United States Borax Co. The total output of various boron compounds by these firms in 1953 was 715,228 short tons valued at over $17 million at the mine.

Two firms produce ferroboron—Molybdenum Corp. of America at Washington, Pa., and Electro Metallurgical Co. at Niagara Falls, N. Y. Molybdenum Corp. of America also produces manganese boride. Ohio Ferro-Alloys Co., Philo, Ohio, produces a ferrosilicon boron.

Elemental amorphous boron is produced by the F. W. Berk Co., Woodridge, N. J.; American Potash & Chemical Corp., Los Angeles, Calif.; Cooper Metallurgical Associates, Cleveland, Ohio; and the Norton Co., Worcester, Mass. The last two concerns also produce the nitride and carbide.

OCCURRENCES

The bulk of production comes from bedded deposits of kernite and borax in the Kramer district. Most of the remainder is recovered as borax, anhydrous sodium tetraborate, or boric acid from the brine of Searles Lake, Calif. Colemanite is produced near Shoshone, Inyo County, Calif. Other deposits of boron minerals in California, Nevada, and Oregon currently are not being worked.

There are many deposits of boron minerals throughout the world. The following are some of the more important deposits that are being worked or have recently produced: Ulexite in Peru, Chile, Bolivia, and Argentina; priceite in Turkey; sassolite in Italy; stassfurtite in Germany; and borax (tincal) in Tibet.

The deposit in the Kramer district is a 200- to 250-foot thick series of beds of kernite and borax interlayered with shale and covered by several hundred feet of alluvial sediments. The ore is mined by shrinkage-stoping methods and is hoisted through vertical shafts to the surface, where it is crushed. The borax is recovered by magnetic separation; the kernite first is calcined, then separated by air. Refined products are made by solution and recrystallization. The product is sold as borax, converted into boric acid by treatment with sulfuric acid, or fused to make anhydrous borax. Some of the borax or boric acid is converted to other borates, such as potassium borate, ammonium borate, and sodium metaborate.

TECHNOLOGY

Borax from Searles Lake is produced by concentrating brine in triple-effect vacuum pans. Potash is then removed by cooling. Finally, borax is crystallized from the solution and refined to commercial grade, or it may be made anhydrous by fusing. Some borax is converted to boric acid. In another process the sodium carbonate is removed with carbon dioxide, and borax is crystallized by cooling and further refined by dissolving in water and recrystallizing.

Borax produced by American Potash and West End Chemical is a byproduct or coproduct.

Other products of these operations are soda ash, salt cake, potash, sodium chloride, bromine, or lithium compounds.

RESERVES

Reserves of soluble salts in Searles Lake are adequate for many years—at least 50—at the present rate of depletion. The area covered by these salts is believed to be about 30 square miles. Reserves of kernite and borax in deposits in the Kramer district are considered very large and have been estimated at least 100 million short tons (*13*).[2] Reserves in other areas, particularly in California, Nevada, and Oregon, add to the total, and the possibility of finding other deposits or extensions of known deposits is considered to be very good.

With the large domestic reserves of boron minerals available, the United States is expected to be self-sufficient in boron compounds for many years and probably will supply much of the world's requirements.

USES

The useful properties of borax include its easy solubility in water to a mildly alkaline and antiseptic solution and its excellent fluxing properties. Although an estimated one-half of the supply goes into the glass and ceramics industry, other uses are legion. It is used in washing, cleansing, and laundering, either directly or in soaps. Boron compounds have many applications in medicine and pharmacy. Borax is used in manufacturing playing cards, plywoods, plasters, and paints; to prevent mildew in starches, to prevent mold on citrus fruits, leather, and textiles, and to retard the growth of fungi in lumber; for fireproofing wood, paper, and fabrics; for neutralizing free acid in leather and in dyeing leather and textiles; in cleaning hides and skins; in insect repellents; and in the manufacture of candles, carpets, drug specialties, dyes, hats, ink, jewelry, oil, paint, and paper. It is a constituent of porcelain enamels, pottery, bricks, china, ceramic ware, glazes, glass, and welding flux. It is used in smelting copper; refining gold and silver; as a deoxidizer of brass and bronzes; in steel; in hairwaving; in bleaches; in electroplating; and as an abrasive. It is an essential plant nutrient. In short, it has an amazingly diversified variety of uses and is utilized in almost every field of industry.

In most of its forms, borax is so inexpensive and plentiful that substitutes are seldom considered.

Boron usually is consumed or incorporated into the ultimate product, and recovery is not possible. Secondary recovery is not a significant factor in the industry.

The following table shows production, consumption and foreign trade data.

TABLE 1.—*Salient statistics of boron minerals and compounds in the United States, 1933–53*

Year	Sold or used by producers		Imports for consumption (refined)[1] (pounds)	Exports (short tons)	Apparent consumption[2] (short tons)
	Gross weight (short tons)	B_2O_3 content (short tons)			
1933	188, 047	56, 000	1, 061	87, 677	101, 440
1934	242, 500	73, 000	335	103, 643	138, 857
1935	272, 967	82, 000	748	114, 447	158, 520
1936	313, 759	94, 000	1, 887	102, 021	211, 739
1937	358, 898	108, 000	724	154, 052	204, 846
1938	215, 662	65, 000	631	77, 519	138, 143
1939	245, 284	81, 800	774	91, 139	154, 145
1940	243, 355	80, 900	752	64, 313	179, 042
1941	301, 282	95, 200	2, 025	41, 793	259, 490
1942	226, 723	77, 600	------------	36, 542	190, 181
1943	256, 633	87, 600	------------	26, 877	229, 756
1944	277, 586	91, 700	------------	32, 759	244, 827
1945	325, 935	104, 600	1, 344	43, 577	282, 359
1946	430, 689	129, 800	100, 544	53, 303	377, 436
1947	501, 935	145, 700	1, 884	85, 736	416, 200
1948	450, 932	134, 700	3, 056	70, 940	379, 994
1949	467, 592	139, 200	886	109, 491	358, 101
1950	647, 735	191, 000	1, 224	142, 580	505, 167
1951	862, 797	241, 000	1, 424	213, 445	649, 353
1952	583, 828	169, 100	860	103, 292	480, 536
1953	715, 228	213, 300	624	139, 317	575, 911

[1] In addition, crude was reported for the following years: 1933—2,138,000 pounds; 1939—348 pounds; 1943—525 pounds; 1950—21,826 pounds; and 1952—88 pounds.

[2] Quantity sold or used by producers plus imports minus exports.

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

PRICES

From the time borax was discovered in California nearly 100 years ago, the price has declined more or less consistently. At that time the refined material was selling for about 35 cents a pound. Bagged granular borax is now selling for about $41 a ton f. o. b. works and about $89 a ton f. o. b. New York, indicating that over half of the delivered price of borax in the principal consuming areas is transportation cost. In contrast to the relative cheapness of borax, boron metal sells up to about $250 a pound for 98-percent purity.

TAXES

Under the Federal income-tax law boron-minerals producers have a percentage depletion allowance of 15 percent of gross income not to exceed 50 percent of the net income without depletion deduction.

TARIFF

Crude or unmanufactured borax, borate of lime, borate of soda, and other borate material, crude and unmanufactured, are on the list of articles free of import duties. Refined sodium borate or borax carries an import duty of ⅛ cent per pound. Boric acid has a rate of 1¾ cents per pound. Boron carbide is dutiable at the rate of 6¼ percent ad valorem. Boron metal is dutiable at the rate of 12½ percent ad valorem; ferroboron is dutiable at the rate of 12½ percent ad valorem. Only small quantities of these materials are imported.

RESEARCH

A major factor in the rapid increase in boron-mineral consumption has been vigorous research by producers and consumers. The principal producers have plans for expanding their research on the applications of boron. The Bureau of Mines also has done considerable work on boron. A program of research on hard borides involves studies of the production and properties of various borides. Boron producers and the steel industry have done much research on the use of boron in steels.

OUTLOOK

Boron in its various forms has widespread use, and consumption in recent years has increased greatly. Numerous new fields of commercial application contribute to the current outlook for continued expansion in use. Small quantities of boron in the form of ferroboron or other boron additive are now being used as a partial substitute for such materials as nickel, chromium, and molybdenum to improve the hardenability of steel. There have been reports of the great potential of the organic boron compounds. Other boron compounds have catalytic properties of possible value to the petroleum industry. At present boron carbide, the hardest material manufactured commercially, is used only in small quantities, but larger markets for this and other hard boron compounds may develop from research now underway.

Known reserves of commercial-grade boron minerals will last for many years. The chances for discovery of new commercial deposits and extensions of known deposits are good. The adequacy of the reserve situation and expanding uses indicate continued growth of the industry.

PROBLEMS

The problems confronting the boron industry are those of developing new uses and promoting wider utilization of boron in its many forms. Although known reserves of boron minerals in a few of the principal deposits are large, a more adequate appraisal of the overall reserve position is needed.

Boron minerals are becoming increasingly important raw materials. Producers, consumers, and Government agencies concerned with this commodity need reliable technical, statistical, and economic information to guide management and Government activities.

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BROMINE

By

Joseph C. Arundale[1]

MOST of the bromine produced is recovered from the sea, burned in the Nation's automobiles, and blown out through the exhaust into the atmosphere.

Summary

Bromine is one of the halogen group of elements, along with iodine, chlorine, and fluorine. It rarely occurs in mineral form but often in mineralized water. There is an inexhaustible source in sea water, which furnishes the bulk of current supplies. It is recovered as a byproduct or coproduct of magnesium, potash, salt, sodium, or calcium compounds. Production in 1953 exceeded 192 million pounds of bromine compounds valued at 35 million dollars and containing 164 million pounds of elemental bromine. By far the largest single use for bromine is as ethylene dibromide in gasoline antiknock additive, where it counteracts the deposition of lead from tetraethyl lead. Other uses are as a soil and food fumigant, on photographic film and in medicine.

The outlook is for higher demand; however, the development of jet engines, redesign of automotive engines, and substitutes are problems in the industry because they could adversely affect demand.

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[1] Assistant chief, Branch of Construction and Chemical Materials, Bureau of Mines.

BACKGROUND

HISTORY

Bromine was discovered about 1824. Early production was from the saline deposits of Stassfurt, Germany, which contain 0.15 to 0.25 percent bromine. It was produced first in the United States about 1846 but was of scientific interest only until the 1860's. By that time medicinal and photographic uses had been developed. Germany and the United States were in active competition for the market until the First World War. A requirement for bromine in war gases was added to the other demands, and annual domestic production from brine-processing plants along the Ohio River and in central Michigan increased to nearly 2 million pounds. The next major impetus to the industry came in the 1920's, when a mixture of tetraethyl lead and ethylene dibromide was introduced as a gasoline antiknock compound. Additional supplies rapidly were made available from well brines in Ohio, Michigan, and West Virginia. In 1924 the cruise of the S. S. *Ethyl*, a floating chemical-research plant, demonstrated the feasibility of extracting bromine from sea water by a precipitation process. Later a process was developed by which the bromine was blown from sea water by air, and this source soon was furnishing the bulk of domestic supplies.

SOURCES AND PRODUCTION

Bromine is unique in being the only nonmetallic element that is liquid under normal conditions of temperature and pressure; however, it volatilizes easily and never occurs uncombined in nature. Its occurrence in mineral form is rare, but it is found widespread in brines and saline deposits. Sea water contains an average of about 60 to 70 parts per million. Some brines contain several hundred parts per million.

In the United States Ethyl-Dow Chemical Co. recovers bromine from sea water at Freeport, Tex. A second sea-water plant is operated by Westvaco Chemical Division of Food Machinery & Chemical Corp. in the San Francisco Bay area. The following firms recover bromine from well brines in Michigan: The Dow Chemical Co., Midland; Great Lakes Chemical Corp. at Filer City; Michigan Chemical Corp. at St. Louis and Manistee; and Morton Salt Co. at Manistee. Pomeroy Salt Co. at Minersville, Ohio, and Westvaco Chemical at South Charleston, W. Va., also treat well brines. American Potash & Chemical Corp. recovers bromine from the brine of Searles Lake in California.

In 1953 the annual domestic production of bromine compounds had exceeded 192 million pounds valued at approximately 35 million dollars and containing over 160 million pounds of elemental bromine.

Bromine also is recovered from the Stassfurt saline deposits in Germany, and in Italy, Japan, France, Tunisia, South Africa, and other countries.

The process by which bromine is recovered from raw sea water involves acidulation of the feed water, followed by chlorination to liberate bromine, which is then blown out by air or steam. The bromine is caught in a soda-ash solution in absorption towers. The resulting sodium bromide and bromate are treated with acid to free the liquid bromine. The process for treating well brines is very similar. The bromine is liberated by chlorine, vaporized by steam, and either condensed as liquid bromine or absorbed in alkali.

USES

Only a small portion of the output is sold as liquid bromine. The bulk is consumed as ethylene dibromide, the balance being made up of sodium bromide, potassium bromide, and minor quantities of several other compounds.

In antiknock compounds containing tetraethyl lead, ethylene dibromide is added to prevent lead from depositing on the cylinders, valves, and spark points of the motor.

Bromine is important in both the organic and inorganic fields of the chemical industry. It has many medicinal and pharmaceutical uses, such as in disinfectants, anesthetics, and the treatment of nervous diseases. In the photography industry, silver bromide is important in photographic film. It is used in manufacturing dyes, ink, resins, and leather and rubber products. The military services have used bromine in poisonous and tear gases. In the metallurgical industry, it has been used as "mining salt" in the bromo-cyanogen process for extracting gold from its ores. Chlorobromomethane is used as a fire-extinguishing compound.

TABLE 1.—*Bromine and bromine compounds* [1] *sold by primary producers in the United States, 1949–53*

[Thousand pounds and thousand dollars]

	1949	1950	1951	1952	1953
Elemental bromine:					
Gross weight	3,429	4,063	[2] 6,420	[2] 6,692	7,834
Bromine content	3,429	4,063	[2] 6,420	[2] 6,692	7,834
Value	$539	$760	[2] $1,312	[2] $1,274	$1,701
Sodium bromide:					
Gross weight	809	994	1,006	910	973
Bromine content	628	772	781	706	756
Value	$209	$257	$279	$262	$277
Potassium bromide:					
Gross weight	1,926	2,879	3,288	2,664	2,793
Bromine content	1,293	1,933	2,208	1,789	1,875
Value	$499	$748	$898	$708	$778
Ammonium bromide:					
Gross weight	265	403	401	(3)	376
Bromine content	216	329	327	(3)	307
Value	$78	$117	$128	(3)	$123
Other, including ethylene dibromide:					
Gross weight	98,407	108,080	141,275	173,452	180,653
Bromine content	83,160	91,405	119,827	147,014	153,371
Value	$14,943	$16,913	$23,563	$28,396	$32,493
Total:					
Gross weight	104,836	116,419	152,390	183,718	192,629
Bromine content	88,726	98,502	129,563	156,202	164,143
Value	$16,268	$18,795	$26,180	$30,639	$35,372

[1] Bromine content calculated as theoretical bromine content present in compound.

[2] Also includes some bromine used to make other products than those listed below. Prior to 1951 this quantity was excluded.

[3] Included with "Other including ethylene dibromide."

Organic bromides are useful soil fumigants. Other bromide mixtures are used as fumigants to protect foodstuffs from insect infestation. Potassium bromate reportedly is used in bread to improve the texture and reduce loss of carbohydrates. Colors made with bromine compounds are used in lipsticks. Some home permanent-wave kits contain bromine compounds as neutralizers. It is used in water sterilization and in sanitation.

The chemical properties of bromine are somewhat similar to those of chlorine and iodine, and some substitution is possible with a sacrifice of cost or efficiency.

PRICES

Soon after the technology of bromine production from brine was developed the price declined and in 1880 was about 25 cents a pound. The wholesale price for bulk elemental bromine remained at about that price (with fluctuations during the two World Wars), and the present price is about 27 cents a pound.

TARIFF

The Tariff Act of 1922 levied 10 cents a pound on bromine and bromine compounds imported into the United States. This rate is still in effect, except that in 1940 the duty on ethylene dibromide was changed to 6 cents a pound plus 30 percent ad valorem. Imports of bromine and bromine compounds are negligible.

TRANSPORTATION

Bromine is a poisonous and corrosive liquid, and the vapors are asphyxiant, therefore it requires special precautions in handling and transportation. In bulk it is shipped in tank cars or drums of lead-lined steel, nickel, or Monel metal. Smaller lots are shipped in cases of nine 6½-pound bottles of glass. Similar care is necessary in handling certain other bromine compounds.

OUTLOOK

The number of automobiles on the Nation's highways increases each day. Improved engine design and the motorists' demand for better performance call for more and better gasoline. This in turn is increasing the demand for antiknock fluids of which bromine is an ingredient. Expanding airplane mileage is adding to the quantity of high-grade gasoline being burned.

More bromine is being used in fumigating soil, grains, and other foods and food-processing equipment.

As the output of magnesium, salt, potash, and other materials from sea water and brines is increased, bromine as a byproduct or coproduct could also be made available in larger quantities, or it can be recovered from sea water and known brine sources without recovery of other minerals in the solution.

These uptrends support the widely held belief that further growth in output can be anticipated. In reality, the prospects are more uncertain owing to development of substitutes and types of engines that do not require high-octane fuel. Designs have been suggested for automotive engines that do not require gasoline additives for proper functioning. Modern jet-aircraft engines do not require a fuel with antiknock additive, although it is often used because an all-purpose fuel is easy to stock.

PROBLEMS

The abundance of bromine and the fact that the bulk is used in a single application manifest the need for new and more diversified markets.

Jet engines do not need fuels with tetraethyl lead additive; therefore, the future of the aircraft industry holds a potential threat to a substantial part of the market for ethylene dibromide.

Nearly 15 percent of the Nation's supply of lead goes into tetraethyl lead and subsequently is lost in the atmosphere. A possible future need for conserving lead thus might imperil the market for bromine.

The supply of bromine and other elements recovered from sea water may be affected by the results of current research on desalting it.

The relatively high price of bromine deters its wider use, for example, as an alternative to chlorine in organic synthesis and as a heavy solution in mineral separation processes.

The combination of ample reserves and somewhat uncertain competitive situation emphasize the need for maintaining active research and other fact-finding activities to provide reliable technical, statistical, and economic information for guiding both management and Government agencies concerned with this commodity.

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CADMIUM

By

Robert L. Mentch[1]

CADMIUM, one of the rarer metals, has outstanding properties for specialized uses that would expand its market manyfold if more of the metal were consistently available and the price lower. Cadmium apparently is destined to remain relatively scarce and its supply unresponsive to demand because the entire primary supply comes as a byproduct of the treatment of ores of other metals, particularly zinc.

Summary

Cadmium is a soft, bluish, silver-white metal. It is used chiefly in electroplating, pigments, and low-melting-point alloys; about 95 percent of the available supply in the United States is consumed for these purposes, with electroplating by far the largest use.

Cadmium minerals are rarely found in commercial quantities, and no ore is mined solely as a source of cadmium. The entire domestic supply of primary cadmium is recovered as a byproduct of treating ores of other metals, particularly zinc.

The United States is the world's leading producer of cadmium; in 1940–53 production at domestic plants furnished approximately two-thirds of the total world output. United States production, however, is not solely from domestic materials. It is estimated that since 1940 about 60 percent of the primary cadmium metal produced in the United States was of foreign origin. Mexico was the chief foreign source, followed by Canada, Peru, and South-West Africa.

The cadmium industry of the United States consists of four groups—primary producers, secondary producers, distributors, and consumers. On the production side primary producers are of preponderant importance and supply over 95 percent of the total annual output; they are, for the most part, large zinc producers. Eleven companies produce primary metallic cadmium at 13 plants in the United States.

Although cadmium is currently in oversupply, during the next 10 years it is expected that demand and supply in the United States will essentially balance. Because of its byproduct relationship with zinc, supply is not responsive to price. Important new uses causing increased demand and possibly price increases may be expected to have these results: More plentiful metals will be substituted in some uses, particularly zinc for electroplating; the supply will be augmented slightly by increased attention to cadmium recovery and to deposits containing cadmium as the chief or important mineral constituent; and imports of metal and cadmium-bearing raw materials will be increased.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

HISTORICAL SKETCH

Cadmium was discovered in 1817 by F. Strohmeyer, a professor at Göttingen, Germany, in a sample of zinc carbonate yielding a yellow zinc oxide. Later, several German chemists, each working independently, found cadmium in laboratory specimens of zinc oxide.

For over 60 years after its discovery, few applications were found for cadmium; and, because of its association with zinc, it was frequently referred to as "the unwanted stepchild of zinc." It was first used in the sulfide form in paint pigments. Later cadmium was used in low-melting alloys, for electroplating iron and steel, and in the form of salts for dentistry, dyeing and calico printing, glassmaking, photography, and chemical reagents.

The first commercial cadmium was produced in Germany; and until 1911 that country was the only important producer, obtaining the metal from cadmium-bearing zinc ores of Upper Silesia. World production during this period was relatively small, reaching a peak of about 100,000 pounds in 1910, with Germany furnishing over 90 percent of the total.

Cadmium was first produced in the United States in 1906 on a pilot-plant scale. In 1907 the Grasselli Chemical Co. of Cleveland, Ohio, recovered 14,000 pounds of the metal as a byproduct of zinc-smelting operations. Production in the United States increased rapidly as new uses for the metal were discovered and improved recovery techniques were introduced. With production of 207,000 pounds in 1917 the United States became the world's largest producer, a position which, except for 1918, has been retained ever since.

PROPERTIES AND USES

Cadmium is a soft, malleable, ductile, bluish, silver-white metal. Its chemical symbol is Cd, atomic weight 112.41, atomic number 48, and most common valence 2. The metal has a specific gravity of 8.642 and melting and boiling points of 320.9° C. and 767° C., respectively.

Cadmium is used chiefly in electroplating, pigments, and low-melting point alloys; about 95 percent of available cadmium is consumed for these purposes, with electroplating by far the largest use. For the period 1940–44, 71 percent of the cadmium consumption of the United States was for electroplating, 11 percent for bearing alloys, and the remainder for pigments, solders, miscellaneous alloys, and various chemicals. More recent information on end uses is not available, but the current pattern is believed to be much the same as in 1940–44, except that the use of cadmium for pigments and low-melting-point alloys is considerably larger than that for bearing alloys, which has declined markedly since World War II.

Cadmium is used as a protective coating for iron and steel, and, to a much smaller extent, for high-copper alloys and other metals and alloys. Although cadmium may be applied by spraying or hot dipping, it is most commonly electrodeposited. Cadmium and zinc (unlike many other electroplated coatings, such as copper, tin, nickel, and chromium) protect steel and copper-alloy base metals electrochemically. Thus metals other than zinc, when substituted for cadmium, must be applied in appreciably greater thicknesses to give equal protection. Cadmium is used as an electrodeposited coating in preference to zinc for the following reasons: (1) In thinner coatings it provides equal protection in some applications, principally where the plated article is subjected to extended alkali or salt-water exposure; (2) the rate of deposition for a given amperage of electric current is larger, hence electric power costs are reduced; (3) the electrical resistance of plated contacts is lower; (4) cadmium retains its metallic luster longer; (5) plated parts are soldered more easily; (6) cadmium is superior in throwing power, or the ability to deposit uniformly on intricately shaped objects; (7) it gives greater simplicity to plating-process control and has the ability to plate on cast and malleable iron and on high-carbon and carburized-steel surfaces; (8) it has higher cathode efficiency, hence less tendency for hydrogen embrittlement than with zinc; (9) its electrodeposits are highly ductile, so that parts to be formed or stamped may be plated before these operations. The chief disadvantage of cadmium plating compared with zinc plating is its higher cost. Items commonly electroplated with cadmium include nails, screws, rivets, bolts, nuts, washers, fasteners, springs, electrical contacts, carburetor and magneto parts, television and radio parts, and miscellaneous parts for a wide variety of products, including aircraft, ordnance, and automobiles.

Cadmium-base bearing metals are used in internal-combustion engines for service under high pressures and temperatures and at high speeds. The bearing alloys are generally of 2 types—the cadmium-nickel bearing, having the general composition of 98.65 cadmium and 1.35 percent nickel, and the cadmium-silver bearing, containing 0.2 to 2.25 percent silver, 0.25 to 2 percent copper, and the balance cadmium. Such bearings were formerly in much greater usage than at present. During World War II this use was the second largest one for cadmium, exceeding 1,000,000 pounds annually for several years. Currently the use of cadmium in bearings is relatively small.

Cadmium forms solders in combination with varying proportions of such metals as copper, lead, tin, zinc, and silver. The most widely used have been the cadmium-silver solders. Solders of cadmium and base metal, containing little or no tin, were developed during World War II because of the scarcity of tin.

Relatively small quantities of cadmium metal are used with lead, bismuth, and tin to make low-melting-point alloys for fire-detection apparatus, fusible elements in automatic sprinkler heads, firedoor release links, automatic shutoffs for gas and electric water-heating systems, safety plugs for compressed-gas cylinders, and temperature-controlled safety clutches.

Cadmium alloys quite easily with copper; master alloys containing up to 50 percent are added to copper and bronze. Low-cadmium copper (0.7 to 1 percent), is ductile and has found wide use as trolley wire owing to its high strength, high conductivity, and resistance to wear. An alloy of copper-zirconium-cadmium, also used for power-transmission lines, is superior in strength and hardness to copper-cadmium alloys.

Compounds of cadmium have a wide variety of uses. Cadmium sulfide and cadmium sulfoselenide are standard agents for imparting high-quality yellow and red colors, respectively, to paint, enamels, soap, rubber, glass, ceramic glazes, paper, textiles, artists' colors, luminescent colors, leather, printing ink, and other products. Virtually all cadmium oxide and hydrate produced are used in electroplating solutions. Cadmium bromide, chloride, and iodide are used in photographic films, process engraving, and lithographing.

Another use of cadmium, potentially large, is the nickel-cadmium alkaline storage battery. If plans of the companies producing these batteries materialize, large quantities of cadmium will be used for this purpose within a few years. Some reported advantages of nickel-cadmium batteries are: Long life (15 to 20 years (*9*);[2] over 3,000 cycles of charge and discharge (*11*)); no damage by overcharging, reverse charging, or short circuits; simple maintenance; delivery of maximum current with minimum voltage drop; high discharge rate without damage to cells; quick charging; and wide range of operating temperature (65° to 165° F). The cost of a nickel-cadmium battery is estimated at 4 to 5 times that of a comparable lead-acid battery. Cadmium-nickel batteries, are used at present for heavy-duty purposes, such as in buses, diesel locomotives, and other heavy machinery. Test batteries are being used in automobiles but not yet extensively.

In addition to its major uses, cadmium occupies an important place in nuclear physics, where it is employed to control the fissionable elements in reactors. In one type of reactor (*6*), cadmium is used as a coating on graphite and in the form of rods, which absorb thermal neutrons, thus controlling the fission of U–235 atoms.

A possible future use of cadmium is in the conversion of sunlight into energy. A solar generator using cadmium sulfide processed into a thin crystal was developed by the Air Force in 1954 and found practical for many purposes.

SUBSTITUTES

The need for substitutes for cadmium is not foreseen in the near future. Should such a situation develop, however, considerable substitution is possible. Although cadmium is desirable in plating for reasons previously stated, it can be partly replaced by zinc, and, to a smaller extent, by chromium-nickel. During World War II about 15 percent of the cadmium supply was conserved by substituting zinc. Loss of cadmium would not involve serious hardships in many instances, as products commonly electroplated include bolts, nuts, washers, hardware, household appliances, toys, and a great variety of other items where the use of cadmium is not indispensable. Tin and bismuth can be substituted in fusible alloys. In pigments, chromates of lead, zinc, and barium can in some instances replace cadmium yellow, but they are not as desirable.

OCCURRENCE

Cadmium minerals are very rarely found in commercial quantities, hence no ore is mined solely to recover cadmium. The most common mineral, greenockite (CdS, 77.8 percent cadmium), occurs chiefly as a yellow stain or coating on the zinc sulfide, sphalerite. Cadmium metal is recovered almost entirely from zinc

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

sulfide ores or other sulfide ores, principally lead and copper, containing zinc mineralization. Except for the orebodies at Franklin and Sterling Hill, N. J., where only traces of cadmium are found, nearly all of the major zinc deposits in the United States contain cadmium in varying quantities. Zinc concentrates containing as much as 1 percent cadmium are unusual; in general, the content averages less than 0.5 percent. Zinc concentrates from the Tri-State area average about 0.35 percent cadmium, and concentrates from mines in the Western States rarely carry more than 0.25 percent cadmium. It is possible that deposits containing cadmium in sufficient quantities to permit exploitation primarily for cadmium values or for recovery of cadmium along with zinc on a coproduct basis may be discovered.

RESERVES

There are no commercial reserves of cadmium ore. Since cadmium is recovered only as a byproduct in connection with the smelting and refining of zinc ores or zinc-bearing ores, cadmium reserves depend upon the size of zinc ore reserves and the cadmium content of those reserves.

An estimate by the Federal Geological Survey in 1944 placed the recoverable cadmium in domestic reserves of zinc ore of all grades (measured, indicated, and inferred) at 100,000,000 pounds (*12*). Estimates of zinc-ore reserves in 1950 (*4*) indicate recoverable cadmium in quantities of approximately the same magnitude as in 1944.

Analyses by the War Production Board (Zinc Division) in the Tri-State district during World War II indicated an average recovery of about 7.4 pounds of cadmium per ton of recoverable zinc. Studies on the yield of cadmium from western ores showed that, in many instances, they yielded as much as the Tri-State ores; in general, however, the yield was smaller. The plants in the Eastern States showed a recovery of less than 2 pounds of cadmium per ton of zinc. The average yield of all domestic ores was estimated at about 4.25 pounds of recoverable cadmium per ton of recoverable zinc. Production records, however, particularly in recent years (1949–53), indicate a recovery of about 5.3 pounds of cadmium per ton of recoverable zinc.

METALLURGY

The entire domestic supply of primary cadmium is recovered from the flue dusts of zinc-blende roasting and sintering furnaces and of copper- and lead-smelting plants, from zinc dust collected in the early stages of distillation in zinc retorts and from the high-cadmium precipitate obtained in purifying zinc electrolyte at electrolytic zinc plants. Other minor metals, principally germanium, thallium, and indium, are also found in zinc-plant flue dusts and are commonly recovered as byproducts.

Zinc-Retort Plants.—Early cadmium recovery was almost exclusively from the dust or "blue powder" collected in the secondary condenser or "prolong" of horizontal zinc retorts. The process employed was very inefficient, giving low metal yields and requiring considerable expenditure of labor. The present emphasis in zinc-retort plants is on recovery from the roasting operations and from the zinc metal itself, rather than from the blue powder produced in the retorting operations.

Basically, the process for recovering cadmium from roasting zinc ores and concentrates comprises the following steps: (1) Roasting the zinc concentrates; (2) sintering the resulting calcines and dust along with cadmium furnace residues, a mixture of zinc chloride and zinc sulfate liquor, and fuel; (3) leaching the Cottrell dust from the sintering machines; (4) precipitating the cadmium as a sponge from the zinc-cadmium solution by adding of zinc dust; and (5) distillation of the prepared sponge in a horizontal, batch-type retort, yielding cadmium metal, blue powder, and residue.

In the New Jersey Zinc Co. vertical-retort process for producing zinc by fractional distillation, cadmium is recovered. Employing the principles of rectification in this distillation, high-boiling-point impurities such as lead and low-boiling-point impurities such as cadmium are separated from relatively impure zinc.

Molten zinc, which constitutes the feed for the fractionating columns in this process, contains both lead and cadmium. Lead has a boiling point of about 1,620° C., zinc 905° C., and cadmium 767° C. Since these elements do not form constant-boiling-point mixtures or stable compounds with each other that vaporize without dissociation, they lend themselves to separation by fractional distillation.

The New Jersey process combines the initial bulk distillation in a vertical retort and purification of the crude metal in fractionating columns where the cadmium is recovered. In purification, the molten crude metal is charged to a lead rectification column where, by a series of fractionations, zinc and cadmium are distilled off lead-free and condensed, while a high-lead-zinc alloy is drawn off at the bottom of the column. The molten zinc-cadmium condensate from the lead column is run into a cadmium rectification column where the cadmium is distilled and condensed. The cadmium is then purified and cast into commercial shapes.

Refined liquid zinc is continuously drawn off from the bottom of this column.

Electrolytic Plants.—The electrolytic production of cadmium first assumed importance in the United States about 1915 as a result of introduction of the electrolytic zinc process, inasmuch as the removal of cadmium from the zinc electrolyte is a necessary purification step before electrolysis. The high-cadmium precipitate obtained in this purification, as well as baghouse fume from lead and copper smelters, is processed in electrolytic cadmium plants.

In the electrolytic zinc process, cadmium and copper are taken into solution with the zinc in the sulfuric acid leach of the roasted zinc ore. These elements are interfering impurities and must be removed before the leach liquor can be sent to the zinc cells. Zinc dust is added to precipitate many of the objectionable elements. Precipitation is usually conducted in two steps. In the first, a limited quantity of zinc dust is used to give a high-copper precipitate, and in the second an excess of dust is added to the filtrate from the first step, yielding a zincky high-cadmium precipitate, which constitutes the feed for the cadmium plant.

Copper- and Lead-Smelting Plants.—An important source of cadmium is the baghouse dust from copper and lead smelters operating on zincky ores containing cadmium. Recovery is effected by a variety of methods.

In one process the burned baghouse dust of the lead blast furnace is refumed and a concentrated cadmium dust recovered and sent to an electrolytic plant for further treatment.

In another method baghouse dust is concentrated by fuming in a reverberatory furnace. The concentrated fume is acid-leached and the cadmium precipitated from the pregnant leach liquor with zinc. The sponge metal so obtained is distilled to recover metallic cadmium.

By another process, which is applicable to both zinc- and lead-smelter products, pulped baghouse fume is treated with chlorine gas and the solubilized cadmium precipitated by zinc dust. The precipitated cadmium sponge may be separated by filtering or centrifuging and electrolyzed after solution in sulfuric acid, or it may be dried in the absence of free oxygen and distilled.

Lithopone Residues.—Another source of cadmium is the zinc-cadmium sludge resulting from purification of zinc sulfate solutions used in manufacturing zinc lithopone. In lithopone production, elements yielding colored compounds must be removed from the zinc sulfate solution. A purification similar to that followed in the treatment of electrolytic zinc electrolyte is followed. Roasted zinc ore is leached with a sulfuric acid solution and the cadmium precipitated in the usual way with zinc dust.

SUPPLY—DISTRIBUTION

Cadmium has been in oversupply in the United States for the past 3 years (1952–54). Primary metal production at domestic plants during this period totaled about 27,300,000 pounds, primary compounds about 300,000 pounds, and net imports approximately 2,200,000 pounds. Supplies in 1952–54 thus totaled nearly 30,000,000 pounds and exceeded apparent consumption of about 25,700,000 pounds. Producers' stocks of primary metal totaled about 4,500,000 pounds at the end of 1954, or more than 6 months' supplies at the average annual rate of consumption prevailing in 1952–54.

Production.—The United States is the world's leading producer of cadmium; for 1940–53 production at domestic plants made up approximately two-thirds of the total world output. The United States production, however, is not solely of domestic origin, as it is estimated that about 60 percent of the primary cadmium metal produced in the United States is recovered from imported flue dust and imported zinc ores and concentrates. Because the geographic origin of the metal produced at United States plants cannot be determined accurately, the entire output is presented as domestic production.

Metallic cadmium was first made in the United States in 1906, and by 1917 the United States had become the world's largest producer. Production increased from 207,000 pounds in 1917 to 2,481,000 in 1929 and by 1939 was at a rate of nearly 4.5 million pounds a year. Beginning in 1940 and continuing through 1953, primary production of cadmium metal was at a very high level; the average yearly output for the 14 years was 7,860,000 pounds. In 1953 production reached an alltime high of 9,682,000 pounds, a 9-percent increase over the previous record established in 1950.

In addition to metal production, considerable quantities of cadmium compounds made directly from primary materials and of secondary metal and compounds are produced in the United States and contribute to the total available supply of cadmium. For 1940–53 production in these 2 categories—primary compounds and secondary metal and compounds (not produced from metal)—averaged 241,000 and 213,000 pounds a year, respectively.

Table 1 shows the total United States supply of cadmium for 1925–53.

Consumption.—No statistics are available on cadmium consumption in the United States before 1929. Domestic usage probably closely approximated output. Apparent consumption increased from 3,129,000 pounds in 1929 to 5,900,000 pounds in 1939, with an intervening drop in 1931–33 resulting from the general

TABLE 1.—*Cadmium supply of the United States, 1925–53*

[Thousand pounds]

Year	Production			Imports (metal)	Total
	Primary metal	Primary compounds [1]	Secondary [2]		
1925	503	113	[3] N. A.	(4)	616
1926	810	167	[3] N. A.	----------	977
1927	1, 075	229	[3] N. A.	22	1, 326
1928	1, 876	240	[3] N. A.	233	2, 349
1929	2, 482	433	[3] N. A.	214	3, 129
1930	2, 778	316	[3] N. A.	40	3, 134
1931	1, 051	337	[3] N. A.	(4)	1, 388
1932	799	260	[3] N. A.	----------	1, 059
1933	2, 277	401	[3] N. A.	109	2, 787
1934	2, 777	567	[3] N. A.	126	3, 470
1935	3, 477	508	[3] N. A.	185	4, 170
1936	3, 633	627	[3] N. A.	576	4, 836
1937	4, 266	419	[3] N. A.	828	5, 513
1938	4, 078	216	[3] N. A.	23	4, 317
1939	4, 411	379	(5)	310	5, 100
1940	6, 154	209	228	28	6, 619
1941	6, 938	296	379	147	7, 760
1942	7, 323	48	316	53	7, 740
1943	8, 396	71	162	49	8, 678
1944	8, 453	326	107	67	8, 953
1945	7, 933	451	72	29	8, 485
1946	6, 200	271	355	18	6, 844
1947	8, 007	501	105	20	8, 633
1948	7, 583	193	121	10	7, 907
1949	8, 024	203	384	157	8, 768
1950	8, 850	341	427	630	10, 248
1951	8, 114	197	168	90	8, 569
1952	8, 388	179	80	1, 479	10, 126
1953	9, 682	85	70	1, 555	11, 392

[1] Cadmium content. Beginning in 1937, excludes compounds made from metal.
[2] Metal and compounds (cadmium content) not produced from metal.
[3] Not available.
[4] Less than 1,000 pounds.
[5] Included with primary compounds.

depression. Stimulated by the national defense program, consumption rose to 7,766,000 pounds in 1941, and because of the great demand for military purposes increased during World War II to average 8,137,000 pounds annually for the period 1942–45. Apparent consumption of primary cadmium slumped to 6,984,000 pounds in 1946, then increased to an average of 7,670,000 pounds a year for 1947–49. In 1950 apparent consumption increased substantially to 9,545,000 pounds, chiefly because of the large military demand generated by the war in Korea. Consumption of cadmium in 1951 was curtailed enough to enable available supplies to meet all military and essential civilian requirements, despite a 24-percent decrease in total supply. Domestic utilization declined to 7,173,000 pounds owing largely to restrictions placed on end uses by the National Production Authority. In 1952 consumption of primary cadmium increased to 9,042,000 pounds, and in 1953 a new high of 9,627,000 pounds was attained.

Apparent consumption of primary cadmium is computed by adding primary production and net imports and adjusting for producers', distributors', and compound manufacturers' stock changes. Except for 1940–44, all figures presented here are calculated apparent consumption. During 1948–53 apparent consumption included large purchases of cadmium for the National Stockpile, hence real domestic consumption for those years was considerably less than the apparent quantity consumed.

In addition to primary consumption, shipments of secondary metal and compounds are included in total distribution. For 1945–53 these shipments averaged 196,000 pounds a year compared with secondary production of 198,000 pounds.

United States distribution of cadmium from 1929 through 1953 is shown in table 2.

TABLE 2.—*Distribution ot cadmium in the United States, 1929–53*

[Thousand pounds]

Year	Apparent consumption (primary)	Secondary shipments	Exports (metal)	Total distribution
1929	3, 129	[2] N. A.	----------	3, 129
1930	3, 134	[2] N. A.	----------	3, 134
1931	1, 388	[2] N. A.	----------	1, 388
1932	1, 059	[2] N. A.	----------	1, 059
1933	2, 787	[2] N. A.	----------	2, 787
1934	3, 470	[2] N. A.	----------	3, 470
1935	4, 169	[2] N. A.	----------	4, 169
1936	4, 836	[2] N. A.	----------	4, 836
1937	5, 652	[2] N. A.	----------	5, 652
1938	4, 100	[2] N. A.	458	4, 558
1939	5, 900	[2] N. A.	----------	5, 900
1940	[1] 6, 178	(3)	387	6, 565
1941	[1] 7, 766	(3)	172	7, 938
1942	[1] 7, 659	(3)	284	7, 943
1943	[1] 7, 381	(3)	157	7, 538
1944	[1] 8, 865	(3)	548	9, 413
1945	8, 643	68	102	8, 813
1946	6, 984	361	140	7, 485
1947	7, 727	135	303	8, 165
1948	7, 797	121	956	8, 874
1949	7, 486	385	566	8, 437
1950	9, 545	427	353	10, 325
1951	7, 172	88	606	7, 866
1952	9, 042	123	[4] 301	9, 466
1953	9, 627	60	[4] 66	9, 753

[1] War Production Board consumption survey.
[2] Not available.
[3] Included with primary consumption.
[4] Includes metal, dross, flue dust, residues, scrap and alloys.

Foreign Trade.[3]—Approximately 60 percent of the primary metallic cadmium produced in the United States is from foreign materials but is classed as domestic output because of lack of data for separating the cadmium recovered from imported ores and concentrates by country. The cadmium content of imported flue dust is recorded; but the content of imported zinc ores and concentrates, from which approximately 60 percent of the cadmium of foreign origin is obtained, is not available.

Almost all of the imported flue-dust-bearing cadmium and a large proportion of the zinc ores and concentrates come from Mexico. Canada, Peru, and South-West Africa are other important sources of cadmium-bearing raw materials. Table 3 shows imports of flue dust for 1936–53.

United States imports of metallic cadmium have been a relatively unimportant part of the total supplies of cadmium available for consumption, except for 1952–53, when large quantities of metal were imported. Before

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[3] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

TABLE 3.—*Imports of cadmium flue dust (cadmium content), 1936-53*[1]

[Pounds]

Year	Mexico	Others	Total
1936	1, 200, 000		1, 200, 000
1937	1, 366, 000		1, 366, 000
1938	1, 476, 000		1, 476, 000
1939	1, 704, 000		1, 704, 000
1940	1, 891, 000		1, 891, 000
1941	1, 984, 000	337, 000	2, 321, 000
1942	1, 724, 000		1, 724, 000
1943	1, 643, 000		1, 643, 000
1944	1, 689, 000		1, 689, 000
1945	2, 193, 000		2, 193, 000
1946	1, 609, 000	44, 000	1, 653, 000
1947	2, 356, 000		2, 356, 000
1948	1, 828, 000		1, 828, 000
1949	1, 787, 000	3, 000	1, 790, 000
1950	1, 602, 000		1, 602, 000
1951	1, 607, 000		1, 607, 000
1952	1, 985, 000	7, 000	1, 992, 000
1953	1, 864, 000		1, 864, 000

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[1] Data not available before 1936.

World War II the largest quantity imported was 829,000 pounds in 1937, 18 percent of the total supply. Imports averaged 108,000 pounds a year for 1940–51, but during this period exports of metal averaged 381,000 pounds annually, more than offsetting any addition to supply by imports. Owing to increased world supplies, imports rose to 1,479,000 pounds in 1952 and in 1953 established a new record of 1,555,000 pounds. Exports totaled 301,000 pounds in 1952 and 66,000 pounds in 1953.

The tariff on imported cadmium metal is 3¾ cents per pound as established in the Geneva Trade Conference of 1947. Cadmium in flue dust is duty free.

United States imports and exports of cadmium metal are given in tables 1 and 2, respectively.

MARKETING AND PRICES

Cadmium is commonly sold as sticks, bars, balls, and, to a smaller extent, ingots. Much metal is also marketed as anodes, or special platers' shapes, which command a premium of approximately 5 to 10 percent above the price for regular shapes. Quotations are generally on a delivered basis.

Cadmium is a comparatively expensive metal; over the period 1925–52 prices ranged from 10 to 20 times above that of zinc. The selling price of commercial cadmium sticks increased over 400 percent during 1939–51; the quoted average in 1939 was 59 cents a pound and in 1951 $2.55 a pound. During World War II the price was fixed at $0.90 a pound by the Office of Price Administration. After controls were removed in 1946, the price of cadmium advanced steadily until December 1950, when it reached $2.55 a pound. This price remained in effect until May 1952, when it dropped to $2.25. In August the quotation fell to $2.00 a pound, and in December one domestic producer quoted commercial sticks at $1.50 a pound but subsequently raised the quotation to $1.75 and shortly thereafter to $2.00. Other sellers adhered to the $2.00 per pound basis throughout December. In 1953 the price remained unchanged at $2.00. The quotation for sticks was reduced to $1.70 on February 1, 1954, and, concurrently, the quoted price for special platers' shapes was lowered to $1.75 and on June 21 further reduced to $1.70.

There are no established purchase schedules for cadmium contained in ores and concentrates. Custom smelters, as a general practice, do not pay shippers of zinc and/or lead ores and concentrates for contained cadmium. Only a few companies have cadmium toll agreements with the smelter. During periods of competitive bidding, however, smelters have occasionally paid for the contained cadmium.

SELF-SUFFICIENCY AND STRATEGIC CONSIDERATIONS

The United States is not self-sufficient in cadmium; approximately 60 percent of the primary metal produced at domestic plants is of foreign origin. Mexico is the largest supplier of cadmium raw materials, followed by Canada and, to a smaller extent, Peru and South-West Africa; therefore, while less than half of the United States output is from domestic materials, the bulk comes from readily accessible sources.

The apparent consumption of primary cadmium averaged 7,991,000 pounds a year from 1940 through 1953. This figure includes large purchases of cadmium for the Government stockpile during 1948–53, hence real consumption was somewhat less than the above figure. For the same period production of primary metal and compounds averaged 8,101,000 pounds annually.

In a prolonged period of semimobilization or even allout war, it is believed that military requirements and essential civilian needs of cadmium could be met. Supplies could be conserved by substituting less critical materials and by imposing restrictions on nonessential uses. Such restrictions might not even proportionately decrease production of nonessential end products, because of the possibility of widespread substitution of other plating materials for cadmium. Based on these factors, on domestic zinc reserves and output, on the relative safety of the principal foreign sources, on high existing stocks, and on the current

National Stockpile and planned purchases for it, the United States is apparently in a relatively secure position for an all-out emergency. Since Mexico and Canada are the principal sources of foreign raw materials, the United States could count on enough cadmium to satisfy minimum essential needs, even if all other foreign supplies were cut off; with tight controls on consumption, the United States might be able to get along on domestic resources alone for several years.

GOVERNMENT PROGRAMS

The Government regulated the cadmium industry during World War II and again in 1951–52, to provide adequate supplies for all essential purposes. Cadmium was on the critical metals list throughout the entire war period.

The task of directing available supplies of cadmium into defense channels was undertaken by the Office of Production Management in 1941. In January 1942, cadmium was placed under full priority control by Order M–65. Companion order M–65–a prohibited the use of cadmium in automotive, trailer, and tractor equipment, building supplies and hardware, house furnishings and equipment, and a number of miscellaneous products. The War Production Board amended M–65 in June to provide for direct Government allocation. A ceiling price of 90 cents a pound on cadmium was established by the Office of Price Administration in January 1942. In July 1943 WPB issued another conservation order to prohibit the use of cadmium in pigments, except for specific purposes, most of which were military. Control by Government agencies—WPB and its successor, the Civilian Production Administration—was in effect in varying degrees until July 1946, when cadmium was removed from price control.

The Government maintained a stockpile of cadmium for emergency use from 1941 through 1947. In 1948 all remaining stocks were transferred to the National Strategic Stockpile, and the Government began purchasing cadmium for this reserve.

On January 1, 1951, the National Production Authority to insure adequate supplies of cadmium for military uses and essential civilian products for which there is no satisfactory substitute, imposed inventory controls and limited the purposes for which cadmium, cadmium-containing items, and cadmium-plated items could be produced. The cadmium conservation order, M–19, was amended on March 16, April 26, and July 30, 1951, and again on March 13, 1952, relaxing restrictions on the use of cadmium to permit its increased use in a wide range of military and civilian products. M–19 was amended because inventories of the metal had been accumulating in the hands of producers, pending placement of a larger volume of defense orders. The cadmium order was revoked May 15, 1952, partly because military orders were not as large as had been anticipated and partly because of the greatly increased availability of the metal because of record imports.

STRUCTURE OF CADMIUM INDUSTRY

The United States cadmium industry can be considered as being composed of four segments: Primary producers, secondary producers, distributors, and consumers. On the production side, primary producers are of preponderant importance, supplying over 95 percent of the total annual output.

Primary cadmium production is closely related to and largely dependent upon activities in the zinc industry and to a much smaller extent, the lead and copper industries. Cadmium is recovered almost exclusively from zinc operations as a highly valuable byproduct. The production of cadmium is an integral part of the zinc industry; but, as the cadmium is not comparable in value to the zinc produced, recovery of cadmium is not on a coproduct basis and from the standpoint of the zinc industry was secondary importance. This is not meant to infer that the cadmium industry is unimportant but merely to emphasize its relationship to the zinc industry.

The primary producers of cadmium are for the most part large zinc producers. Eleven companies produce primary metallic cadmium at 13 plants in the United States. The largest producer is American Smelting & Refining Co. Other large producers include Anaconda Copper Mining Co., American Zinc Co. of Illinois, Sullivan Mining Co., National Zinc Co., Inc., The Eagle-Picher Co., St. Joseph Lead Co., and The New Jersey Zinc Co. Bunker Hill & Sullivan Mining & Concentrating Co., American Steel & Wire Division of the United States Steel Corp., and the Sherwin-Williams Co. are the only other producers of primary cadmium metal in the United States.

Several zinc- and lead-producing plants that do not produce refined cadmium have facilities for collecting cadmium fume, dust, sponge, or residues. These companies include American Zinc Co. of Illinois, Monsanto, Ill., and Dumas, Tex.; Matthiessen & Hegeler Zinc Co., La Salle, Ill.; American Smelting & Refining Co., Amarillo, Tex., and Alton, Ill.; Blackwell Zinc Co., Blackwell, Okla.; Athletic Mining & Smelting Co. and the Residue Co., both at Fort Smith, Ark.; Hegeler Zinc Co., at Danville,

TABLE 4.—*Primary cadmium production capacity in the United States in 1953*

Company	Plant location	Annual capacity (thousand pounds)[1]
American Smelting & Refining Co.	Denver, Colo	(2)
	Corpus Christi, Tex	(2)
American Zinc Co. of Illinois	Fairmont City, Ill. (East St. Louis).	2,000
Anaconda Copper Mining Co	Great Falls, Mont	2,000
Bunker Hill & Sullivan Mining & Concentrating Co.	Bradley, Idaho	365
Eagle-Picher Co	Henryetta, Okla	400
National Zinc Co., Inc	Bartlesville, Okla	720
New Jersey Zinc Co	Depue, Ill	(2)
St. Joseph Lead Co	Herculaneum, Mo	300
	Josephtown, Pa	300
Sherwin-Williams Co	Coffeyville, Kans	(2)
Sullivan Mining Co	Kellogg, Idaho	600
United States Steel Corp. (American Steel & Wire Div.).	Donora, Pa	(2)
Total (approximate)		12,000

[1] Largely estimated; rated capacity is flexible, depending to a great extent on the quantity and grade of raw materials available.

[2] Concealed to avoid disclosure of individual plant capacity.

Ill.; The New Jersey Zinc Co., Canon City, Colo., and Palmerton, Pa.; International Smelting & Refining Co. at Tooele, Utah; and United States Smelting, Refining & Mining Co., Midvale, Utah. These companies either sell the cadmium-bearing materials to the cadmium producers, have it treated for them on a toll basis, or transfer it to other company-owned plants that produce cadmium metal.

Two companies, the American Smelting & Refining Co. and Harshaw Chemical Co., produce relatively small quantities of cadmium compounds from primary raw materials at their plants in Denver, Colo., and Elyria, Ohio, respectively. Being of primary origin, such production is an addition to the new supply.

Recovery of cadmium metal and compounds from scrap materials constitutes a comparatively minor segment of the cadmium industry. Several companies produce the metal, principally from old bearings and other alloys, and compounds from secondary residues. Recovery of the metal and production of compounds (not produced from secondary metal) is an addition to the total supply of cadmium. Following is a list of producers and plants:

Company	*Plant*	*Product*
Arkansas Metals Co	Jonesboro, Ark	Secondary metal.
National Zinc Co., Inc	Bartlesville, Okla	Do.
Neo-Smelting & Refining, Inc	Whitestone, N. Y	Do.
Harshaw Chemical Co	Elyria, Ohio	Secondary compounds.
Chemicals-Pigments-Metals Division of the Glidden Co.	Baltimore, Md	Do.

The middlemen in the cadmium industry are the distributors. They buy the metal from the primary producers and sell to consumers. Cadmium distributors in the United States are divided into two general groups: (1) The companies that, for the most part, produce plating salts, supply special platers' shapes as well as regular grades of cadmium, and provide technical service for their customers who are chiefly electroplaters; and (2) buyers of domestic and imported cadmium for resale.

The importance of distributors in the cadmium industry is indicated by the volume of transactions. In 1952 over half of total shipments to consumers were by dealers, but imports were exceptionally high that year. Normally dealers account for about one-third of total cadmium sales.

There are literally thousands of consumers of cadmium in the United States, most of whom use relatively small quantities of the metal. The largest group of consumers use the metal for electroplating various manufactured products. Bearing manufacturers, pigment producers, and soldermakers are the other principal consuming industries.

Of the chief consumers, only in the production of pigments is consumption of cadmium concentrated among a few companies. Chemicals-Pigments-Metals Division of the Glidden Co., Baltimore, Md.; Harshaw Chemical Co., Elyria, Ohio; and Kentucky Color & Chemical Co., Louisville, Ky., use the bulk of the metallic cadmium consumed in pigment making. American Smelting & Refining Co. produces large quantities of cadmium sulfide directly from raw materials. In the other major consuming groups, particularly electroplating, there is no apparent concentration of consumption among a few companies; a large number of consumers use quantities ranging from less than 100 pounds to 100,000 a year.

WORLD PRODUCTION

The United States is by far the world's largest producer of cadmium and supplies about two-thirds of the total annual world output. United States production is not solely from domestic sources, of course; output from domestic sources constitutes about 25 to 30 percent of the world total.

Other countries producing cadmium are, with one exception, leading zinc producers. South-West Africa, though not a large zinc producer, ranks among the largest cadmium producers by virtue of the high cadmium content of the ores mined. South-West Africa does not produce refined metal; lead-zinc-copper concentrates containing large quantities of cadmium are exported, principally to the United States, the United Kingdom, Belgium, and France, where the metals, including cadmium, are recovered.

Mexico, Canada, and Australia, all large zinc producers, are also important cadmium producers. Virtually all Mexican cadmium is exported in flue dust and zinc concentrates, chiefly to the United States. Canada and Australia produce refined cadmium metal. Belgium and the United Kingdom, working on imported materials, are also significant cadmium producers. Japan, Norway, and Italy, operating on domestic materials, and France, treating imports, produce sizable quantities of cadmium. Germany formerly recovered large quantities of cadmium from the cadmium-rich zinc ore of Upper Silesia, but with the cession of that territory to Poland after World War II, Germany has been less important as a source of cadmium. Table 5 shows free-world production of cadmium for 1940–53.

TABLE 5.—*Free-world production of primary cadmium metal, by principal countries, 1940–53*

[Quantities in thousand pounds]
(Compiled by Berenice B. Mitchell)

Country	1940	1941	1942	1943	1944	1945	1946	1947	1948	1949	1950	1951	1952	1953
Australia	386	430	366	353	560	493	494	422	647	582	659	517	646	658
Belgium	[1] N. A.	51	89	70	2	[1] N. A.	196	190	348	326	805	992	1, 213	1, 323
Canada	908	1, 251	1, 149	787	527	646	803	719	766	846	848	1, 327	949	1, 316
France	220	44	22	22	12	15	104	95	110	128	158	187	195	220
Germany	653	820	536	608	461	[1] N. A.	2	3	8	11	[1] N. A.	154	141	227
Italy	474	406	271	158	86	30	55	85	103	163	165	450	293	401
Japan	[1] N. A.	[1] N. A.	225	247	187	49	17	19	66	116	199	259	367	397
Mexico [2]	1, 798	1, 999	1, 883	1, 768	1, 504	2, 321	1, 581	1, 715	1, 995	1, 808	1, 519	1, 969	1, 618	2, 114
Norway	63	55	30	25	14	26	62	111	137	157	174	221	163	165
South-West Africa [3]	87	497	--------	--------	--------	--------	--------	--------	1, 140	1, 662	1, 344	1, 434	1, 112	1, 194
United Kingdom	404	328	342	354	468	394	238	218	249	226	262	326	347	380
United States	6, 154	6, 938	7, 323	8, 396	8, 453	7, 933	6, 200	8, 007	7, 583	8, 024	8, 850	8, 114	8, 388	9, 682

[1] Not available.
[2] Cadmium content of flue dust exported for treatment elsewhere.
[3] Cadmium content of concentrates exported for treatment elsewhere.

OUTLOOK

It is difficult to forecast with any assurance the supply-demand picture for cadmium over an extended period. The most logical assumption, however, is that supply and demand will be in virtual balance during the ensuing decade, with probable increases in both. An increase in demand is predicated upon increased military requirements in a state of partial mobilization and greater civilian usage because of such factors as population growth and continuing high industrial activity. Increased supplies are predicated upon continued high and increasing domestic smelter output of zinc (with a parallel increase in cadmium recovery) and upon high imports.

The immediate or short-term outlook for cadmium shows supplies adequate for all purposes, inasmuch as cadmium has been in oversupply for the past 3 years (1952–54). In the long run (more than 10 years), however, the outlook for balance between demand and supply is less promising; the demand for cadmium is expected to increase somewhat, while domestic production remains relatively stable owing to inelasticity of the industry. Therefore the long-range view is one of tighter supplies, greater dependence on foreign sources of both metal and raw materials, and possible shortages.

United States cadmium production, in all probability, will be largely adequate to meet the expected demand during the next 10 years. The possibility of increasing domestic output of primary metal and primary compounds (not produced from metal) above the record of 9,682,000 pounds established in 1953 depends directly on the magnitude of domestic smelter production of zinc, which in turn depends upon domestic mine production of zinc and imports of zinc ores and concentrates.

Because of the mineralogical relationship of cadmium to zinc, virtually the only means of increasing the output of cadmium is to increase the mine output of zinc and/or to achieve greater efficiency in cadmium recovery. It appears unlikely that domestic zinc production will be expanded enough to provide any substantial increase in cadmium output. Furthermore, the possibility of discovering ore bodies containing cadmium as the principal economic constituent is remote. Moreover, the declining importance of the Tri-State district might cause a decrease in cadmium recovery, inasmuch as the Tri-State ores are richer in cadmium than most other domestic ores. It appears therefore that any increase in domestic cadmium output will depend largely upon the quantity of zinc ores and concentrates imported. If imports are increased appreciably, a substantial increase in cadmium output can be expected.

Although the quantity of cadmium recovered per ton of zinc has increased markedly during the past 15 years owing to installation of more cadmium-recovery facilities, the prospect of a further substantial increase is considered unlikely. Such increases as will come from installation of additional recovery units and from improving the efficiency of existing units will probably not exceed 10 to 15 percent of present production and, moreover, could largely be offset by the decline in the cadmium content of domestic ores.

It is conceivable that sizable quantities of cadmium could be recovered from zinc metal. Much of the Prime Western, Select, Brass Special, and Intermediate grades of slab zinc contains substantial quantities of cadmium. The tendency toward increased use of the higher purity grades of zinc will reduce the demand for Prime Western and lower purity grades in favor of the higher grades, and thus tend to increase the recovery of cadmium, inasmuch as cadmium recovery is higher at plants producing high-grade zinc than at horizontal retorts.

Since most uses of cadmium are dissipative, the cadmium supply cannot be augmented to any appreciable extent by greater recovery from secondary sources. It is unlikely that secondary production will exceed 5 percent of the total supply.

World production of zinc and, owing to its direct byproduct relationship to zinc, cadmium in 1953 was the highest on record. The world's potential to produce zinc and cadmium was greater than ever before in 1953; moreover, actual as well as potential production of zinc, and consequently cadmium, are expected to rise materially within the next 10 years. Countries in which cadmium production will probably increase substantially include Canada, Peru, Belgian Congo, and North Africa.

A potentially large use of cadmium is for nickel-cadmium storage batteries. Standard European-type batteries contain approximately 7 pounds of cadmium, but certain models (suitable for automobiles) containing from about 1.5 to 3 pounds have been manufactured in the United States. The ultimate utilization of cadmium in batteries is difficult to predict, but if manufacturers' plans materialize this use will become large. Even if nickel-cadmium batteries finds wide consumer acceptance, volume production will ultimately be limited by availability of supplies.

A new and relatively minor use of cadmium is in nuclear physics, where the metal is employed to control the fissionable elements in reactors. A possible use of cadmium is in conversion of sunlight into energy. A solar generator using cadmium sulfide processed into a thin crystal form has recently been developed and found practical for many purposes.

PROBLEMS

Among the basic problems of the cadmium industry are elasticity of demand and inelasticity of supply. During emergency periods of high military spending, the demand for cadmium is very large, inasmuch as cadmium has many military applications. Owing to its byproduct relationship to other metals, particularly zinc, production can be expanded only in proportion to expansion of zinc production. During World War II, for instance, and again during the Korean War, output of cadmium was inadequate for both military and civilian needs. Consequently during such emergency periods control measures were introduced to limit civilian usage and insure large enough supplies for military purposes, while nonmilitary uses therefore were obliged to be met with substitutes. When military buying slackened and cadmium was again freely available, civilian users were reluctant to change back to cadmium partly because they feared recurring shortages and partly because they found that the substitutes employed were acceptable. Large imports during periods of easy supply and decreased imports during periods of scarcity are deleterious to producers and consumers alike and tend to accentuate the basic problem of the industry.

The ever-present knowledge among cadmium consumers that cadmium is one of the rarer metals and is apparently destined to remain relatively scarce is the chief factor limiting substantial expansion in use. The relatively high price of cadmium also acts as a deterrent to more widespread use of the metal.

How to achieve greater efficiency in the recovery of cadmium and thus increase production is an important problem to the cadmium industry and one in which much has been accomplished and more remains to be done. Any increase in cadmium output assuring consumers of greater long-term availability of supplies and lower prices would tend to encourage expansion of established uses and development of new uses.

There is a great need for comprehensive statistics on the consumption of cadmium, by uses. At present the only consumption data available are those collected by the Federal Bureau of Mines, covering shipments to consumers (not classified) and calculated apparent consumption. Consumption data by uses would be of considerable value to industry in market research and analysis and to Government agencies in reaching decisions that affect the industry.

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CEMENT

By

Oliver S. North[1]

THE MANUFACTURE of portland cement, which has been described as the process of "putting a mountain through a sieve," has grown into a colossus in the nonmetallic field. Portland cement concrete is so low-priced, easy to use, and durable that it has become the foundation stone of the construction industry.

Summary

For the sixth consecutive year, a new record was established in 1952, when the value of shipments of domestically produced portland cement was approximately $639 million. Local and sometimes general shortages of a temporary nature have stimulated rapid growth of this industry in recent years.

Portland-cement clinker is formed by burning, at high temperatures in rotary kilns, properly proportioned mixtures containing essentially lime, alumina, silica, and iron oxides. The clinker is cooled and stored. Finished portland cement is made by grinding together intimately cement clinker and a small quantity of gypsum. Finished portland cement may be packed in 1-cubic-foot bags for shipment or shipped in bulk form in trucks, railroad hopper cars, or boats.

Reserves of the raw materials used in the manufacture of portland cement are widely distributed throughout the United States. In some areas supplies are large, whereas in others deposits are smaller, lower in quality, or disadvantageously located from the standpoint of demand, transportation facilities, or other economic factors.

Portland cement competes successfully with other construction materials in many applications.

The technical aspects of portland-cement production and concrete utilization have been intensively studied; however, much remains to be done, and experiments, tests, and studies are being continued by members of the industry, the Portland Cement Association, and other individuals and organizations.

There is need for continued investigations of occurrences of raw materials, particularly in those localities now apparently lacking economic deposits of material considered suitable for portland-cement manufacture, study of the chemistry of portland cement and concrete, improvement of mining and processing methods, and investigation of modern construction practices that utilize large lightweight-concrete units.

Although concrete technology is better understood than ever before, a wide range of unsolved problems still challenges the technologist. The industry has a variety of technical and economic problems and recognizes that their solution needs a strong research program.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

"Cement" may be defined as any adhesive substance capable of uniting or binding together fragments or masses of solid matter into a compact whole. Such a definition embraces a large number of very different substances, including the plastic materials employed to bind aggregates, stone, etc., in the construction of buildings and engineering works. These latter materials usually are spoken of as "hydraulic"—those that have the property of "setting" or hardening under water—or "nonhydraulic"—those that do not set when used under water.

For the purposes of this paper, such materials as lime, Sorel cement, and gypsum plaster are excluded, and the discussion is confined to portland, portland blast-furnace slag, natural, masonry, and puzzolan cements and hydraulic lime.

TYPES OF CEMENT

Portland cement may be defined as the product obtained by calcining to incipient fusion a finely ground artificial mixture containing essentially lime, silica, alumina, and iron oxides in certain definite proportions and grinding the resulting clinker with a small quantity of gypsum. Natural cement and hydraulic lime are produced by calcining, at relatively low temperatures, naturally impure limestones containing approximately the required percentages of lime, silica, alumina, and iron oxides. Slag-lime (also called puzzolan) cement is produced by mixing powdered slaked lime with volcanic ash or blast-furnace slag, generally the latter in the United States; it differs from portland blast-furnace slag cement in that the latter is an intergrind of granulated blast-furnace slag with portland-cement clinker instead of with lime.

Masonry cements (portland-cement base) are produced by intimately intergrinding portland-cement clinker or finished portland cement with limestone and an air-entraining plasticizer to a fineness greater than portland cement. The product has strong hydraulic properties, as well as the high plasticity and water-retention ability essential in mortars for brick or other masonry work. A similar masonry cement, referred to in Bureau of Mines literature as masonry (natural), is produced in essentially the same way, except that natural cement is interground with a small quantity of portland cement. Sometimes a small quantity of limestone is interground in masonry (natural) cement. Also, a masonry cement is compounded of slag-lime (puzzolan) cement and a small proportion of portland cement.

Of these varieties, that known as portland cement has become by far the most widely used owing to the ability of manufacturers to produce a more uniform and dependable product than was possible from the naturally occurring rock alone, even though at higher cost. However, through the years improved processing practices and economies, coupled with such economic factors as high demand in regions deficient or lacking deposits of the natural rock and dependent on local materials as a means of saving high freight costs, have brought the average mill value per barrel of portland cement to a figure approaching that of natural cement.

SIZE AND ORGANIZATION OF CEMENT INDUSTRY

The quantity (in 376-pound barrels) and value of each variety of cement shipped in 1952 was approximately as follows: Portland, 251,370,000 barrels valued at $639 million; natural and puzzolan cements and hydraulic lime, 2.0 million barrels valued at $6.0 million; masonry cement (portland base), 10,775,000 barrels valued at $33.3 million; and masonry cement (natural base), 1.4 million barrels valued at $4 million.

It should be pointed out in connection with the above figures that the marketing unit of portland cement is the 376-pound barrel. On the other hand, nearly all masonry cements, whether portland or natural base, are marketed in barrel units of 276 to 280 pounds. However, for statistical uniformity the quantities of masonry cements are converted to equivalent 376-pound barrels. Therefore, the number of "barrels" of those varieties actually shipped is about one-third higher than is shown in Bureau of Mines statistics; for example, a total of about 14,350,000 barrels of portland-base masonry and 2 million barrels of natural-base masonry cement was shipped in 1952. The same practice is followed with natural and puzzolan cements and hydraulic lime.

In 1952 shipments of portland cement were reported by 156 plants in 37 States and Puerto Rico. Many companies owned single plants, while others controlled from 2 to 12 or more plants at widely separated points. Lehigh Portland Cement Co., Allentown, Pa., and

Ideal Cement Co., Denver, led in number of plants, with 14 each. Other companies operating over 5 plants each were: Lone Star Cement Corp., New York City, 12; Universal Atlas Cement Co., New York City, 10; Alpha Portland Cement Co., Easton, Pa., 8; Penn-Dixie Cement Corp., Nazareth, Pa., 7; and Marquette Cement Mfg. Co., Chicago, Medusa Portland Cement Co., Cleveland, and General Portland Cement Co., Chicago, 6 each. The cement plant in South Dakota is State owned. Natural, masonry (natural), and puzzolan cements and hydraulic lime were manufactured in 8 plants, owned by different companies, in 6 States. Masonry cements (portland-base) were manufactured at 107 plants in 33 States.

Although portland-cement plants are well distributed throughout the country, some areas show greater concentrations of facilities than others. Natural cement first was manufactured in eastern Pennsylvania in the middle part of the 19th century, and portland cement gained an early foothold there before 1880. Because of the occurrence there of immense deposits of the raw materials, including coal for fuel, and the proximity to a populous and commercially active area, many plants were built and have continued in operation. At present two counties (Northampton and Lehigh) of the so-called Lehigh district of eastern Pennsylvania have 13 plants. Another group of five plants is in Greene and Columbia Counties, N. Y., along the Hudson River. Five plants are closely grouped in the Birmingham area, 3 are in La Salle County, Ill., 6 are in eastern Kansas, 4 are near Dallas, Tex., and there are 6 in the general Los Angeles area. Another plant was under construction in the Los Angeles area at the end of 1954.

TECHNOLOGY

A large number of raw materials theoretically is available for making portland cement, but the number has been reduced to a comparative few under existing commercial conditions. Argillaceous limestone, or "cement rock," of very close to the required lime–silica–alumina–iron oxides proportions sometimes is combined with minor quantities of other materials to produce the proper mixture. More often the necessary lime is supplied by one material (limestone, oystershells, or marl), the silica by sand or sandstone (or as a component in clays and shales), the alumina by clays and shales, and the iron oxide by iron-bearing materials or by high-iron clays and shales.

The four broad "type combinations" of portland cement raw materials, as classified in Minerals Yearbook, are (1) cement rock-pure limestone, (2) limestone-clay (or shale), (3) marl-clay, and (4) blast-furnace slag-limestone.

Although in many areas ample deposits of raw materials suitable for use in manufacturing portland cement exist, in other areas quarries have been abandoned because of depletion of rock of the required quality. Suitable raw-material sources are scarce in some areas. The magnitude and quality of raw material deposits are major factors in determining the geographical distribution of the industry.

Besides the major materials, the following minerals and other substances, some of which are used only very rarely, are among those added to help obtain a raw mixture of correct chemical and physical characteristics: Bentonite, diaspore, diatomaceous shale, fuller's earth, iron ore, mill scale, pyrite cinders and ore, diatomite, fluorspar, pumicite, flue dust, pitch, red mud and rock, hydrated lime, tufa, cinders, and calcium chloride.

Percentage depletion at the 5-percent rate is granted on clay and shale used in manufacturing cement. Sand, pumice, and marine shells also come under the 5-percent rate. Limestone, fuller's earth, quartzite, and diatomaceous earth are accorded the 15-percent rate.

Cement is such a low-priced commodity that it is imperative that every economy of rock crushing, transportation, grinding, and calcining be utilized. The rock is recovered from large quarries (or, in a few instances, from underground mines), or the marl or shell is dug from open pits or dredged from offshore deposits.

The hard materials are reduced by crushing in large jaw, gyratory, or roll crushers to about 6 inches maximum size. Secondary crushers then reduce them further to about three-quarter inch maximum size. At this point, the various types of raw materials are analyzed.

The several components then are fed into the final raw grinding unit, a battery of tube or ball mills, in which the material is ground together to very fine particle sizes, with about 85 to 90 percent passing through a 200-mesh sieve. The mixture then is analyzed, as precise control of composition is maintained at this stage. The final raw-grinding unit may be either dry or wet, depending on whether the dry or the wet process is used at the plant.

Two different processes, which vary in method but not in principle, are used in the manufacture of cement. In the so-called "dry process" the mixture of raw materials is fed into kilns in a dry condition; in the "wet process" an aqueous slurry of controlled consistency is burned. The advantages of each process have been discussed in the literature at some length. Choice usually is based on the nature of the raw materials. The wet process usually is preferred with the normally wet materials such as marls, shells, or wet clays, or where heavy-medium flotation is used to beneficiate low-grade raw materials. Of new plants built since 1946, 8 have been wet-process plants with a total initial annual capacity of about 12.5 million barrels, while the other 4 were dry-process plants with a total initial annual capacity of about 5.5 million barrels.

The flotation process originally was applied to the beneficiation of metallic ores and, later, to many of the nonmetallics. It was used successfully several years ago by a portland-cement plant in Pennsylvania to improve the quality of its limestone and now is employed in plants in many countries. Excess silica or silicates and other deleterious constituents present in the limestone can be removed by its use. Portland cement meeting current specifications can be made from limestone formerly considered unsuitable for this use. Flotation is practicable for such a low-value product as raw cement material because rather fine grinding is necessary for the material before calcining, regardless of whether or not flotation is applied.

A horizontal, rotary portland-cement kiln is one of the largest pieces of moving machinery used in any industry; some such kilns are up to 500 feet in length. Portland-cement kilns are steel cylinders lined with the necessary refractories, inclined slightly, and rotated slowly. The fuel, which may be oil, gas, or powdered coal or mixtures thereof, is blown into the kilns under pressure, creating tongues of flame that reach 30 to 40 feet into the kiln. The raw mixture is fed automatically into the upper end of the kiln. Hot gases from the flames in the lower end of the kiln pass over the raw mixture, expelling the moisture and, by driving off carbon dioxide, converting carbonates to oxides. As the material advances slowly down the tube it reaches the zone of highest temperature (2,700° F.)—the so-called burning zone—in the lower third of the kiln, where chemical and physical changes take place. The material reaches the point of incipient fusion, and complex silicates and aluminates are formed. When it emerges from the kiln it is a new substance in the form of hard, glassy balls called clinker, which may be passed through a cooler before it is sent to storage or may be cooled while being transported to storage by drag chain.

Cement clinker is relatively inert after it has cooled and may be stored some months without serious deterioration, particularly in dry climates. When it is taken from storage to be ground into cement, the required quantity—approximately 2 to 4 percent—of gypsum is added to regulate setting time. "Finishing" mills—usually ball or tube mills—reduce the clinker to extremely fine particle sizes. Finished portland cement may be packed in bags holding 94 pounds or may be shipped in bulk. Bags are usually of paper, although a very small quantity is packed in cloth bags. An increasingly larger percentage of the total output is shipped in bulk form, in large trucks, railroad hopper cars, or barges. The standard 376-pound barrel is equivalent to four 94-pound bags. Bags holding 94 pounds are assumed to contain 1 cubic foot of portland cement.

As manufactured by most companies, there are no major byproducts of portland cement. In preparing a mixture for the kilns, especially when flotation beneficiation is employed, several types of material may be rejected from the kiln mixture. They either are sent to waste or, if a market can be found, sold as a byproduct of the manufacturing process. Materials that have been recovered from the flotation circuit and marketed include mica and graphitic fines. Some portland-cement kiln dusts have been retained and treated to recover potash.

From time to time, especially when the demand for cement is weak, portland-cement plants turn to the manufacture or processing of other materials that they may be equipped and favorably located to handle. These include crushed stone for agricultural use, road purposes, or flux; fertilizers; and the custom crushing of other minerals or materials. Idle kilns occasionally are converted to the production of lime or utilized for other purposes, such as the drying and calcining of bauxite, manganese ore, or phosphate, the sintering of clays and shales to obtain lightweight aggregates, the nodulization or agglomeration of iron ore for open hearth and blast furnaces, etc.

USES

Portland cement has little utility alone but rather is the chemically reactive material which, in the presence of water, binds mineral aggregates into concrete. The addition of water gives rise to a hard, hydrous gel that binds the mineral aggregates firmly together.

As the binder in concrete for engineering construction, such as architectural concrete, highway and airport pavement, farm uses, residential construction, hydraulic dams and irrigation systems, and many other applications, portland cement competes in the market with asphalt, structural steel, clay products, building stone, and other materials. Sometimes flat paving stone, mortared with cement or left unmortared, is used for streets and sidewalks. A property owner may build principally with wood or brick rather than with concrete block, prefabricated-concrete units, or poured concrete, but almost without exception every structure uses a certain quantity of portland or masonry cement in the foundation, basement, mortars, floor, or roof. The choice of principal structural material is governed by many factors of cost, personal preference, building-code specifications, etc.

The early uses of portland-cement concrete in the United States were in heavy construction, for foundations, retaining walls, dams, piers, bridge abutments, sidewalks, and highway and street pavements. In all of these uses, strength and durability were the primary requirements. Although such uses are very important even now, others have become prominent, and the following use classification has been suggested:

A. Architectural concrete.
B. Residential, commercial, and public construction.
C. Pavement.
D. Miscellaneous farm uses.
E. Concrete pipe.
F. Sanitary construction.
G. Water supply and irrigation.
H. Flood and soil erosion control.
I. Recreation facilities.
J. Lightweight concrete, girders, beams, **etc.**
K. Soil cement.
L. Miscellaneous uses.

STATISTICS

Statistics for 1926–52, covering production, shipments, imports, exports, and apparent consumption of the hydraulic cements included in this paper, appear in tables 1, 2, and 3.

TABLE 1.—*Production of cement in the United States, 1926–52* [1]

Year	Production (in barrels)		
	Portland	Natural, masonry-natural and puzzolan [2]	Total, all cements
1952	249, 256, 154	3, 401, 684	252, 657, 838
1951	246, 022, 476	3, 449, 463	249, 471, 939
1950	226, 025, 849	4, 246, 299	230, 272, 148
1949	209, 727, 417	3, 185, 229	212, 912, 646
1948	205, 448, 263	3, 440, 248	208, 888, 511
1947	186, 519, 347	2, 951, 098	189, 470, 445
1946	164, 064, 188	2, 474, 674	166, 538, 862
1945	102, 804, 884	1, 483, 763	104, 288, 647
1944	90, 905, 696	1, 246, 703	92, 152, 399
1943	133, 423, 788	1, 830, 266	135, 254, 054
1942	182, 781, 184	2, 560, 425	185, 341, 609
1941	164, 030, 559	2, 875, 962	166, 906, 521
1940	130, 216, 511	2, 534, 566	132, 751, 077
1939	122, 259, 154	2, 439, 110	124, 698, 264
1938	105, 357, 000	1, 820, 795	107, 177, 795
1937	116, 174, 708	1, 900, 643	118, 075, 351
1936	112, 649, 782	1, 819, 488	114, 469, 270
1935	76, 741, 570	1, 006, 064	77, 747, 634
1934	77, 747, 765	671, 588	78, 419, 353
1933	63, 473, 189	511, 201	63, 984, 390
1932	76, 740, 945	456, 785	77, 197, 730
1931	125, 429, 071	1, 241, 803	126, 670, 874
1930	161, 197, 228	1, 792, 083	162, 989, 311
1929	170, 646, 036	2, 209, 465	172, 855, 501
1928	176, 298, 846	2, 210, 404	178, 509, 250
1927	173, 206, 513	2, 123, 868	175, 330, 381
1926	164, 530, 170	2, 104, 891	166, 635, 061

It will be noted that new output records have been set each year since 1947.

As shown in table 3, foreign trade has never assumed much importance in the portland-cement industry. Most imported cements either have been used along certain sections of the east coast that are some distance from the nearest plant in the United States or else are special types of cement used in such small quantities that domestic manufacturers find it impracticable to manufacture and market them. Since World War II, exports have been greater than usual because of high demand from Canada, Venezuela, and other countries in the Western Hemisphere; but that market for American cements has declined, as many of those nations have become more nearly self-sufficient through the construction of new plants within their own borders. Manufacturing costs in most of those countries usually are markedly higher than in the United States, but the cost of transporting American cements makes locally produced cements cheaper on a delivered basis.

TARIFFS

Imports from other countries into the United States are subject to duties as follows: Roman (natural), portland and other hydraulic cements and cement clinker (except from Communist-dominated countries, in which case the rate is 6 cents per 100 pounds), 2¼ cents per 100 pounds; white, nonstaining cement, 3 cents per 100 pounds; other hydraulic cements not otherwise specified, 5 percent ad valorem.

TRANSPORTATION

The transportation cost factor not only limits the volume of imports and exports but also plays a major role in governing the movement of cement within an area and from one area to another and makes difficult the easing of overages and shortages that have developed in various parts of the country in recent years.

The so-called Basing-Point Decision, which declared illegal the practice whereby several cement companies would quote identical prices for their products at designated pricing points, forbids collusive pricing.

The United States can be considered self-sufficient as far as cement is concerned, although the plants sometimes are not as advantageously located as they might be. There are heavy concentrations of productive capacity in Pennsylvania, New York, and Kansas, whereas other areas have shown the most active recent demand. New Plants have been built in Texas and the South, while some California mills have increased their capacities to keep abreast of the market. In addition, an Arizona mill has relieved a part of the demand of that State on the output in southern California

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[1] Source of data: Bureau of Mines, Minerals Yearbooks, Cement chapters, 1932–52; Mineral Resources of the United States, Cement chapters, 1926–31 (includes Puerto Rico, 1939–52, and Hawaii, 1945–46).

[2] 1934–52: Includes figures for hydraulic lime from 2 producers (with 1 plant each, located, respectively, at Riverton, Va., and Highcliff, Wis.).

TABLE 2.—*Shipments of cement in the United States, 1926–52* [1]

Year	Shipments from mills					
	Portland			Natural, masonry-natural, and puzzolan [2]		
	Barrels of 376 pounds	Value		Barrels of 376 pounds	Value	
		Total	Average per barrel		Total	Average per barrel
1952	251, 368, 503	$638, 512, 228	$2. 54	3, 447, 390	$9, 751, 837	$2. 83
1951	241, 153, 272	613, 170, 483	2. 54	3, 475, 423	9, 832, 956	2. 83
1950	227, 756, 636	535, 321, 123	2. 35	4, 218, 580	10, 629, 586	2. 52
1949	206, 080, 325	473, 177, 032	2. 30	3, 233, 525	8, 006, 361	2. 48
1948	204, 304, 662	445, 678, 073	2. 18	3, 375, 135	7, 734, 289	2. 29
1947	187, 491, 869	356, 213, 976	1. 90	2, 927, 885	5, 764, 398	1. 97
1946	169, 567, 593	292, 396, 343	1. 72	2, 533, 106	4, 155, 171	1. 64
1945	106, 353, 595	173, 337, 010	1. 63	1, 479, 513	2, 093, 848	1. 42
1944	94, 271, 881	150, 357, 754	1. 59	1, 320, 274	1, 638, 892	1. 24
1943	127, 631, 859	200, 103, 216	1. 57	1, 846, 803	2, 357, 112	1. 28
1942	185, 300, 884	283, 237, 028	1. 53	2, 508, 324	3, 668, 169	1. 46
1941	167, 439, 237	246, 621, 914	1. 47	2, 926, 203	3, 967, 567	1. 36
1940	130, 349, 786	190, 078, 068	1. 46	2, 514, 597	3, 386, 801	1. 35
1939	122, 651, 459	180, 893, 208	1. 47	2, 405, 135	3, 361, 724	1. 40
1938	106, 324, 127	153, 977, 226	1. 45	1, 867, 949	2, 725, 776	1. 46
1937	113, 804, 782	168, 835, 208	1. 48	1, 873, 400	2, 578, 885	1. 38
1936	112, 849, 979	170, 415, 302	1. 51	1, 760, 993	2, 362, 396	1. 34
1935	75, 232, 917	113, 372, 182	1. 51	1, 011, 411	1, 437, 542	1. 42
1934	75, 901, 279	116, 921, 084	1. 54	678, 204	960, 732	1. 42
1933	64, 282, 756	85, 583, 916	1. 33	477, 761	644, 750	1. 35
1932	80, 843, 187	82, 021, 723	1. 01	524, 844	696, 474	1. 33
1931	127, 150, 534	140, 959, 906	1. 11	1, 226, 850	1, 619, 920	1. 32
1930	159, 059, 334	228, 779, 756	1. 44	1, 787, 016	2, 469, 531	1. 38
1929	169, 868, 322	252, 153, 789	1. 48	2, 159, 130	2, 950, 717	1. 37
1928	175, 838, 332	275, 972, 945	1. 57	2, 213, 645	2, 910, 097	1. 31
1927	171, 864, 728	278, 854, 647	1. 62	2, 158, 323	2, 881, 029	1. 33
1926	162, 187, 090	277, 965, 473	1. 71	2, 031, 851	2, 820, 110	1. 39

[1] Source of data: Bureau of Mines, Minerals Yearbooks, Cement chapters, 1932–52; Mineral Resources of the United States, Cement chapters, 1926–31 (includes Puerto Rico, 1939–52, and Hawaii, 1945–46).

[2] 1934–52: Includes figures for hydraulic lime from 2 producers (with 1 plant each, located, respectively, at Riverton, Va., and Highcliff, Wis.).

The Arizona mill currently is expanding its capacity, and a new plant is under construction in southern California.

The defense program has given impetus to demand in certain areas throughout the United States. Construction of Army and Navy installations, large atomic plants, airfields, and the like require large quantities of cement and have in the past caused such dislocations in nearby cement trade channels as local shortages and rationing, temporary unavailability of cement, etc. Since such construction projects bring with them increased store, home, and road building in the immediate vicinity, the supply situation sometimes has become acute. Cement then has been brought in from considerable distances at an increased cost to users.

The average f. o. b. mill value of portland cement has increased steadily since 1940, rising from $1.46 per barrel that year to $2.54 per barrel in 1951 and 1952. Preliminary data for 1953 indicate another increase of several cents per barrel in average mill value. However, the percentage increases in price for cement and for aggregates used with it in concrete have lagged far behind such other competitive building materials as lumber during this period. This increase in portland-cement price reflects the general business trend toward higher wages, higher cost for all supplies and equipment, and higher taxes. Except for such economies in manufacture as lower fuel and power consumption and greater output per man-hour than ever before, the cost of portland cement would be considerably higher than it is.

The cost of constructing new plants has soared, and estimates of cost for a new plant now range between $9 and $12 per annual barrel of capacity, with the minimum annual economic capacity in the neighborhood of 1

TABLE 3.—*Consumption of cement in the United States, 1926–52* [1]

Year	Imports, (barrels)	Exports, (barrels)	Apparent domestic consumption, all cements, (barrels) [2]	Active plants	
				Portland	Natural, masonry-natural, and puzzolan [3]
1952	475, 986	3, 174, 405	252, 117, 474	156	8
1951	921, 953	2, 932, 787	242, 617, 861	155	9
1950	1, 409, 974	2, 418, 435	230, 966, 755	150	9
1949	109, 821	4, 561, 899	204, 861, 772	150	9
1948	282, 752	5, 922, 163	202, 040, 386	150	9
1947	4, 606	[4]6,771,250	[4]183,454,387	150	9
1946	3, 734	5, 163, 362	166, 941, 071	153	9
1945	323	6, 474, 721	101, 358, 710	145	9
1944	169	4, 040, 405	91, 551, 919	151	9
1943	13, 658	1, 731, 956	127, 760, 364	153	10
1942	644	1, 100, 826	186, 709, 026	155	11
1941	43, 466	2, 556, 234	167, 852, 672	155	12
1940	538, 060	1, 667, 595	131, 734, 848	152	12
1939	1, 913, 853	1, 146, 339	125, 824, 108	150	12
1938	1, 727, 411	558, 226	109, 361, 261	151	12
1937	1, 803, 932	378, 554	117, 103, 560	150	12
1936	1, 658, 902	334, 673	115, 935, 201	149	13
1935	619, 404	416, 099	76, 447, 633	150	13
1934	265, 997	566, 171	76, 279, 309	150	14
1933	477, 193	680, 307	64, 512, 057	152	13
1932	468, 139	374, 581	81, 461, 589	160	15
1931	469, 598	429, 653	128, 417, 329	160	12
1930	984, 807	755, 778	161, 075, 379	163	11
1929	1, 745, 345	885, 321	172, 887, 476	163	11
1928	2, 302, 475	824, 656	179, 529, 796	156	11
1927	2, 065, 730	816, 726	175, 272, 055	153	11
1926	3, 244, 223	974, 326	166, 488, 838	140	11

[1] Source of data: Bureau of Mines, Minerals Yearbooks, Cement chapters, 1932–52; Mineral Resources of the United States, Cement chapters, 1926–31 (includes Puerto Rico, 1939–52, and Hawaii, 1945–46).

[2] Computed on the basis of shipments plus imports minus exports.

[3] 1934–52: Includes figures for hydraulic lime from 2 producers (with 1 plant each, located, respectively, at Riverton, Va., and Highcliff, Wis.).

[4] In 1947, 198,723 barrels, valued at $839,916, shipped under the U. S. Army civilian supply program, is excluded from exports shown but deducted from apparent consumption.

million barrels. Most portland-cement producers have expanded capacity by improving and enlarging their crushing, grinding, and kiln departments wherever possible in preference to erecting new plants. Most plants recently built in the United States have designed annual capacities in the 1.5 million to 3 million-barrel range.

RESEARCH

The Portland Cement Association, 33 West Grand Ave., Chicago, Ill., is the principal trade organization of the portland-cement industry. It does not concern itself with the sale, distribution, or commercial phases of the industry but confines its activities to scientific research, development of new or improved products and methods, technical service, and promotional and educative efforts, including safety work. The broad program of the association is financed from dues contributed by a majority of the portland-cement manufacturers of the United States and Canada.

In 1950 the association completed its new research and development laboratories at Skokie, Ill., a suburb of Chicago. These are the largest and most completely equipped laboratories in the world devoted exclusively to research and development in the uses of portland cement and concrete. In addition, field laboratories and experimental projects are maintained at other locations.

The American Concrete Institute also disseminates information on the design, construction, manufacture, use, and maintenance of concrete products and structures.

Besides the work of the Portland Cement Association and the American Concrete Institute, many companies maintain individual experimental laboratories for work on their own products, and private, commercial, engineering college, and Government laboratories conduct tests and experiments.

OUTLOOK

One significant use trend is that of prestressing portland-cement concrete to impart to it the tensile strength necessary to make it more useful, particularly in girders and beams. Also recently developed, in part to save structural steel, were the lightweight concretes that have been used in increasing quantities as the required aggregate became available and when methods of producing a light, cellular concrete became known. Other use trends include improvement and wider use of soil cement, the development of tilt-up and other methods of prefabrication for more rapid and economical construction of houses and commercial buildings, and the increasing use of air-entraining cements and concrete for improved durability of concrete exposed to freezing and thawing or heavy salt exposure.

Trends in manufacturing methods and practices include the growing use of preheaters on kiln raw materials to increase production and save fuel; the increasing number of plants using marine shells as raw material; the increasing use of air-conveying systems for moving finished cement; and the improved heat-transfer and heat-recuperation equipment now employed. A marked change toward longer kilns of higher capacity is noted, both in the new plants and in remodeled or expanded old plants. New sources of raw materials include the marginal materials that can be made usable through flotation beneficiation, and continued development of the use of beneficiation can be expected.

The geographical pattern of distribution of portland-cement facilities and capacities is shifting to the South, Texas, and California. New plants have been built within the last few years in Virginia, Mississippi, Louisiana, Texas, Arizona, Florida, and Alabama.

The ownership pattern for portland-cement plants has changed but slightly. The high initial cost of a new plant, along with other deterring factors, discourages the investment of capital by organizations not already concerned with cement production.

Portland-cement consumption bears a close relationship to local and regional developments. When a locality or an area is commercially active, construction of all of the facilities in which concrete plays a vital part is increased, and the demand for portland cement is greater. The health of the industry, therefore, is closely tied to the construction outlook.

PROBLEMS

The problems confronting the portland-cement industry include improvement of production methods, development of better concretes to meet specific needs, encouragement of the use of concrete wherever possible in place of scarcer or more expensive materials, education of the public to proper methods of preparing and using good concrete, and solution of the economic problems of cost and distribution that generally face industries.

Of major concern to the industry is the trend of demand. Continued high consumption of portland cement depends on high-level commercial, road, airport, and dam building and the like and to a certain extent on successful solution of concrete cost problems so that it may be sold at a price that does not discourage such construction.

An increasing number of portland-cement plants face problems relating to the availability or economics of raw materials. In some areas, where demands have grown enough to justify the erection of new portland-cement plants, prospective producers sometimes encounter difficulties in finding suitable raw materials in adequate quantity and favorably located.

Research on new and expanding applications and wider dissemination of information about them are needed. Special attention should be given to those uses in which cement can serve as a substitute or alternative for scarcer commodities.

Increasing attention has been devoted to the sometimes harmful effect of the reaction between alkalies in the cement and certain constituents in aggregates. Where reactive aggregates are found, limits are placed on the alkali specifications of the cement. Concrete technologists are engaged in studies relating to this reaction and to remedial materials and practices.

Mining, crushing, grinding, calcining, and finishing practices all have been improved, but there is a continuing need for further improvement of production and processing methods that will provide greater efficiency and higher quality.

Compilation and distribution of more detailed and comprehensive technical, statistical, and economic information are needed to provide the industry and Government agencies with basic facts required for the most effective management and guidance of policy decisions.

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CESIUM

By

John D. Sargent[1]

CESIUM, of snooperscope and sniperscope fame, is the most compressible, most alkaline, and most electropositive of all the elements. These and other chemical superlatives should cause it to be a metal of increasing importance.

Summary

Cesium, a soft silver-white metal resembling potassium, is one of the rarer elements that has assumed strategic importance because of its use in photoelectric cells. It is the heaviest of the alkali series of metals, the most electropositive, and the softest. The metal is liquid near room temperature and reacts strongly with oxygen and moisture; hence, it must be stored in a vacuum or immersed in an inert liquid.

Pollucite, the chief ore mineral of cesium, is rarely found in quantity. Cesium sometimes occurs as a constituent of beryl, lepidolite, rhodizite, and carnallite and in the mother liquors derived from processing lithium and beryllium minerals.

The principal use of cesium is in photoelectric cells employed in photography, photometry, television, and signaling devices. Cesium is also used in scintillation counters and various optical detection devices.

The current production of cesium is adequate to meet the present limited demand.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

HISTORY

In 1860, Bunsen and Kirckhoff noted the element cesium while making a spectroscopic examination of a sample of spring water from Durkheim, Germany, and proposed the name cesium from the Latin word meaning sky blue because of the characteristic color it imparted to the visible spectrum. Cesium was the first element discovered by the use of the spectroscope; the following year, rubidium, which commonly occurs with cesium, was discovered by the same method (*7*).[2]

In 1864 Pisani analyzed the mineral pollucite and determined that the metal called potassium since 1846 was, in reality, cesium (*7*). Setterberg isolated the element in 1881.

OCCURRENCE

Cesium occurs in concentrations as high as 1 percent in lepidolite, beryl, microcline feldspar, and carnallite and as a trace in most lithium minerals; however, the most important source is the mineral pollucite, a hydrous silicate of aluminum and cesium, containing a theoretical maximum of 42.5 percent Cs_2O (*4*).

Pollucite often appears clear and colorless, somewhat resembling quartz. However, it is sometimes translucent to opaque, with a fine-grained texture resembling massive beryl. Occasionally it appears dull and almost gumlike (*4*). It is not easily recognized by casual inspection, and probably many occurrences are passed unnoticed in the working of granitic pegmatite dikes for other minerals. Pollucite's crystal system is isometric, often in cubes; it has a hardness of 6.5 and specific gravity of 2.90.

Lepidolite is a complex lithia mica containing 4 to 6 percent of Li_2O, as much as 0.77 percent of Cs_2O, and as much as 3 percent of Rb_2O (*7*).

Cesium is never found in nature in a metallic state. Although it ranks 37th in abundance in the earth's crust, it rarely occurs in large concentrations. Detectable quantities of cesium are found in plant and animal organisms, mineral waters, and soils. The content of cesium in sea water has been determined to be up to 0.02 mg. per liter and up to 0.3 mg. per liter in bittern. Granites and feldspars in Finland have been reported to contain 0.005 to 0.006 percent Cs_2O.

GENERAL PROPERTIES

In the pure form cesium is a soft, silver-white alkali metal, but it is frequently golden yellow due to the presence of oxide or nitride. It is much more compressible than most metals, as shown by a recent experiment which succeeded in compressing the metal 62.5 percent under 34,000 pounds of pressure per square inch.

The melting point of cesium is about 28.5° C.; hence it is liquid near room temperature. It has the lowest boiling point (670° C.) and highest specific gravity (1.873 at 20° C.) of the alkali metals. On exposure to air, it tarnishes quickly and melts, or it may burst into flame as oxidation occurs. When thrown upon water at ordinary temperature, it reacts vigorously, decomposing the water rapidly and burning with a reddish violet flame. The reaction between moisture and cesium cannot be detected below $-116°$ C., which may be compared with $-108°$ C. for rubidium, $-105°$ C. for potassium, and $-98°$ C. for sodium. Cesium is the most active of all metals and the most electropositive of all known elements; it shows strong ionization tendencies. In electron emission and photoelectric effect, it is the most active of the alkali metals. It has the largest atomic volume of all the known elements. Cesium is the most easily fusible of the alkali metals. It dissolves in acids and alcohols, reacting with them to form salts and alcoholates.

Cesium absorbs carbon monoxide at low temperatures and low pressures. It has a remarkable affinity for oxygen and ignites spontaneously in dry air at room temperatures; hence the metal must be kept in a vacuum or immersed in mineral oil, naphtha, kerosine, or paraffin.

In the fission of uranium-235, various long-lived radioisotopes of cesium are among the more abundant end products. Cesium-135 and cesium-137 are the important cesium radioisotopes. Cesium-137 has a half life of 33 years and cesium-135 a half life of 2,900,000 years.

METALLURGY

The first samples of cesium were obtained by Setterberg in 1881 by the electrolysis of a fused mixture of CsCn and $Ba(CN)_2$ (*4*). Three methods for preparing cesium and its compounds are practiced:

1. Calcium and ammonium chloride method.
2. Cesium alum method.
3. Direct-reduction method.

The calcium and ammonium chloride method consists of (1) heating a mixture of ore, $CaCl_2$, and NH_4Cl in a nickel crucible for 1 hour at dull-red heat; (2) cooling and extracting with water; (3) removing most of the calcium by addition of H_2SO_4, filtering, and precipitating the remaining calcium with $(NH_4)_2CO_3$; and

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[2] Italicized numbers in parentheses refer to items in the bibliography at the end of this chapter.

(4) precipitating the cesium as a chloroplatinate (Cs_2PtCl_6).

The cesium-alum method consists of (1) digesting finely ground ore with H_2SO_4; (2) allowing the cesium alum to crystallize from the solution; (3) dissolving the precipitate in HCl; (4) precipitating the cesium with $SnCl_2$; (5) purifying the product by dissolving the precipitate in HCl and removing the Sn with H_2S; and (6) adding antimony trichloride to the filtrate to form a double chloride with antimony.

The direct-reduction method consists of reducing cesium chloride to the metal using calcium, aluminum, or magnesium as the reducing agent.

USES

Cesium is used in photoelectric cells as the active agent in the emission of electrons. The advantage of cesium is that its range of sensitivity corresponds closely to that of the human eye (*2*). Cesium cells respond to minutely varying intensities of visible light and are utilized in photometry, television, sound films, and signaling devices.

Radio vacuum tubes employ substantial quantities of cesium as a getter. Some types of radiodetector tubes utilize cesium metal to enhance detecting qualities. The metal is also used in vapor lamps adapted for infrared signaling by the military services. During World War II an infrared lamp employing cesium vapor was developed by the Navy for ship-to-shore and ship-to-ship invisible signaling (*2*). Cesium has also assumed strategic importance because of its use in infrared photography. Vapor rectifiers use cesium on a limited scale, and it has been employed as a catalyst in polymerization of organic compounds (*2*).

Cesium is used in scintillation counters and various optical and detecting devices, such as the sniperscope and snooperscope. The sniperscope is an instrument for viewing objects in the dark. It operates by means of rays, transmitted by an infrared-lamp source, striking objects up to 10 miles or more away and being reflected back. These reflected rays strike a cesium surface within the scope, causing emission of electrons which are then transformed into a beam focused on a fluorescent screen to produce a visible image. The snooperscope is the same as the sniperscope but is adapted for use by night watchmen and guards and for border-patrolling purposes. Crystals of cesium halides are used in infrared spectrometry, particularly in the longer wave-length bands.

PRODUCTION AND CONSUMPTION

Before 1932 cesium was considered to be a laboratory curiosity, and since that time the demand for cesium and its compounds has totaled only a few hundred pounds annually. During World War II the quantities of cesium-bearing ore produced in the United States were inadequate to meet defense requirements, and the deficit was made up by importing pollucite from Brazil, South-West Africa, and the Union of South Africa.

Domestic production statistics for cesium ore are incomplete, but the total United States production to date is estimated to be less than 300 tons (*2*). Pollucite recovered from pegmatite mines in the Black Hills of South Dakota and in Maine was the only important mineral. The Tin Mountain mine near Custer, S. Dak., was formerly the chief source of this ore in the United States. An output of about 100 tons was reported in South-West Africa during 1946–51. Jooste lithium mines, near Karibib, South-West Africa, is now the largest producer and supplier of pollucite. The grade of the ore varies from 1 percent up to 34 percent Cs_2O, averaging about 25 percent Cs_2O. Several tons of ore has been recovered annually in the Union of South Africa as a byproduct of beryl mining. Six tons of this ore was exported to the United States in 1952. Important quantities of cesium ore (chiefly pollucite) have been reported at the Boliden mines, Varutrask, Sweden, and in the Bikita tin fields of Southern Rhodesia.

The domestic producers of cesium and its compounds are De Rewal International Rare Metals Co., Philadelphia, Pa.; Fairmount Chemical Co., Newark, N. J.; General Electric Co., Scotia, N. Y.; A. D. Mackay, Inc., New York, N. Y.; Foote Mineral Co., Philadelphia, Pa.; and Harshaw Chemical Co., Cleveland, Ohio.

PRICES

During 1953 the following prices per gram for cesium and cesium compounds were quoted:

Cesium metal	\$2–\$5.
Cesium carbonate	35 cents.
Cesium chloride	30 cents.
Cesium chromate	43 cents.
Cesium nitrate	30 cents.

Large-quantity orders could be obtained at lower prices.

Pollucite ore, chiefly foreign, has been sold at about \$400 to \$600 per ton, depending on Cs_2O content, compared to a price ranging up to \$15 per pound or \$30,000 per ton for average-grade material in 1926–27 (*2*).

RESEARCH

Early research was directed toward developing high-purity cesium metal. Considerable research in applications for cesium metal and salts has been carried out by large industrial firms, such as General Electric Co., Westinghouse Electric Co., and Radio Corp. of America. Radio Corp. of America developed the cesium vacuum tubes for the radio industry, creating a sudden large demand for the metal in 1926–30. The demand subsided shortly thereafter with a change in types of tubes.

The Harshaw Chemical Co., Cleveland, Ohio, has conducted research on cesium bromide and cesium iodide for use in scintillation counters and various optical devices. The company has developed and marketed cesium bromide optical crystals. Cesium iodide research is still in the development stage, but the compound is known to have potential optical and scintillation counting uses.

The National Bureau of Standards has conducted research directed toward determining the value of cesium as a microwave frequency standard in electronics and spectroscopy (*2*). They have also engaged in research on the purification of the metal.

OUTLOOK

The physical and chemical nature of cesium appears to assure its widespread utilization in the future. The supply of cesium has been adequate to meet the limited demand of recent years but would not be enough to meet a sharp rise in future annual requirements. Efforts directed toward discovery of adequate sources of the metal, development of improved recovery processes, and investigation of the application of cesium to industrial uses can be expected.

PROBLEMS

The major cesium problem is procuring an adequate and dependable supply of cesium ores. Reserves of pollucite, a rare pegmatite mineral, are very difficult to estimate, and there are few known potentially commercial deposits. The domestic reserves of known low-grade ores and materials that contain cesium are relatively small. Cesium and its compounds have many potential industrial, scientific, and military applications, but its limited availability, high cost, and its unfamiliarity to most consumers has retarded its progress in industry. Cesium is difficult to extract from its minerals and difficult to produce in high-purity metal form; moreover, it requires special care in handling, storing, and shipping.

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CHROMIUM

By

Charles Katlin[1]

MIDAS-LIKE, chromium enhances the value of everything it touches. Without traces of chromium, the beautiful and valuable emerald would be comparatively worthless beryl, and the richly colored ruby would be ordinary corundum. Chromium makes paints more brilliant and steels stronger and more durable. Coatings of bright chromium plate measured in millionths of an inch add beauty and value to automobiles, household appliances, and metalware of all sorts. And chromium alloys used in jet aircraft make possible the resistance to high temperature, corrosion, and fatigue that is essential in passing the sound barrier.

Summary

New uses and increased need for almost all uses brought the consumption of chromium to a record level in 1953 with the United States consuming one-third of the world production of chromite. Reserves of specification grade ores are not being replenished at the high rate of consumption, with the result that the metallurgical industry, the largest consumer of chromite, is using a progressively greater percentage of lower-grade ores.

At present, Turkey, Southern Rhodesia, and New Caledonia are the principal United States sources of high grade metallurgical chromite. The Philippines and Cuba produce most of the refractory ore, and all chemical chromite comes from the Union of South Africa. Domestic production for consumption is virtually nil; almost all current United States production is being purchased by the Government at high-incentive prices.

Chromite is used metallurgically in making ferrochromium alloys and metal, which in turn are used to manufacture stainless steels, tool steels, high-temperature alloys, and various other alloys. Refractories manufacturers make chromite bricks and cements, principally for lining steel furnaces. Chromium chemicals are used chiefly in pigments, metal treating, and tanning. Virtually all uses of chromium are increased in wartime.

It is anticipated that chromium metal will be produced by the chemical industry as an outlet for its unused capacity, particularly in view of interest in the recently created ductile chromium.

Expanded uses for chromium, coupled with approaching depletion of the world's known special grades of chromite, will eventually necessitate use of both lower grade ores and concentrates for all purposes, creating problems for the consuming industries.

Dependence on distant foreign sources for most United States chromite needs will remain until nearby sources, such as Cuba's laterites, can be utilized commercially as a source of chromium.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

ORGANIZATION AND DISTRIBUTION OF CHROMIUM INDUSTRY

The American chromium industry is composed mainly of those companies using chromite for manufacturing chromium alloys, refractories, and chemicals. Table 1 lists the principal chromite-consuming firms in the United States, the location of plants, and the metallurgical products made. As electric power (4,500 to 8,000 kw.-h. per ton for ferrochromium) is an important item in production costs, the metallurgical plants are located near low-cost power areas. Most plants producing chrome refractories and chemicals are situated near the east and west coasts to minimize ore-transportation costs.

In recent years substantial domestic production of chromite has been possible only when stimulated by Government purchases at above-market prices. Because domestic chromite deposits are usually remote from consuming plants, very few operations can ship to consumers profitably in an open market; the high cost of transportation, added to the high cost of labor and the small size of most deposits, make competition with foreign ores extremely difficult.

Since opening of the General Services Administration's Chrome Purchase Depot at Grants Pass, Oreg., on August 3, 1951, to stimulate domestic production, the number of chrome producers has increased steadily from one small operator before the program to about 150 by the end of 1953. Production has been from California, Oregon, and Montana. All Montana production is from the Mouat mine operated by the American Chrome Co., under the stimulus of a loan and purchase contract with the Defense Materials Procurement Agency (and currently under the jurisdiction of the General Services Administration). Under the terms of the contract, 900,000 short dry tons of chrome concentrates having a minimum chromic oxide content of 38 percent is to be shipped to the Government within 8 productive years; this material is offgrade with a Cr-Fe ratio of about 1.5 : 1.

TECHNOLOGY

There is only one economic source of chromium—the mineral chromite. This mineral originates in peridotite rock (dunite, saxonite, etc.) or its alteration product, serpentine, both of which are readily recognized by prospectors. These rocks occur principally along mountain chains, ancient or modern, so the geographic distribution of possible chromium-bearing zones is much better known than for some other minerals.

The mineral chromite varies widely in composition, and its use is related thereto. It is basically a combination of the oxides of chromium, iron, aluminum, and magnesium within the formula $(Mg\ Fe)\ (Cr, Al, Fe)_2O_4$.

Over half of all chromite consumed is used for alloying purposes, about one-third goes into refractories, and chemicals manufacture takes the remainder. Although ores containing 48 percent and higher Cr_2O_3 and with a chrome-iron ratio of 3:1 are preferred by the metallurgical industry for making ferrochromium and other alloys, ores as low as 31 percent Cr_2O_3 are used for blending. Most ore used for refractory purposes is hard and lumpy and has a combined chromic oxide and alumina content of over 60 percent, the chromic oxide content being 31 to 33 percent; maximum silica and iron contents are about 5 and 10 percent, respectively. Some ores used in refractories contain up to 45 percent Cr_2O_3, with correspondingly lower alumina values and silica and iron percentages up to 11 and 19 percent, respectively. Transvaal chemical ore, averaging 44.5 percent Cr_2O_3 and with a Cr-Fe ratio of about 1.6:1, is the only chrome ore used for chemicals manufacture. The three types of ore can be interchanged to a certain extent; however, high alumina and silica ores are undesirable for chemical use. Limitations on the metallurgical use of lower grade ores are determined by price, effect on plant capacity, and type of product made.

Chromite is mined both from underground and opencut operations. When concentration is employed, gravity methods utilizing crushers, grinding mills, jigs, classifiers, and wet shaking tables are normally used. Heavy-medium separation and the Humphreys spiral concentrator are also used, as well as flotation and magnetic concentration methods.

In the production of chromium ferroalloys, used as chromium additives in steelmaking, chrome ore is always reduced in the electric furnace, and low-carbon ferrochromium is the chief product. This low-carbon alloy is usually made from chrome silicide (ferrochrome-silicon) and chromite. High-carbon ferrochromium is the next most important product. Chrome silicide and high-carbon ferrochromium are usually made directly from the ore. Since chrome silicide has a low carbon content and is considerably cheaper than low-carbon ferrochromium, a substantial quantity is sold as an alloying additive. Exothermic ferrochromium is used when the chilling effect of adding conventional ferrochromium to the molten metal is undesirable. One such product is composed of ferrochromium, a small quantity of ferrosilicon, and sodium nitrate and is produced in both low- and high-carbon grades. Exothermic silicon-chrome is another low-carbon additive available for steelmaking.

Some chromite is added directly to steel furnaces by a process first developed by the

TABLE 1.—*Principal United States consumers of chromite and producers of chromium products*

Company	Plant	Products
Metallurgical		
Baltimore Works of the Armco Steel Corp. (formerly Rustless Iron & Steel Co.).	Baltimore, Md	Stainless steel and iron.
Chromium Mining & Smelting Co.[1]	Riverdale, Ill	Exothermic high- and low-carbon ferrochromium; exothermic silicon-chrome alloy.
Electro Metallurgical Co	Niagara Falls, N. Y. Ashtabula, Ohio Holcombs Rock, Va Alloy, W. Va Marietta, Ohio	Low- and high-carbon ferrochromium; chromium briquets; chromium metal; ferrochrome-silicon alloys; exothermic ferrochromium; exothermic silicon-chrome alloy.
Metal & Thermit Corp.[1]	Carteret, N. J	Chromium metal.
Montana Ferro Alloys Co	Woodstock, Tenn	High-carbon ferrochromium; ferrochrome-silicon.
Ohio Ferro-Alloys Co	Brilliant, Ohio	Low- and high-carbon ferrochromium; ferrochrome-silicon.
Pacific Northwest Alloys, Inc	Mead, Wash	Low- and high-carbon ferrochromium.
Pittsburgh Metallurgical Co	Calvert City, Ky Charleston, S. C	High-carbon ferrochromium; ferrochrome-silicon.
	Niagara Falls, N. Y	High-carbon ferrochromium.
Vanadium Corp. of America	Niagara Falls, N. Y	Low- and high-carbon ferrochromium; chromium briquets; ferrochrome-silicon.
	Graham, W. Va	Low-carbon ferrochromium; ferrochrome-silicon.
Refractories		
Basic Refractories, Inc	Maple Grove, Ohio	
Botfield Refractories Co.[2]	Philadelphia, Pa	
General Refractories Co	Baltimore, Md Los Angeles, Calif	
Harbison-Walker Refractories Co	Baltimore, Md Warm Springs, Calif	
Kaiser Aluminum & Chemical Corp	Moss Landing, Calif	
E. J. Lavino & Co	Plymouth Meeting, Pa Newark, Calif	
Quigley Co.[2]	Old Bridge, N. J	
Frank Samuel & Co.[3]	Conshohocken, Pa	
Chemicals		
Columbia-Southern Chemical Corp. (formerly Natural Products Refining Co.).	Jersey City, N. J	
Diamond Alkali Co	Painesville, Ohio Kearny, N. J	
Imperial Paper & Color Corp	Glens Falls, N. Y	
Mutual Chemical Division, Allied Chemical & Dye Corp. (formerly Mutual Chemical Co. of America).	Baltimore, Md	

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[1] Does not use chromite.
[2] Consume processed chromite only, not raw chrome ore.
[3] Processes raw chromite for use by refractories manufacturers, steel companies, and miscellaneous users.

Rustless Iron & Steel Co. of Baltimore, Md. (now known as the Baltimore Works of the Armco Steel Corp.). In this process, chromite is added to the steel bath, and ferrosilicon acts as the reducing agent.

Chromium metal is currently being made by aluminothermic and electrolytic methods, the latter recently having been initiated commercially at a new plant at Marietta, Ohio, based on a method evolved by the Bureau of Mines. The aluminothermic method involves smelting chromic oxide (chrome oxide green) with aluminum powder to produce a metal containing a minimum of 97 percent Cr and a maximum of 1 percent Fe. The current commercial production of electrolytic metal uses high-carbon ferrochromium as a source of chromium and produces a material containing a minimum of 99 percent Cr. Electrolytic metal can also be made from chrome ore and from chrome chemicals.

Chromium refractories consist of bricks and ground preparations known as plastic mixes, cements, and ramming mixtures for lining furnace hearths, cementing bricks, and repairing furnace linings. The bricks have a wide range of chrome content, some being nearly all chromite but most being mixed with magnesia in all proportions. As chromite is the cheaper of these two ingredients, it is normally added in the maximum percentages feasible for a particular use. In making chrome refractory brick, hard, lumpy ore is preferred because the chromite is ground to certain specifications, as particle size is highly important. Chrome brick may be chemically bonded, burned, or unburned and cased in sheet metal, which fuses to the brick when in use.

Chromium chemicals are made by roasting finely ground chromite with soda ash and lime or soda ash alone, then water-leaching the resulting calcine containing an impure sodium chromate and acidifying the leach liquor to obtain pure sodium chromate or sodium bichromate, from which all other chromium chemicals are derived. Approximately 1.4 parts of chromite is required to produce 1 part of sodium bichromate.

SUBSTITUTES

Currently no commercial substitutes are known for most chemical and metallurgical uses of chromite. Titanium has been heralded as a possible replacement for stainless steel, but extractive titanium technology has not progressed to the point where that possibility need be considered here. For many refractory purposes, higher priced magnesia can be substituted for chromite. In answer to a specific question posed by the Bureau of Mines as to the possibilities for conserving refractory chrome ore in the steel industry, the Technical Committee on Refractories of the American Iron and Steel Institute stated in September 1950 that steel-plant consumption of chrome ore for refractory use could be reduced approximately 25 percent without adversely affecting steel production by substituting magnesite, in the main. The degree of substitution of magnesite for chromite is debatable; opinions range from virtually total substitution to considerably less than the 25 percent cited.

SECONDARY SOURCES AND RECOVERY

Secondary chromium is obtained only from the use of chromium-containing ferrous scrap. In 1952, an average of about 15,000 short tons of purchased stainless steel scrap was consumed per month; this scrap contained at least 10 percent chromium.

RESERVES [2]

United States inferred reserves of chrome ores currently acceptable to most industrial users total about 60,000 tons of contained Cr_2O_3.

TABLE 2.—*Salient statistics of chromite in the United States, 1949–53*

[Short tons]

	1949	1950	1951	1952	1953
Domestic production (mine shipments)	433	404	7, 056	21, 304	58, 817
Imports for consumption	1, 203, 852	1, 303, 713	1, 429, 020	1, 708, 969	2, 226, 610
Total new supply	1, 204, 285	1, 304, 117	1, 436, 076	1, 730, 273	2, 285, 427
Exports	2, 382	2, 044	2, 030	2, 531	1, 166
Consumption	672, 773	980, 369	1, 212, 480	1, 185, 460	1, 335, 755
Stocks Dec. 31 (consumers')	756, 995	606, 271	637, 453	754, 299	1, 015, 878
World production	2, 300, 000	2, 600, 000	3, 100, 000	3, 600, 000	4, 000, 000

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[2] It should be noted that estimates on ore reserves indicate order of magnitude only. Exploration in most countries has not been adequate, and the nature of many chromite deposits (pods, discontinuous stringers, disseminations, etc.) is such that most reserve figures are simply more or less educated guesses.

These ores occur in Alaska, California, and Oregon. Reserves amounting to 3,000,000 tons of Cr_2O_3 are contained in the extensive layered deposits of the Stillwater complex of Montana; these Montana ores are subgrade, however, and, although they can be used by the metallurgical, refractory, and chemical industries, they are not generally desired in current practice. Additional large reserves of subgrade material are contained in the ancient and modern black beach sands of the Oregon coast, which contain 2 to 8 percent Cr_2O_3 and can be beneficiated to yield about 250,000 tons of Cr_2O_3 in a 40-percent concentrate.

TABLE 3.—*Chromite production (mine shipments) in the United States, 1952–53, by States*

[Short tons]

State	1952		1953	
	Shipments	Value	Shipments	Value
California	14,713	[1] $1,269,000	26,512	$2,078,461
Montana	------	------	26,089	869,958
Oregon	6,591	507,981	6,216	484,453
Total	21,304	[1] 1,777,000	58,817	3,432,872

[1] Partly estimated.

Only a very small part of these several resources will yield usable lump ore. Concentrates averaging less than 40 percent Cr_2O_3 with a Cr-Fe ratio of 1.5 : 1 will comprise the main output.

Canada has about 10 million tons of subgrade ore similar to Montana's; this ore is in the Bird River area of Manitoba and contains about 2 million tons of Cr_2O_3.

World resources of chromite have been estimated at about 300 million tons of contained Cr_2O_3 (*21*).[3] That reserves of the three ore types are not proportionate to the consumption requirements is evident from the following comparison of the estimated distribution of reserves by grade of ore or potential concentrates with the present consumption pattern:

	Percent	
Grade in current practice:	*Reserves*	*Consumption*
Metallurgical	12–15	50–52
High-alumina refractory	3–5	32–33
High-iron chemical	55–60	14–18
Laterites	25–30	0

It is estimated that about 80 percent of world lode chromite reserves (exclusive of laterites) are in the Transvaal (Union of South Africa) and Southern Rhodesia (Federation of Rhodesia and Nyasaland) combined. Southern Rhodesia's reserves of standard and marginal metallurgical ore in the Great Dyke alone total several million tons. Reserves of chemical ore in the Great Dyke and in the Transvaal's Bushveld complex run to hundreds of millions of tons, in layers that range from a few inches up to 4 or 5 feet thick, extend for miles, and can be mined as bedded deposits. All current production of ore used by the chemical industry comes from the Transvaal, partly because of the limited transportation facilities serving the Rhodesian deposits. Some high-chromium ores are found in the Bushveld complex, but the Cr-Fe ratio averages only about 1.9 : 1; these ores are used in ferroalloy manufacture.

Turkey is the world's greatest source of high-grade metallurgical ore; but its reserves, although not adequately defined, are not deemed to be extensive enough to maintain normal production for more than a generation. Turkish reserves have been estimated at about 6 million short tons; this is considered conservative. Most of this material is metallurgical grade, but there is some refractory ore also.

New Caledonia is the only other current major source of high-grade metallurgical chromite. Data on actual reserves are not available, but exploration during the last decade has apparently indicated a substantial tonnage. Over half a million short tons of chromite has been produced in the past 5 years; output apparently is capable of increasing considerably with demand.

As far as current United States requirements are concerned, the Republic of the Philippines is the principal source of refractory chrome ore, with reserves of about 8 million metric tons, mostly at the Masinloc deposit in the Zambales Mountains of Luzon. There are also about 1 million tons of metallurgical ore, most of it at the Acoje deposit on Luzon.

Cuba is the second largest source of refractory ore, and its inferred potential resources total 2.5 million short tons containing about 750,000 tons of Cr_2O_3; 1.75 million tons is in the Camaguey district and 750,000 tons in Oriente Province. Oriente also contains 150,000 tons of low-grade metallurgical chromite containing about 45,000 tons of Cr_2O_3. During World War II Cuban production reached a maximum of 390,000 short tons in 1943. Because of high labor costs and somewhat inferior-grade ore due to higher silica content, the Camaguey district could not compete with the Philippines and was closed down soon after the war ended.

Additional deposits of high-grade metallurgical ore occur in Pakistan, Iran, and Afghanistan; the last has an estimated 30,000 tons of high-grade chromite out of an estimated 180,000 tons of reserves. The potential in these 3 countries is considered to be fairly large. Yugoslavia has about 2,000,000 tons of chromite, Brazil 150,000 tons, and Greece an estimated

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[3] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

TABLE 4.—*World production of chromite, by countries,*[1] *1949–53*

[Metric tons]

(Compiled by Berenice B. Mitchell)

Country [1]	1949	1950	1951	1952	1953
North America:					
Canada	327	--------	--------	--------	--------
Cuba	97, 368	65, 820	79, 065	61, 808	70, 039
Guatemala	300	289	1, 138	105	400
United States	393	367	6, 401	19, 327	53, 358
South America: Brazil	[2] 3	3, 227	2, 416	[2] 420	[2] 910
Europe:					
Albania	(3)	[4] 52, 000	(3)	(3)	(3)
Greece	3, 381	12, 631	25, 333	32, 162	21, 924
Portugal	88	45	33	108	(3)
U. S. S. R. [4] [5]	350, 000	500, 000	600, 000	600, 000	600, 000
Yugoslavia	109, 120	114, 736	99, 639	107, 222	126, 961
Asia: [5]					
Afghanistan	1, 000	550	75	--------	--------
Cyprus (exports)	14, 875	18, 441	12, 653	13, 487	8, 269
India	19, 728	16, 998	16, 970	[6] 36, 768	[4] 20, 000
Iran	--------	--------	(3)	8, 825	20, 000
Japan	27, 108	32, 964	40, 944	47, 151	37, 632
Pakistan	17, 194	18, 416	18, 006	17, 706	23, 818
Philippines	246, 744	250, 511	334, 571	543, 571	557, 090
Turkey	451, 566	420, 529	602, 220	675, 156	[4] 690, 000
Africa:					
Egypt	50	36	--------	--------	--------
Sierra Leone	22, 101	7, 518	16, 455	23, 870	24, 745
Southern Rhodesia	243, 506	291, 525	300, 267	322, 667	420, 052
Union of South Africa	404, 351	496, 324	545, 306	580, 024	724, 444
Oceania:					
Australia	642	905	1, 402	1, 402	[4] 2, 000
New Caledonia	88, 992	84, 801	88, 792	107, 653	121, 060
Total (estimate)	2, 100, 000	2, 400, 000	2, 800, 000	3, 300, 000	3, 600, 000

[1] In addition to countries listed, Argentina, Bulgaria, and Rumania produce chromite, but data on output are not available; estimates by author of chapter included in total.

[2] Exports.

[3] Estimate.

[4] Data not available; estimate by author of chapter included in total.

[5] Output from U. S. S. R. in Asia included with U. S. S. R. in Europe.

[6] Does not include 21,603 tons of low-grade ore accumulated from production from 1943 through 1948.

million tons, two-thirds of which is refractory. India, Japan, and Sierre Leone also have substantial reserves.

Within the Soviet bloc, the U. S. S. R. has extensive reserves of high-grade metallurgical and refractory ores. Albania also is believed to have reserves, and Rumania has a quantity of very poor grade material, extremely high in iron.

Chromiferous laterites are found principally in Cuba, the Philippines, Celebes, French West Africa, and the U. S. S. R. Cuban reserves of such material are estimated at 3 billion tons containing about 50 million tons of Cr_2O_3.

USES

Most chromium consumption is closely related to the metallurgical industry, despite the fact that the ore is used by the refractory and chemical industries as well. Distribution of chrome-ore consumption in the United States, percentagewise, reveals that in 1953, 56 percent was used in ferroalloys manufacture, 33 percent in the making of refractories, and 11 percent in chemical production. Most of the chrome refractories (brick and cement) are used by the metallurgical industry for lining and patching furnaces, and a fair percentage of the chemicals is used for metal-treating and plating and the manufacture of chromium metal.

Although the largest and best known use of chromium is in the manufacture of stainless steels, its qualities of resistance to corrosion, heat, friction, impact, creep, and wear make its use essential in making high-strength, low-alloy steels for structural purposes, in high-temperature alloys for jet engines and gas turbines, in high-speed tool steels, and in various other alloys. Chromium plating from chromic acid solutions for engineering as well as decorative purposes is an important use, though tonnagewise its consumption of chromium is relatively unimportant. Additional uses of chromium chemicals are in leather-tanning solutions, pigments, pyrotechnics, photographic supplies, etc. It has been reported

TABLE 5.—*Consumption of chromite and tenor of ore used by primary consumer groups in the United States, 1949–53*

[Short tons]

	Metallurgical		Refractory		Chemical		Total	
	Gross weight (short tons)	Average Cr_2O_3 (percent)	Gross weight (short tons)	Average Cr_2O_3 (percent)	Gross weight (short tons)	Average Cr_2O_3 (percent)	Gross weight (short tons)	Average Cr_2O_3 (percent)
1949	288, 518	47. 6	268, 925	33. 5	115, 330	44. 1	672, 773	41. 3
1950	491, 685	47. 8	353, 642	34. 0	135, 042	44. 6	980, 369	42. 4
1951	573, 075	48. 1	440, 771	34. 7	198, 634	44. 3	1, 212, 480	42. 6
1952	676, 624	47. 1	387, 085	33. 8	121, 751	44. 4	1, 185, 460	42. 9
1953	742, 822	46. 3	441, 155	33. 6	151, 778	44. 5	1, 335, 755	42. 7

TABLE 6.—*Chromite imported for consumption in the United States, 1952–53, by countries, and by grades*

[Short tons]

(Compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce)

Grade and country	1952			1953		
	Gross weight	Cr_2O_3 content	Value	Gross weight	Cr_2O_3 content	Value
Metallurgical:						
Afghanistan	1, 006	464	$25, 875	47	28	$1, 987
Cuba	37, 879	14, 921	1, 114, 602	29, 656	11, 721	822, 057
Greece	246	118	7, 920	375	164	12, 920
Guatemala	1, 458	846	51, 050	----------	----------	----------
India	6, 581	2, 931	205, 748	6, 782	3, 069	240, 820
Iran	----------	----------	----------	5, 816	2, 912	209, 072
New Caledonia [1]	58, 776	29, 778	2, 072, 175	98, 877	48, 737	3, 667, 256
Pakistan	3, 372	1, 577	132, 355	14, 038	6, 316	534, 118
Philippines	41, 175	19, 490	1, 117, 833	84, 935	38, 938	2, 408, 876
Sierra Leone [2]	26, 846	10, 769	850, 810	15, 008	6, 425	490, 424
Southern Rhodesia	172, 515	82, 514	4, 813, 553	325, 836	152, 947	10, 169, 223
Turkey	460, 236	211, 301	16, 166, 896	565, 120	259, 718	21, 739, 645
Union of South Africa	87, 990	40, 253	1, 125, 828	198, 114	90, 551	3, 253, 126
Yugoslavia	21, 661	9, 511	876, 599	48, 675	22, 099	1, 769, 168
Total	919, 741	424, 473	28, 561, 244	1, 393, 279	643, 625	45, 318, 692
Refractory:						
Cuba	58, 875	21, 396	1, 028, 127	57, 630	19, 644	1, 086, 518
Greece	----------	----------	----------	1, 003	401	27, 300
India	----------	----------	----------	7, 842	3, 188	111, 864
Philippines	502, 527	170, 662	6, 790, 447	544, 523	177, 566	7, 256, 195
Southern Rhodesia	11, 057	4, 784	214, 831	2, 530	1, 045	41, 699
Turkey	3, 343	1, 313	33, 441	----------	----------	----------
Union of South Africa	22, 610	9, 355	217, 147	14, 835	6, 069	164, 831
Total	598, 412	207, 510	8, 283, 993	628, 363	207, 913	8, 688, 407
Chemical: Union of South Africa	190, 816	84, 502	1, 749, 838	204, 968	91, 087	2, 033, 620
Total all grades:						
Afghanistan	1, 006	464	25, 875	47	28	1, 987
Cuba	96, 754	36, 317	2, 142, 729	87, 286	31, 365	1, 908, 575
Greece	246	118	7, 920	1, 378	565	40, 220
Guatemala	1, 458	846	51, 050	----------	----------	----------
India	6, 581	2, 931	205, 748	14, 624	6, 257	352, 684
Iran	----------	----------	----------	5, 816	2, 912	209, 072
New Caledonia [1]	58, 776	29, 778	2, 072, 175	98, 877	48, 737	3, 667, 256
Pakistan	3, 372	1, 577	132, 355	14, 038	6, 316	534, 118
Philippines	543, 702	190, 152	7, 908, 280	629, 458	216, 504	9, 665, 071
Sierra Leone [2]	26, 846	10, 769	850, 810	15, 008	6, 425	490, 424
Southern Rhodesia	183, 572	87, 298	5, 028, 384	328, 366	153, 992	10, 210, 922
Turkey	463, 579	212, 614	16, 200, 337	565, 120	259, 718	21, 739, 645
Union of South Africa	301, 416	134, 110	3, 092, 813	417, 917	187, 707	5, 451, 577
Yugoslavia	21, 661	9, 511	876, 599	48, 675	22, 099	1, 769, 168
Grand total	1, 708, 969	716, 485	38, 595, 075	2, 226, 610	942, 625	56, 040, 719

[1] Assumed source: classified in import statistics under "French Pacific Islands."

[2] Assumed source: classified in import statistics under "British West Africa."

that 38 percent of the total production of chromium chemicals is being used in pigment manufacture, 21 percent in surface treating and plating of metals, 18 percent in leather tanning, 3 percent each in textiles and chemical manufacture, and the remainder for unknown and miscellaneous purposes. All-chromite refractory brick is used as one of the layered constituents of open-hearth furnace bottoms, but elsewhere, brick containing both chromite and magnesia is generally employed. Since chrome brick is chemically neutral, it is used to separate acid roofs from basic bottoms in open-hearth furnaces. Additional uses are in copper furnaces, glass furnace checkers, cement kilns, and the paper industry.

STATISTICAL INFORMATION

Statistical data are obtained monthly from industry by the Bureau of Mines on domestic production, consumption, and inventories of chrome ore, metal, and alloys, as well as some additional information on the end use of chromium alloys and metal and the production of chromium chemicals.

United States import data are obtained by the Bureau of the Census from Customs receipts and compiled with some assistance from the Bureau of Mines.

SELF-SUFFICIENCY

American chromite requirements are met almost entirely by imports. Although reserves of offgrade and currently undesirable material are substantial, enough production to provide 10 percent of requirements is possible only by high incentive prices. The same applies to the small reserve of specification grade material, production of which was only a little over 2 percent of requirements in 1953, despite the subsidy prices.

STRATEGIC CONSIDERATIONS

Chromite requirements in wartime are much greater than during normal periods of high industrial activity. Because all important sources of the better grades of chromite are at the end of long sea voyages, the available supply in wartime cannot be relied on to meet demand. Therefore, large tonnages of chromite are needed for stockpiling. Concentrating and consuming plants in the United States are widely scattered and do not present an unusual plant-protection problem.

PRICES

Toward the close of 1953 the prices of some foreign chromites quoted in E&MJ Metal and Mineral Markets dropped \$2 to \$4 (see table 8). According to other sources, however, chromite prices as a whole were stable throughout the year. The apparent conflict is undoubtedly due to the fact that much of the ore is committed on a long-term contract basis. Recent reports indicate current prices of \$21 to \$22 for South African 44-percent grade (Chemical), a nominal price of \$45 to \$46 for Turkish chromite with 48 percent Cr_2O_3 and a 3 : 1 Cr-Fe ratio (Metallurgical grade), and a long-term contract price of \$43 to \$44 for Rhodesian ore of the same Metallurgical grade. A nominal price of \$26 per long dry ton was reported by American

TABLE 7.—*Prices per long dry ton offered by General Services Administration for domestic chromite (base 48 percent Cr_2O_3, chrome-iron ratio 3:1) delivered to Grants Pass, Oreg., 1951–57* [1] [2]

Chrome-iron ratio	Chromic oxide (Cr_2O_3), percent												
	42	43	44	45	46	47	48	49	50	51	52	53	54
3.5:1	\$117	\$120	\$123	\$126	\$129	\$132	\$135	\$139	\$143	\$147	\$151	\$155	\$159
3.4:1	113	116	119	122	125	128	131	135	139	143	147	151	155
3.3:1	109	112	115	118	121	124	127	131	135	139	143	147	151
3.2:1	105	108	111	114	117	120	123	127	131	135	139	143	147
3.1:1	101	104	107	110	113	116	119	123	127	131	135	139	143
3.0:1	97	100	103	106	109	112	115	119	123	127	131	135	139
2.9:1	94	97	100	103	106	109	112	116	120	124	128	132	136
2.8:1	91	94	97	100	103	106	109	113	117	121	125	129	133
2.7:1	88	91	94	97	100	103	106	110	114	118	122	126	130
2.6:1	85	88	91	94	97	100	103	107	111	115	119	123	127
2.5:1	82	85	88	91	94	97	100	104	108	112	116	120	124
2.4:1	79	82	85	88	91	94	97	101	105	109	113	117	121
2.3:1	76	79	82	85	88	91	94	98	102	106	110	114	118
2.2:1	73	76	79	82	85	88	91	95	99	103	107	111	115
2.1:1	70	73	76	79	82	85	88	92	96	100	104	108	112
2.0:1	67	70	73	76	79	82	85	89	93	97	101	105	109

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[1] Fractions appearing on analysis reports are prorated in computing premiums and penalties.

[2] Prices shown are for lump ore; payments for fines and concentrates are \$5 a ton lower.

Metal Market for refractory ore containing 34 percent Cr_2O_3.

Virtually all domestically produced chromite is purchased by the Government at incentive prices. A base price of $115 per long dry ton is paid for metallurgical lump ore containing 48 percent Cr_2O_3 and having a 3 : 1 Cr-Fe ratio. (Prices for various specifications are listed in table 7.) Offgrade Montana concentrates containing 38 percent Cr_2O_3 and having a low Cr-Fe ratio were bought by the Government at $34.97 per short dry ton ($39.17 per long dry ton).

TABLE 8.—*Price quotations for various grades of foreign chromite in 1953*

(E&MJ Metal and Mineral Markets)

Source	Cr_2O_3 (percent)	Cr-Fe ratio	Price per long ton [1]	
			Jan. 1	Dec. 31
Pakistan	48	[2] 3:1	$53-$54	$51-$52
Rhodesia	48	[2] 3:1	44- 46	44- 46
Do	48	2.8:1	40- 42	40- 42
Do	48	---	32- 33	32- 34
South Africa (Transvaal)	48	---	34- 35	33- 34
Do	44	---	27- 28	23- 24
Turkey	48	[2] 3:1	55- 56	53- 54

[1] Quotations are on a dry basis, subject to penalties if guarantees are not met, f. o. b. cars, east coast ports.

[2] Lump ore.

Prices of ferrochromium have experienced no change since November 25, 1952, when the Office of Price Stabilization increased the price of high-carbon ferrochromium (65 to 69 percent Cr, 4 to 9 percent C) from 21¾ cents per pound of contained chromium to the present 24¾ cents per pound, f. o. b. destination, continental United States, carload bulk, according to E&MJ Metal and Mineral Markets; low-carbon ferrochromium rose from 30½ cents to 34½ cents per pound of contained chromium for the 0.06-percent-carbon grade. Chromium metal is currently quoted at $1.16 per pound for 97-percent grade with 0.5 percent carbon and $1.25 for metal with 9 to 11 percent carbon, wholesale lots, f. o. b. New York; electrolytic chromium, commercial grade, 99 percent minimum, is quoted at $1.16 per pound compared to the minimum price of $3.00 per pound before September 1954. Current basic chrome-brick prices are $80 per short ton for burned brick and $86 for chemical-bonded brick, f. o. b. shipping point.

TARIFF

No import duty is imposed on chrome ores. In accordance with the Tariff Act of 1930 as superseded by various trade agreements, the following tariffs are imposed on imports of chromium products from nations signatory to those agreements:

Ferrochromium containing 3 percent carbon or over is subject to a tariff of ⅝ cent per pound of contained chromium. The tariff rate on ferrochromium containing less than 3 percent carbon, chromium metal, chromium carbide, ferrochrome silicon and chrome silicide, chromium nickel, and chromium vanadium is 12½ percent ad valorem. Alloys that number chromium and at least one of the following elements among their constituents are dutiable at 20 percent ad valorem: Calcium, zirconium, titanium, barium, boron, strontium, thorium, or vanadium. Chromic acid and chromic oxide and other chrome colors enter at 12½ percent ad valorem.

Imports from all countries are subject to the same tariff rate for chrome bricks and shapes (25 percent ad valorem), sodium chromate and bichromate (1¾ cents per pound), and potassium chromate and bichromate (2¼ cents per pound).

Imports from countries not participating in the aforementioned agreements (such as the Soviet Union) are subject to tariffs of 2½ cents per pound of contained chromium for high-carbon ferrochromium, 30 percent ad valorem for low-carbon ferrochromium and chromium metal, and 25 percent ad valorem for everything else.

TRANSPORTATION

Lack of adequate transportation facilities is an important limiting factor in the acquisition of chromite imports from the Union of South Africa and Southern Rhodesia. Both countries have stockpiles of about one-half million tons each, awaiting shipment to the two ports in Mozambique—Lourenço Marques and Beira. Transportation facilities elsewhere are generally adequate.

CONSERVATION PRACTICES

As with any relatively high price item, conservation of chromium is inherent in the cost. Ferroalloy manufacturers and steelmakers endeavor to extract the maximum percentage of chromium from the molten slag in their metallurgical processes. Chemicals manufacturers rework their residue muds to permit only a minimum loss of chromium in their processes.

Chromium could be conserved during wartime by limiting the supply of stainless steel and other alloy steels for civilian consumption and by restricting the quantity of chromium chemicals available for pigments, dyes, pyrotechnics, and tanning to essential uses only. Paradoxically, however, chromium is a specific in the conservation of other materials as well as labor and power. By reducing wear and corrosion, replacement of material is minimized, as is the labor required to do so; in other instances such as the use of chromium solutions in leather tanning, chromium hastens processes considerably. Conservation of chromium therefore must be considered in the light of its effect on other factors of the general economy.

RESEARCH

Experimentation has been and is being conducted by the Bureau of Mines on the use of offgrade domestic chromites in making chromium ferroalloys, refractories, and chemicals. Industry has cooperated in the use-testing of some resulting products, such as high-carbon ferrochromium, chrome silicide, and refractory chrome-magnesia brick; all were found to be satisfactory for certain uses. Additional research is being conducted on the recovery and utilization of the chromium contained in Cuban laterites. It is assumed from the increasing use of the South African chemical ores and other submetallurgical-grade ores by the metallurgical industry, as well as the lowering of the average chrome content of the alloys produced, that industry researchers are seeking to broaden the ore-requirements base as a matter of economic necessity now, and absolute necessity in the foreseeable future.

OUTLOOK

SIGNIFICANT CURRENT TRENDS

Expanded uses for chromium and approaching depletion of the world's high-grade ore reserves are the important trends in the chromium industry. Preparing for the inevitable shortage of high-grade chromite, the ferroalloy manufacturers are using lower grade ores in making lower chromium content ferrochrome alloys; for example, one plant is now making high-carbon ferrochromium containing 56 percent Cr as opposed to the standard 68 percent product formerly made. More interest is also being shown in low-carbon ferrochrome-silicon alloys, which contain 30 to 56 percent chromium.

FORECAST OF SUPPLIES AND REQUIREMENTS

Chromite requirements in 1975 probably will double the 1950 level. It is expected that many new uses for chromium will be found; and those current will be expanded, such as in the production of high-temperature alloys for use in jet engines and gas turbines.

Refractory chrome products are used chiefly in lining steel furnaces; therefore the use of refractory chromite will increase as steel production rises.

Producing chrome chemicals at considerably less than capacity and with adequate supplies of chromite to draw from, the chemical industry will probably begin to manufacture chromium metal as an outlet for its chemicals. The research of the Bureau of Mines and others in creating high-purity ductile chromium metal will increase the demand for the metal.

At the peak consumption rate of 1953, known refractory ores will last 20 to 30 years. Metallurgical ores will be augmented increasingly by lower grades, because, as reserves decline, the high-grade deposits will be unable to maintain production at the required rate. Prices will increase as mining becomes more difficult, creating an additional incentive for using cheap and plentiful lower grade ores. The depletion of shipping-grade ores will be accompanied by an increase in the ratio of concentrates to lump ore.

The principal world sources of chromite for the next decade will continue to be about as they have been for the last 10 years, with a significant shift away from the present sources of ore for metallurgical use within a generation.

PROBLEMS

World reserves of special grades of chromite, such as the high-chrome, low-iron "Metallurgical grade" and the high-alumina, low-iron"Refractory grade" are gradually being depleted. This will eventually necessitate use of the lower grade ores for all purposes, creating technical and economic problems for the refractory and metallurgical industries.

Use of lower grade chrome ores for ferroalloy manufacture results in lower chrome-content ferroalloys. This will require modification of procedures by the consumers of ferroalloys. Use of chrome concentrates for ferroalloy manufacture will require special procedures to prevent dust losses and unusual losses to the slag caused by entrapment of the fine chromite particles.

Virtually all chromite consumed in the United States is obtained from foreign sources; and conditions within supplying countries, such as the shortage of transportation, can interrupt the United States supply. Dependence of the industry on only 1 or 2 distant sources for a specific grade of chromite increases the tenuity of the supply position. European consumers are steadily expanding their requirements for chromite, creating greater competition with those in the United States.

Domestic reserves of chromite are relatively small compared with requirements, and many are offgrade. The largest potential source of chromium in the Western Hemisphere is Cuba's lateritic iron ore, but this material cannot be considered a reserve until satisfactory methods are devised for utilizing it.

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CLAYS

By

Brooke L. Gunsallus[1]

CLAY occurs in one form or another in all countries of the world. Man's utilization of clay has progressed from historical clay tablets, household utensils, and building brick of early civilizations to the present high level of technically supervised manufacturing. Despite the vast amount of modern research, improved processes, technical control, and mass-production methods in all branches of the clay-working industry of today, there remains a host of problems to challenge ceramists for generations.

Summary

Clays are hydrous aluminum silicates mixed with varying proportions of impurities. These silicates occur in numerous mineral forms with varied properties and give the various clays used in industry.

The principal industrial clays are kaolin or china clay, ball clay, fire clay, bentonite, fuller's earth, and miscellaneous clays. About 75 percent of all clay produced is used in manufacturing formed and fired ceramic products.

The total clay sold or used by producers in the United States in 1953 was 42.4 million short tons valued at over 125 million dollars. Formerly, large quantities of kaolin were imported, but in 1953 only about 6 percent of the domestic consumption was imported. Georgia produced about 71 percent, South Carolina 17 percent, and North Carolina, Pennsylvania, Florida, Alabama, Illinois, Nevada, California, Utah, and Virginia 12 percent of the domestic kaolin output. The only commercial domestic output of catalytic-grade halloysite (one of the kaolin minerals) came from Utah. About 90 percent of the ball-clay production came from western Kentucky and western Tennessee.

Over half of the domestic consumption of kaolin is as a filler or coating in paper. Other important uses are in rubber, refractories, and whitewares. Ball clay and fire clay are used almost exclusively by the ceramic industry. Bentonite and fuller's earth are used for clarifying and cracking petroleum products, foundry facings, drilling muds, and numerous other nonceramic purposes. Miscellaneous clays are used for heavy clay products and cement. The properties of all these clays vary widely with the different types and also vary within the same type.

Where clays are used in ceramics, significant properties include plasticity, dry and fired color, dry and fired strength, dry and fired shrinkage (or expansion), fired apparent porosity, density, softening point on firing, vitrification range, bloating, adsorption, and particle size and shape.

In the past, control and evaluation of clay properties, particularly in the ceramic industries, have been made by simulated service tests that often are cumbersome and expensive. In recent years, because of increased production and the need for more precise controls and manufacturing procedures, more

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[1] Commodity-industry analyst, Bureau of Mines.

scientific methods have become desirable. There has been a large expansion in ceramic research conducted by industry and by Government, educational, and private institutions. Much of this research has involved studies of the fundamental properties of clays and of the crystalline structure of clay minerals.

Total clay production will continue to increase with industrial expansion and growth in population. The rate of increase may not be the same for all kinds of clay because of the variable demand in use, difference in availability of sources, and the degree of interchangeability.

Regional shortages exist in certain kinds of high-grade clays. Freight rates, for the most part, are a governing factor in determining the shipping radius. Only clays of high unit value can be marketed at points remote from the source of supply. Comprehensive investigations are needed in some areas to determine the requirements for types of clay in local short supply.

BACKGROUND

Clays are hydrous aluminum silicates of a great many mineral species, containing varying proportions of impurities. No two clays are exactly alike; and it is seldom possible to substitute one clay for another in industry without considerable experimentation, even though extensive research data have been established.

Clays are classified in this chapter as kaolin, ball clay, fire clay, bentonite, fuller's earth, and miscellaneous clays, in accordance with a simplified classification used by the Bureau of Mines for statistical canvassing. In a well-known classification by ceramic properties (*15*),[2] clays are subdivided into 28 classes and numerous subclasses.

Although by far the greater part of clay consumption is in the manufacture of formed and fired ceramic products, large quantities are used in cement manufacture, paper manufacture, rotary-drilling mud, fillers, filtering and decolorizing agents, insecticide and fungicide carriers, absorbents, and numerous other uses.

The total clay sold or used by producers in the United States in 1953 was 42.4 million short tons, valued at over 125 million dollars. Distribution of the quantities of clay used, by kinds and uses, is given in table 1.

GENERAL PROPERTIES

Although clays, even within a group classification, vary widely in physical and chemical properties, a number of attributes to some degree are common to all. For most of the uses of clay chemical composition is unimportant, except when knowledge of the quantity of certain major constituents or impurities may help to qualify a clay sample for further investigation in a specific category. Even then the cost of an analysis may exceed that of a simple ceramic test that might be more meaningful to a competent clayworker or technologist. Chemical analyses sometimes are useful for determining the cause of manufacturing troubles and may indicate possible methods of overcoming them. Chemical analyses are more important in controlling clays for use in refractories, since analysis of the products is frequently stipulated.

Many significant properties of clays are evaluated by simulated service tests, some of which are cumbersome and time- and labor-consuming. Although ceramic technology has advanced rapidly, there is still room for the judgment of individuals, based on visual observation. For example, the quantity of water used with clays may be carefully metered, but the final judgment as to whether the consistency is correct for ware forming depends on frequent tests by feel and visual observation.

The significant properties of clays include plasticity, dry and fired color, dry and fired strength, dry and fired shrinkage (or expansion), dry and fired apparent porosity, softening point on firing (pyrometric cone equivalent—P. C. E.), vitrification range, bloating properties, adsorption, particle shape, particle size, and many others.

Plasticity is the capability of a material to be deformed continuously and permanently in any direction without rupture, under a stress exceeding the yield value. Many clays develop good plasticity upon the addition of water, some develop only poor plasticity, and others (such as flint clays) are so indurated that they require extremely fine grinding to develop even weak plasticity. The dry strength and drying shrinkage and sometimes the fired strength and firing shrinkage may be related to plasticity.

Vitrification range is the temperature range throughout which the product maintains commercially satisfactory properties, usually representing approximately the minimum absorption resulting from development of essentially the maximum firing shrinkage.

Refractoriness is the most important property of fire clays and also is important in other clays used in forming ceramic bodies. The most significant measure of refractoriness is the pyrometric cone equivalent (P. C. E.). A series of triangular pyramids known as pyrometric cones is used to measure time-temperature effects on ceramic mixtures. Each cone bends at a known temperature when heated under standard conditions of time, temperature, and atmosphere. The American Society for Testing Materials has classified fire clays used to produce refractories in part by P. C. E. values (*4*).

The ability of refractories to resist spalling is a complex phenomenon related to the coefficient of expansion, strength under service conditions, and other properties, including the

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[2] Figures in parentheses refer to items in the bibliography at end of this chapter.

TABLE 1.—*Clays sold or used by producers in the United States in 1953, by kinds and uses*

[Short tons]

Use	Kaolin	Ball clay	Fire clay and stoneware clay	Bentonite	Fuller's earth	Miscellaneous clays, including slip clay	Total
Pottery and stoneware:							
Whiteware, etc	125, 065	209, 825	482				335, 372
Stoneware, including chemical stoneware	12, 227	475	56, 430	99		947	70, 178
Art pottery and flower pots	4, 209	9, 416	10, 716			43, 403	67, 744
Slip for glazing	400	500		41			941
Total	141, 901	220, 216	67, 628	140		44, 350	474, 235
Tile, high-grade	26, 896	40, 438	103, 854			59, 059	230, 247
Kiln furniture:							
Saggers, pins, and stilts	6, 788	4, 916	11, 857				23, 561
Wads						1, 650	1, 650
Total	6, 788	4, 916	11, 857			1, 650	25, 211
Architectural terra cotta			39, 150				39, 150
Paper:							
Filler	437, 151						437, 151
Coating	558, 194			2, 314			560, 508
Total	995, 345			2, 314			997, 659
Rubber	241, 052		18, 475	103			259, 630
Linoleum	38, 510		10, 427	120			49, 057
Paints:							
Filler or extender	28, 782		432	101		543	29, 858
Calcimine	1, 400			68			1, 468
Total	30, 182		432	169		543	31, 326
Portland and other hydraulic cements	46, 128		17, 041	185		8, 539, 981	8, 603, 335
Refractories:							
Firebrick and block	149, 386	16, 118	3, 817, 595	1, 180		3, 359	3, 987, 638
Bauxite, high-alumina brick			61, 661				61, 661
Fire-clay mortar, including clay processed for laying firebrick	25, 072	2, 500	490, 292			2, 557	520, 421
Clay crucibles	208	50					258
Glass refractories	1, 015		77, 736				78, 751
Zinc retorts and condensers			51, 900	7, 524			59, 424
Foundries and steelwork	1, 160	500	733, 019	347, 056		14, 670	1, 096, 405
Other refractories	2, 401		137, 406			2, 184	141, 991
Total	179, 242	19, 168	5, 369, 609	355, 760		22, 770	5, 946, 549
Heavy clay products: Building brick, paving brick, drain tile, sewer pipe, and kindred products		2, 615	4, 534, 069			18, 243, 673	22, 780, 357
Miscellaneous:							
Rotary-drilling mud				583, 373	52, 429	167, 978	803, 780
Filtering and decolorizing oils (raw and activated earths)	66, 554			251, 107	[1] 145, 666		463, 327
Other filtering and clarifying				5, 628	7, 709		13, 337
Artificial abrasives	150		885				1, 035
Absorbent uses (oily floors, etc.)	1, 185			40	130, 108		131, 333
Asbestos products	2, 350		291				2, 641
Chemicals	4, 382		91, 891	13, 819			110, 092
Enameling		2, 074		129			2, 203
Fertilizers	11, 106			40	1, 800	188	13, 134
Filler (other than paper or paint)	15, 304	11, 335		9, 862	13, 423	6, 097	56, 021
Insecticides and fungicides	31, 810			4, 621	75, 695	187	112, 313
Plaster and plaster products	3, 702			21			3, 723
Lightweight aggregate						1, 166, 553	1, 166, 553
Other uses	41, 387		1, 504	42, 540	9, 007	25, 278	119, 716
Total	177, 930	13, 409	94, 571	911, 180	435, 837	1, 366, 281	2, 999, 208
Grand total:							
1953	1, 883, 974	300, 762	10, 267, 113	1, 269, 971	435, 837	28, 278, 307	42, 435, 964
1952	1, 829, 102	305, 083	11, 285, 173	[2] 1, 317, 979	422, 853	[2] 26, 507, 910	41, 668, 100

[1] Comprises the following: Mineral oils, 92,107 tons; vegetable oils, 17,979 tons.

[2] Revised figures.

presence or absence of mineral inversion points, accompanied by volume changes.

Clays and shales of the same types employed for manufacturing brick, tile, sewer pipe, and similar wares frequently have natural bloating properties that make them suitable for the production of lightweight aggregates. The constituents responsible for the bloating are other minerals and materials associated with the clay minerals. These include carbonaceous material, various iron compounds, limestone, dolomite, and gypsum (potential sources of gases for bloating) and feldspars, micas, and free quartz, with uncertain roles in the bloating phenomenon.

The color of dry clay is important for certain filler applications, such as paper and paint, but for most uses it is unimportant. Iron compounds and organic matter are the common colorants. A low content of iron compounds, titania, and other metal oxide colorants is essential to whiteness in fired products.

RESEARCH

In the past, clay-research approach was mostly empirical. Technical knowledge was kept a secret on a family, plant, or company basis, causing resistance to technical changes such as the substitution of new clays in ceramic body mixes.

In recent years the volume of ceramic research conducted by Government agencies and by educational, industrial, and other private institutions, has expanded remarkably. Much of this research has involved studies of the fundamental properties of clays and of the crystalline structure of clay minerals by microscope, X-ray, thermal expansion, and differential thermal analysis methods (*1*). There also have been extensive and diversified studies of raw materials utilization and product development.

KAOLIN

Kaolin (china clay) is a relatively pure clay material, usually consisting predominantly of kaolinite ($Al_2O_3.2SiO_2.2H_2O$) and related hydrous aluminum silicates. The common kaolin minerals other than kaolinite are halloysite, dickite, and anauxite, any of which may predominate in some deposits (*16*). Catalytic-grade halloysite is a halloysite with high natural catalytic activity that can be used to manufacture a catalyst for use in hydrogenation or cracking of high-sulfur petroleums.

Kaolins are not uniform in physical properties, but the chemical composition ranges within narrow limits. Good grades of kaolin contain 37 to 40 percent alumina (Al_2O_3), 45 to 55 percent silica (SiO_2), up to about 1 percent iron reported as ferric oxide (Fe_2O_3), minor titania (TiO_2), alkaline-earth and alkali metal oxides, and the balance, 8 to 15 percent, loss on ignition. Whiteness in the dry or fired condition is an important property for many uses of kaolin, as in paper filler and whiteware. Kaolin for use in refractories should have as low a content of alkalies and other fluxes as practicable. Most kaolins have low plasticity, low dry and fired strength, high drying and firing shrinkage, and high fired porosity.

In 1953, the domestic production of kaolin exceeded 1.8 million short tons valued at over 27 million dollars. Formerly large quantities of kaolin were imported, but currently only about 6 percent of the domestic consumption is imported, mostly from the United Kingdom.

OCCURRENCE

Kaolins are divided, according to origin, into two general classifications—residual kaolins and sedimentary kaolins. They are classified by mine producers in the eastern United States as soft, semihard, hard, or flint kaolins. This classification, though rule-of-thumb, is a rough guide to the selection of materials for specific applications. The more important residual kaolin deposits are in western North Carolina. Sedimentary kaolins have been worked in Georgia, South Carolina, and Pennsylvania. Other kaolin deposits occur in Alabama, California, Florida, Illinois, Nevada, Utah, Washington, and Virginia.

MINING AND PREPARATION

With few exceptions, kaolin is mined in the United States by open-pit methods. Most of the kaolin is beneficiated to remove impurities—principally mica, feldspar, quartz, and heavy minerals. Beneficiation usually consists of a series of disintegrating, washing, screening, classification, thickening, filtering, and drying operations. Magnetic separation often is used to remove iron and other magnetic contaminants. Drying may be followed by disintegration in hammer mills and air classification to produce dry pulverized material of specified particle size.

Hard kaolins can be beneficiated to the same grade as the soft kaolins by more costly milling. If soft kaolin is available and can be processed to meet the desired specifications, hard kaolin seldom is used. However, the hard kaolins of Georgia have become very important in refractories manufacture, for which their properties are desirable.

Control of product and improved beneficiation techniques in the preparation of kaolins have been greatly extended in the last 2 decades to satisfy increasingly rigid consumer specifications. As a result, the domestic kaolin industry has steadily improved its competitive position with imported kaolin.

USES

Kaolins have a wide variety of uses, depending upon their purity and their physical and chemical properties. These uses may be divided into four general groups as follows:

FILLERS

Kaolin is used as a filler in manufacturing paper, rubber, oilcloth and linoleum, paint, and numerous minor applications.

For paper manufacture, freedom from grit, degree of whiteness, particle-size distribution, rheological properties and retention[3] are of primary importance in determining the value of a kaolin. Specifications are much more exacting for kaolins used for coating high-grade papers than for those used for paper filler.

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[3] Retention is the proportion of the clay introduced into the paper-making process that is retained in the finished paper.

Clay has been used in rubber compounding since 1920 to supply the need for an inexpensive reinforcing material. It has proved very suitable for this purpose, particularly where resistance to abrasive wear is an important requirement. Desirable properties are natural dispersibility, uniformity, purity, fine particle size, and absorption. These properties have brought clay into active competition with both carbon black and zinc oxide. Synthetic rubber may or may not require the use of kaolin as a filler in the same proportion as it is used in products made from natural rubber. This depends to a marked degree on the types of synthetic rubber and the manufacturing processes employed in the future.

Methods of testing clays for oilcloth and linoleum are not standardized. In general, a clay should be white and free from grit, should slake readily to a smooth cream or slip, and should have a comparatively low oil absorption.

The only reliable and conclusive test of the value of kaolin for use in exterior paint in particular and other paints in general is the endurance of the paint in service for a period of years, but some accelerated tests have proved valuable. The color of the clay when mixed with oil is of first importance. Many kaolins that are a good white when dry turn dark when mixed with oil. Other important properties are freedom from grit and low oil absorption. Other things being equal, the finer the grain of a clay, the better it is as a paint extender and pigment.

Other products in which kaolin is used as a filler include plaster and plaster products, textiles, window shades, calcimine, crayons, toilet and tooth powders, soaps, soft polishing compounds, and matches. Industry-wide tests and specifications are usually lacking.

WHITEWARE

Whiteware refers to nonrefractory clay products that are white or nearly so after firing. Whiteware includes sanitary ware, electrical porcelain, and white pottery. Stoneware, earthenware, tableware, floor tile, and wall tile, if made of a white-firing body, also are classed as whiteware. The body composition of whiteware varies with the product, the process of manufacture, and the relative cost of available materials. The kaolin content averages about 25 percent of the total dry weight of the ingredients, ranging from about 10 to 50 percent. The ball-clay content ranges from 10 to 30 percent; feldspar from 5 to 40 percent; and silica (flint) up to 40 percent. Talc is used to replace part or all of the feldspar in some artware bodies, and nepheline syenite is of increasing importance as a substitute for feldspar in vitreous whiteware bodies.

REFRACTORIES

In recent years the consumption of kaolin in the manufacture of refractories in the United States has increased. In 1953 refractories manufacturers consumed 9 percent of the total domestic kaolin production. About 95 percent of the requirements for refractory kaolin was supplied by the sedimentary kaolins of Georgia. Refractories produced from kaolin of this type are comparable with the best grades of refractories manufactured for special uses in Pennsylvania and Missouri.

CEMENT

Kaolin and other white-burning clays are used in manufacturing white portland cement.

PRODUCTION, CONSUMPTION, AND FOREIGN TRADE

Kaolin sold or used by producers in the United States, 1925–53, by uses, is shown in table 2. Table 3 lists the production, imports, and apparent consumption in the United States, 1925–53.

Production, except for a few years, has grown steadily, reaching its peak in 1951. The apparent consumption has followed the same upward trend, even though the imports declined steadily from 1926–44 and then held steady to the present time.

Domestic output in 1953 totaled 1,883,974 short tons valued at $27,092,181, representing a 195-percent increase in tonnage over that produced in 1936. Georgia was the most important producing State, with 1,341,725 short tons. South Carolina ranked second in importance, with a production of 327,594 short tons. The balance of the domestic output was obtained from Alabama, California, Florida, North Carolina, Pennsylvania, Utah, and Virginia. Georgia production from 1936 to 1953 increased 220 percent, indicating that the kaolin industry in this State has developed at a greater rate than the overall domestic industry.

The principal reason for the steady gain in kaolin production lay in improved production methods, assisted by the protection given by the Tariff Act of 1922 and subsequent tariff acts.

Impetus was given to substitution of domestic kaolin for English kaolin by the shortage resulting from curtailed imports during World War I. As soon as consumers were convinced that domestic kaolin was uniform in properties and could be satisfactorily substituted for, or blended with, English kaolin in whiteware bodies and paper, the ratio of domestic to imported kaolin increased until, in 1953, imported kaolin represented only about 6 percent of the total consumption compared with 88 percent in 1910, 50 percent in 1925, and 18 percent in 1936. Imports from the United Kingdom have

TABLE 2.—*Kaolin sold or used by producers in the United States, 1925–53, by uses*

Year	Pottery and stoneware		High-grade tile		Paper filler		Paper coating		Rubber		Linoleum and oilcloth		Paints		Cement		Refractories, kiln furniture, etc.		Miscellaneous		Total sold or used (short tons)
	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	
1925	66,410	18.1	21,126	5.6	128,033	34.9	9,963	2.7	29,926	8.2	7,976	2.2	16,583	4.5	50,607	13.8	17,785	4.8	18,910	5.2	367,319
1926	72,729	16.8	23,862	5.5	156,897	36.3	3,543	.8	25,190	5.8	9,256	2.1	16,323	3.8	57,116	13.2	42,342	10.0	24,957	5.7	432,215
1927	65,154	14.4	22,790	5.0	192,307	42.3	3,510	.8	27,428	6.0	14,511	3.2	16,057	3.5	58,594	12.9	34,486	7.6	19,408	4.3	454,245
1928	67,008	13.5	21,096	4.3	203,235	41.0	34,000	6.9	36,654	7.4	13,036	2.6	16,455	3.3	45,328	9.1	40,745	8.2	18,585	3.7	496,142
1929	72,922	14.1	21,743	4.2	235,410	45.5	33,034	6.4	34,296	6.6	14,648	2.8	16,699	3.2	42,552	8.2	38,495	7.4	8,370	1.6	518,169
1930	45,686	8.9	18,732	3.7	244,005	47.8	34,711	6.8	29,387	5.8	9,542	1.9	11,657	2.3	62,542	12.3	35,265	6.9	18,827	3.6	510,354
1931	37,506	8.5	10,262	2.3	245,126	55.3	28,243	6.4	31,931	7.2	6,718	1.5	10,580	2.4	31,130	7.0	34,399	7.8	7,405	1.6	443,300
1932	23,103	7.0	4,931	1.4	206,665	59.9	23,624	6.8	33,474	9.7	4,728	1.3	7,291	2.1	12,394	3.5	25,609	7.4	3,175	.9	344,994
1933	33,076	8.0	7,636	1.9	228,718	55.6	27,217	6.6	49,615	12.1	4,994	1.2	10,274	2.5	17,653	4.3	23,641	5.8	8,409	2.0	411,233
1934	32,200	7.6	6,520	1.5	218,425	51.2	31,077	7.3	48,259	11.3	5,227	1.2	11,477	2.7	18,210	4.3	32,207	7.6	22,733	5.3	426,335
1935	40,617	7.8	12,612	2.4	273,687	52.3	45,284	8.7	58,846	11.2	6,329	1.2	10,124	1.9	27,838	5.3	34,626	6.6	13,693	2.6	523,656
1936	52,603	8.2	13,955	2.2	322,970	50.6	57,056	8.9	72,712	11.4	1,620	.3	13,662	2.1	26,304	4.1	52,377	8.2	25,680	4.0	638,939
1937	50,838	6.9	17,012	2.3	335,031	45.8	91,146	12.4	86,007	11.7	6,368	.9	7,227	1.0	32,788	4.5	78,692	10.8	27,173	3.7	732,282
1938	40,280	6.8	11,785	2.0	291,359	49.0	71,020	11.9	70,799	11.9	4,751	.8	7,859	1.3	34,238	5.7	42,202	7.1	20,761	3.5	595,054
1939	59,847	7.6	17,932	2.3	346,037	44.3	116,980	15.0	90,287	11.6	6,284	.8	9,660	1.2	41,913	5.4	59,320	7.6	32,544	4.2	780,804
1940	71,649	8.6	19,339	2.3	367,032	44.0	127,300	15.3	92,640	11.1	5,668	.7	10,016	1.2	40,001	4.8	73,096	8.8	26,709	3.2	833,450
1941	102,449	9.4	23,132	2.1	431,493	39.7	173,572	16.0	127,055	11.7	9,813	.9	14,453	1.3	42,761	3.9	126,797	11.7	36,323	3.3	1,087,848
1942	86,029	9.1	12,313	1.3	364,154	38.5	192,819	20.3	51,334	5.4	6,304	.7	16,692	1.8	12,315	1.3	162,234	17.1	42,394	4.5	946,588
1943	86,921	9.3	9,282	1.0	350,408	37.7	194,778	21.0	50,964	5.5	4,468	.5	16,732	1.8	12,674	1.3	149,457	16.1	53,753	5.8	929,437
1944	83,904	9.6	9,802	1.1	337,600	38.7	179,002	20.5	59,288	6.8	4,058	.5	17,893	2.0	15,335	1.7	105,334	12.1	60,540	7.0	873,056
1945	74,062	7.9	9,544	1.0	362,323	38.5	188,029	20.0	109,936	11.7	14,593	1.6	21,592	2.3	13,586	1.5	96,404	10.2	49,919	5.3	939,988
1946	111,537	8.4	14,321	1.1	448,756	33.9	248,527	18.8	162,393	12.3	13,323	1.0	22,228	1.7	47,559	3.6	153,228	11.6	100,431	7.6	1,322,303
1947	120,222	8.4	22,690	1.6	473,907	33.2	301,853	21.2	166,201	11.7	21,140	1.5	19,366	1.3	41,347	2.9	161,852	11.4	96,528	6.8	1,425,106
1948	145,797	9.3	29,292	1.9	472,360	30.1	399,768	25.5	176,965	11.3	28,285	1.8	15,629	1.0	56,157	3.5	158,343	10.1	86,252	5.5	1,568,848
1949	122,328	8.6	22,233	1.6	385,500	27.2	376,755	26.6	197,341	13.9	26,650	1.9	13,379	1.0	54,749	3.9	131,482	9.3	85,120	6.0	1,415,537
1950	143,245	8.2	28,573	1.6	440,313	25.2	463,195	26.5	263,306	15.0	34,986	2.0	16,283	.9	65,430	3.7	164,875	9.4	130,652	7.5	1,750,858
1951	141,733	7.6	33,729	1.8	533,490	28.6	431,111	23.1	231,331	12.4	36,163	1.9	22,719	1.2	77,486	4.2	206,744	11.1	151,793	8.1	1,866,299
1952	125,917	6.9	25,246	1.4	459,184	25.1	492,491	26.9	240,982	13.2	39,781	2.2	31,784	1.7	43,113	2.3	206,344	11.3	164,260	9.0	1,829,102
1953	141,901	7.5	26,896	1.4	437,151	23.2	558,194	29.6	241,052	12.8	38,510	2.1	30,182	1.6	46,128	2.5	186,030	9.9	177,930	9.4	1,883,974

TABLE 3.—*Kaolin production,*[1] *imports, and apparent consumption,*[2] *United States, 1925–53*

Year	Production [1]		Imports		Apparent domestic consumption [2]		Proportion of imports to domestic consumption (percent)	
	Short tons	Value	Short tons	Value	Short tons	Value	Short tons	Value
1925	367, 319	$3, 220, 719	372, 552	$3, 195, 694	739, 871	$6, 416, 413	50. 4	49. 8
1926	432, 215	3, 771, 568	396, 219	3, 484, 054	828, 434	7, 255, 622	47. 8	48. 0
1927	454, 245	3, 809, 834	339, 014	2, 937, 113	793, 259	6, 746, 947	42. 7	43. 5
1928	496, 142	4, 088, 003	307, 304	2, 962, 269	803, 446	7, 050, 272	38. 2	42. 0
1929	518, 169	4, 281, 301	279, 981	2, 700, 296	798, 150	6, 981, 597	35. 1	38. 7
1930	510, 354	3, 893, 814	236, 251	2, 197, 540	746, 605	6, 091, 354	31. 9	36. 1
1931	443, 300	2, 946, 953	151, 426	1, 056, 393	594, 726	4, 003, 346	25. 5	26. 4
1932	344, 994	2, 011, 208	99, 807	461, 191	444, 801	2, 472, 399	22. 4	18. 6
1933	411, 233	2, 366, 339	116, 180	632, 437	527, 413	2, 998, 776	22. 0	21. 1
1934	426, 335	2, 699, 016	100, 775	752, 993	527, 110	3, 452, 009	19. 1	21. 8
1935	523, 656	3, 765, 268	125, 963	959, 821	649, 619	4, 725, 089	19. 4	20. 3
1936	638, 939	4, 537, 738	139, 797	1, 110, 780	778, 736	5, 648, 518	17. 9	19. 7
1937	732, 282	5, 349, 636	146, 523	1, 211, 266	878, 805	6, 560, 902	16. 7	18. 5
1938	595, 054	4, 740, 880	84, 180	753, 858	679, 234	5, 494, 738	12. 4	13. 7
1939	780, 804	6, 200, 606	114, 696	1, 015, 813	895, 500	7, 216, 419	12. 8	14. 1
1940	833, 450	6, 994, 106	105, 567	870, 421	939, 017	7, 864, 527	11. 2	11. 1
1941	1, 087, 848	9, 205, 892	85, 141	901, 973	1, 172, 989	10, 107, 865	7. 3	8. 9
1942	946, 588	8, 037, 233	56, 498	671, 081	1, 003, 086	8, 708, 314	5. 6	7. 7
1943	929, 437	8, 067, 322	49, 612	584, 532	979, 049	8, 651, 854	5. 1	6. 8
1944	873, 056	7, 858, 740	45, 950	487, 237	919, 006	8, 345, 977	5. 0	5. 8
1945	939, 988	9, 072, 927	57, 590	871, 541	997, 578	9, 944, 468	5. 8	8. 8
1946	1, 322, 303	13, 553, 767	79, 644	1, 440, 746	1, 401, 947	14, 994, 513	5. 7	9. 6
1947	1, 425, 106	17, 107, 963	82, 628	1, 330, 001	1, 507, 734	18, 437, 964	5. 5	7. 2
1948	1, 568, 848	19, 756, 738	99, 930	1, 650, 102	1, 668, 778	21, 406, 840	6. 0	7. 7
1949	1, 415, 537	19, 007, 547	77, 226	1, 156, 803	1, 492, 763	20, 164, 350	5. 2	5. 7
1950	1, 750, 858	23, 943, 782	122, 974	1, 619, 384	1, 873, 832	25, 563, 166	6. 6	6. 3
1951	1, 866, 299	25, 324, 554	110, 475	1, 581, 378	1, 976, 774	26, 905, 932	5. 6	5. 9
1952	1, 829, 102	25, 205, 836	103, 937	1, 526, 920	1, 933, 039	26, 732, 756	5. 4	5. 7
1953	1, 883, 974	27, 092, 181	118, 775	1, 854, 248	2, 002, 749	28, 946, 429	5. 9	6. 4

[1] Sold or used by producers. [2] Production plus imports.

composed approximately 95 percent of all kaolin imported into this country since 1914. Kaolin exports from the United States in 1953 were 43,590 short tons valued at $795,043.

The paper industry utilizes imported kaolin as a filler and to a minor extent for coating. The whiteware division of the ceramic industry utilizes nearly all imports not consumed by the paper industry. Most of the imported kaolin is used in the New England States, largely because delivered prices are comparable to the cost of domestic kaolin in that area. In other United States localities, English kaolin is more expensive than domestic and is not extensively used.

By a reciprocal trade agreement with Great Britain, a tariff of $1.25 per long ton was made effective January 1, 1948, and was still in effect in 1954. The rate is only half that allowed by the Traiff Act of 1930.

The domestic apparent consumption of kaolin increased 157 percent in quantity from 1936 to 1953 compared with an increase of 119 percent in the Federal Reserve Board's Index of Industrial Production.

PRICES

Quotations of Georgia kaolin, as reported in E&MJ Metal and Mineral Markets, for filler and ceramic grades were given in December 1950 as $8.50 to $9.50 per ton, depending upon grade, for crushed material, and $13 to $17 for pulverized, in paper bags. North Carolina china clays, ceramic grades, in bulk, carlots, were quoted at $20.25 to $22.25 per ton. Florida kaolins were quoted by the same source at $18.75 per ton for purified and crushed; $24.75 for washed and air-floated clays; and $38.50 for air-floated enamel grade. Crude Pennsylvania kaolin was quoted at $5 to $7.50

per ton and "purified" kaolin at $21 to $24. These prices were substantially the same as those quoted in 1949. No price changes were reported up to November 1, 1954.

Prices of imported china clay in January 1954 were quoted by the Oil, Paint and Drug Reporter as follows: White lump, carlots, ex dock (Philadelphia, Pa., and Portland, Maine), $20 to $40 per long ton; powdered, ex dock, in bags $50 per net ton; and powdered, l. c. l., ex warehouse, $60 to $65. The average value of domestic kaolin sold or used as reported to the Bureau of Mines in 1953 rose to $14.38 compared with $13.78 for 1952, $13.68 for 1951, and $13.43 in 1950.

BALL CLAY

The name "ball clay" originated in England, where because of its soft pliable nature, it was mined from open pits by cutting into ball-like chunks weighing 30 to 35 pounds.

Ball clay, like kaolin, is principally kaolinite and other aluminum silicates. It varies in chemical composition and physical and ceramic properties. Ball clay of good quality is plastic, refractory, and of sedimentary origin. Compared with kaolin, ball clay contains more organic matter and iron compounds and less alumina and has a higher ignition loss. It is seldom as refractory or as white as kaolin in the raw or fired state. Ball clay is fine textured and characterized by high drying and firing shrinkage, high dry and fired strength, and low absorption when fired to vitrification.

OCCURRENCE

Most commercial ball-clay deposits of the world occur in close association with coal or lignite deposits, although the coal seams themselves may not be economically workable.

The largest deposits of commercial value in the United States are in western Kentucky and western Tennessee. In these States the ball-clay deposits occur in the Tertiary formation as large lenses associated with other refractory clays. Deposits in Cretaceous beds in New Jersey also are important.

The following States reported production in 1953: Kentucky, Tennessee, Maryland, Mississippi, California, and New Jersey. Approximately 88 percent of all domestic production in 1953 came from Kentucky and Tennessee.

The domestic ball-clay reserves are adequate to meet any anticipated demand.

PREPARATION

Ball-clay producers are delivering and offering for sale today the best prepared and standardized product in the history of the industry. It is common practice to dry, crush, and screen ball clay and also to pulverize and air-float it, to remove objectionable impurities and make a more uniform product.

USES

All but 2 to 3 percent of the ball-clay consumption of the United States is used in ceramics. Its functions are to act as a plasticizer to control working properties and as a bonding agent. With few exceptions, ball clay is an important ingredient in whiteware. It is used also in many other ceramic body formulas. The quantity of ball clay used in specific ceramic compositions ranges up to about 40 percent.

There has been a growing trend among manufacturers of sanitary ware and semivitreous dinnerware to substitute ball clay for kaolin. The reasons for making such a substitution were to obtain a more dense body at a lower firing temperature, and to impart increased workability and dry strength to take advantage of new mechanical appliances and methods developed in the manufacture of whiteware bodies in the pottery industry.

The nonceramic uses of ball clays are principally as fillers.

PRODUCTION, CONSUMPTION, AND FOREIGN TRADE

The domestic ball clay sold or used by producers, 1933–53, by uses, is given in table 4. Before 1933 the peak production was 120,000 short tons (1928). Production declined to 47,573 tons in 1932. In 1937 the industry had recovered to the position it held in 1928, and after 1938 a definite upswing in production began that culminated in the alltime high of 344,981 short tons in 1951.

The apparent consumption of ball clay increased 148 percent in quantity from 1936–53. This is nearly the same as the rate of increase in apparent consumption of kaolin, which increased 151 percent during the same period.

Increased requirements for ball clay have been met by expanded domestic production rather than by increased imports. Standardization of quality and improved methods of preparation are the chief factors that have enabled domestic ball-clay producers to increase the ratio of domestic to imported ball-clay sales in the post-World War II market.

From 1949–53, except for a negligible quantity from Canada, all imports of ball clay originated in England. Before World War II occasional shipments came from Belgium, Germany, and the Netherlands.

Before 1936 the Bureau of Foreign Commerce and Navigation, now the Bureau of Foreign Commerce, classified imports of ball clay under "Clays and earths n. s. p. f." For this reason,

TABLE 4.—*Domestic ball clay sold or used by producers, 1933–53, by uses*

Year	Whiteware		Pottery and stoneware [1] (except whiteware)		High-grade tile		Architectural terra cotta		Filler [2]		Refractories [3]		Miscellaneous [4]		Total sold or used (short tons)
	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	
1933	43, 835	68. 0	5, 792	8. 9	5, 724	8. 8	2, 300	3. 6	406	0. 6	5, 032	7. 8	1, 462	2. 3	64, 551
1934	48, 780	77. 6	680	1. 0	6, 599	10. 5	1, 768	2. 9	2, 463	3. 9	1, 792	2. 8	795	1. 3	62, 877
1935	65, 009	67. 5	2, 068	2. 1	16, 233	16. 9	1, 117	1. 2	6, 155	6. 4	4, 416	4. 6	1, 262	1. 3	96, 260
1936	68, 801	67. 9	2, 224	2. 2	18, 109	17. 9	4, 735	4. 7	5, 301	5. 2	1, 598	1. 6	556	. 5	101, 324
1937	89, 399	73. 6	523	. 4	22, 675	18. 7	5, 027	4. 1	2, 774	2. 3	803	. 7	269	. 2	121, 470
1938	75, 647	79. 7	195	. 2	16, 005	16. 8	935	1. 0	936	1. 0	490	. 5	760	. 8	94, 968
1939	95, 522	74. 3	3, 227	2. 5	15, 261	11. 9	3, 985	3. 0	4, 428	3. 5	1, 265	1. 0	4, 913	3. 8	128, 601
1940	110, 447	78. 5	2, 199	1. 6	20, 211	14. 4	2, 000	1. 4	4, 500	3. 2	302	. 2	1, 048	. 7	140, 707
1941	148, 600	74. 9	1, 653	. 8	32, 048	16. 1	217	. 1	10, 792	5. 5	1, 797	. 9	3, 338	1. 7	198, 445
1942	136, 181	84. 0	2, 151	1. 3	9, 793	6. 0	75	(5)	7, 438	4. 6	3, 871	2. 4	2, 784	1. 7	162, 293
1943	132, 237	89. 5	1, 100	. 7	6, 720	4. 6	none	--------	2, 384	1. 6	1, 668	1. 1	3, 676	2. 5	147, 785
1944	138, 923	89. 2	560	. 4	8, 980	5. 8	none	--------	none	--------	2, 101	1. 3	5, 103	3. 3	155, 667
1945	156, 739	89. 8	1, 680	1. 0	8, 017	5. 0	1, 962	. 6	none	--------	3, 318	2. 0	2, 808	1. 6	174, 524
1946	205, 015	84. 3	10, 631	4. 4	17, 104	7. 1	1, 729	. 7	258	(5)	2, 635	1. 1	5, 773	2. 4	243, 145
1947	215, 113	79. 9	9, 233	3. 5	25, 672	9. 5	1, 400	. 5	--------	--------	11, 832	4. 4	5, 800	2. 2	269, 050
1948	242, 333	81. 0	8, 896	3. 0	23, 842	8. 0	none	--------	3, 965	1. 3	12, 636	4. 3	7, 307	2. 4	298, 979
1949	194, 177	78. 1	5, 965	2. 4	27, 349	11. 0	1, 270	. 5	300	(5)	12, 239	5. 0	7, 583	3. 0	248, 883
1950	248, 899	76. 7	6, 825	2. 2	38, 228	11. 7	1, 278	. 4	none	--------	18, 310	5. 6	10, 874	3. 4	324, 414
1951	260, 832	75. 6	6, 750	2. 0	42, 333	12. 2	1, 295	. 4	none	--------	22, 091	6. 4	11, 680	3. 4	344, 981
1952	212, 631	69. 7	11, 064	3. 6	36, 255	11. 9	1, 195	. 4	none	--------	30, 578	10. 0	13, 360	4. 4	305, 083
1953	209, 825	69. 8	10, 391	3. 5	40, 438	13. 4	none	--------	none	--------	24, 084	8. 0	16, 024	5. 3	300, 762

[1] Includes chemical stoneware, stoneware, art pottery, etc.
[2] Includes filler for paper, rubber, linoleum, oilcloth, and paints.
[3] Includes kiln furniture, saggers, etc.
[4] Includes artificial abrasives, enameling, asbestos products and other uses.
[5] Less than 0.05 percent.

SOURCE: Production reports submitted to Bureau of Mines.

no specific figures are available before 1936. In that year 24 percent of the apparent United States consumption of ball clay was imported. Imports represented approximately 7 percent of the total apparent domestic consumption in 1953.

Separate statistics for ball clay exported from the United States are not available. Whiteware and high-grade tile manfacture consume the greater part of all ball-clay imports. No specific breakdown of imports by uses is available.

The duty on "unwrought and unmanufactured ball clay" is $1 per long ton, and the duty on "wrought or manufactured ball clay" is $2 per long ton. These rates have been in effect since 1922.

PRICES

The latest price quotations appeared in E&MJ Metal and Mineral Markets in 1950. They showed no change from those given for 1949 and were as follows: Tennessee—crude ball clay, $10 per short ton, and air-floated and pulverized, $19.50 per ton; Maryland—shredded, in bulk, $7 to $9, and air-floated, in bags, $14 to $17.50 per ton. No quotations for Kentucky ball clay in 1950 were given in E&MJ Metal and Mineral Markets. In 1953 the average value per ton for ball clay, as reported by producers to the Bureau of Mines, was $11.27 compared with $12.97 in 1952, $10.80 in 1951, and $12.27 in 1950.

FIRE CLAY

Fire clays are those clays capable of withstanding high temperatures; they include most clays having a softening point above 2,900° F. Fire clay is often classified according to workability, as plastic, semiplastic, semiflint, and flint, or by mineralogical composition, as siliceous, kaolinitic, or aluminous. A plastic fire clay, when mixed with water, is readily workable and can be shaped by hand. Flint clays are hard, nonplastic clays with a waxy luster and conchoidal fracture that develop weak plasticity when finely ground in water. Semiflint and semiplastic clays are intermediate in properties. Minerals of the kaolinite group predominate in fire clays.

Buff- and cream-firing plastic fire clays are used widely as stoneware clays. Refractoriness is not a requisite for this purpose, but the relative abundance of fire clays of suitable working properties has encouraged their use for stoneware manufacture.

Fire clays having a P. C. E. of 33 or higher are in demand for the manufacture of superduty refractories and also to enrich lower grade fire clays. In addition to having a high P. C. E., they must withstand rigid thermal-spalling and high-temperature-load tests. Superduty fireclay refractories are used in iron blast furnaces, boiler furnaces, fireboxes, certain parts of glass furnaces ("tanks"), rotary cement kilns, and many other applications in which severe service requirements must be met.

High-duty refractories are made from a fire clay or a blend of fire clays having a P. C. E. of 31–32 or higher. They are more resistant to spalling than are the fire clays of lower refractoriness and may be burned hard enough to result in marked improvement in resistance to disintegration by carbon monoxide gas.

In certain parts of industrial furnaces, the refractory structure is exposed to moderate or ow temperatures, and intermediate-duty refractories of fire-clay blends having a P. C. E. of 30–31 and low-duty refractories of clay blends having a P. C. E. as low as 25 minus give excellent service. Clays used in making ladle brick and certain other pouring-pit refractories may have a P. C. E. as low as 15.

OCCURRENCE

Flint fire clays occur in a number of districts throughout the United States. The most important deposits are in Pennsylvania, Ohio, Kentucky, and Missouri, predominantly in the Carboniferous, associated with coal. Most of the mining is underground.

High-duty and low-duty plastic fire clays are more widespread than flint clays and fortunately are accessible to most industrial areas. Pennsylvania, Ohio, West Virginia, Illinois, Missouri, Texas, Colorado, Mississippi, and California contain important deposits.

Extensive deposits of clays suitable for superduty firebrick manufacturing occur in the Ozark and North Central regions of Missouri and to smaller degrees in Pennsylvania and Kentucky. In many refractory-producing areas, superduty clays are absent or in short supply.

USES

All industries that use high temperatures rely heavily on fire-clay refractories for furnace structures. The largest consuming industry is iron and steel, but other important industry users are copper, aluminum, cement, glass, public utilities, ceramics, petroleum, and the chemical industries. Large tonnages of fire-clay refractories also are used in marine boilers. In 1953 refractories, including kiln furniture, consumed approximately 52 percent of the total fire-clay output (table 1).

Building brick, paving brick, sewer pipe, wall tile, architectural terra cotta, and kindred products consume large quantities of fire clay. Buff- and cream-firing plastic fire clays are used extensively in stoneware manufacture.

PRODUCTION, CONSUMPTION, AND FOREIGN TRADE

The quantity of fire clay sold or used in 1953 was the third largest in the history of the industry, being exceeded only by the 1952 and 1951 (record year) figures (table 5). In 1953 Ohio was the largest producer, followed by Pennsylvania, Missouri, West Virginia, Indiana, California, Colorado, Texas, and Kentucky (table 6). These States furnished 85 percent of the total output. The remainder was supplied by 19 States.

TABLE 5.—*Fire clay, including stoneware clay, sold or used by producers in the United States, 1925–53*

Year	Short tons	Value
1925	2, 644, 372	$7, 474, 510
1926	2, 903, 348	8, 304, 611
1927	2, 798, 116	7, 923, 373
1928	2, 880, 784	7, 687, 884
1929	3, 266, 261	8, 289, 487
1930	2, 622, 994	6, 217, 176
1931	1, 530, 627	3, 872, 953
1932	775, 729	2, 139, 581
1933	1, 147, 085	3, 169, 677
1934	1, 345, 587	3, 833, 312
1935	1, 818, 781	4, 602, 845
1936	2, 471, 575	6, 135, 564
1937	2, 785, 344	7, 180, 938
1938	1, 458, 941	4, 060, 160
1939	2, 222, 295	5, 801, 993
1940	2, 765, 247	7, 046, 746
1941	4, 167, 567	10, 187, 733
1942	4, 839, 332	11, 381, 506
1943	4, 701, 144	11, 169, 922
1944	6, 344, 383	14, 167, 118
1945	6, 090, 411	15, 587, 034
1946	7, 907, 974	20, 827, 220
1947	9, 038, 680	26, 208, 602
1948	9, 849, 914	29, 424, 034
1949	8, 571, 844	25, 358, 503
1950	9, 535, 867	29, 019, 372
1951	11, 852, 517	48, 740, 596
1952	11, 285, 173	48, 383, 470
1953	10, 267, 113	37, 802, 989

The principal consumers of fire clay in 1953 were refractories, using about 5.3 million tons, and heavy clay products, using about 4.5 million tons. The chief use of fire clay for refractories is in the manufacture of firebrick and shapes, which consumed about 3.8 million tons.

Separate figures for fire-clay imports are not shown in foreign trade statistics. Exports of fire clay in 1953 totaled 90,897 short tons valued at $919,928. Canada received 82 percent of the total exports and Mexico 13 percent. The remaining 5 percent, comprising small-tonnage shipments, was exported to many destinations in Central and South America, Europe, Asia, and Africa.

PRICES

The average value of fire clay in the United States, as reported by producers, was $3.68 per short ton in 1953. Values range from about $1 per ton for the more abundant clays amenable

TABLE 6.—*Fire clay, including stoneware clay, sold or used by producers in the United States, 1952–53, by States*[1]

State	Sold by producer		Used by producer		Total	
	Short tons	Value	Short tons	Value	Short tons	Value
1952						
Alabama	145, 567	$292, 788	74, 442	$464, 663	220, 009	$757, 451
Arkansas	(2)	(2)	(2)	(2)	386, 111	1, 337, 542
California	196, 954	631, 351	365, 181	1, 086, 613	562, 135	1, 717, 964
Colorado	159, 972	377, 635	140, 902	410, 979	300, 874	788, 614
Illinois	193, 076	962, 605	257, 368	582, 063	450, 444	1, 544, 668
Indiana	280, 629	440, 010	116, 707	292, 015	397, 336	732, 025
Kentucky	96, 308	418, 111	429, 930	3, 031, 935	526, 238	3, 450, 046
Maryland	9, 937	45, 319	165, 465	503, 539	175, 402	548, 858
Missouri [3]	585, 691	1, 776, 186	1, 265, 111	8, 683, 401	1, 850, 802	10, 459, 587
New Jersey	80, 225	721, 960	199, 538	877, 789	279, 763	1, 599, 749
Ohio	792, 894	2, 814, 155	2, 074, 563	8, 124, 919	2, 867, 457	10, 939, 074
Pennsylvania	245, 062	972, 854	1, 718, 173	9, 594, 840	1, 963, 235	10, 567, 694
South Carolina	(2)	(2)	(2)	(2)	7, 547	18, 250
Tennessee	(2)	(2)	(2)	(2)	21, 290	203, 845
Texas	15, 716	168, 242	342, 750	895, 763	358, 466	1, 064, 005
Utah	5, 215	28, 682	29, 254	78, 689	34, 469	107, 371
Washington	8, 300	12, 338	71, 671	143, 317	79, 971	155, 655
West Virginia	(2)	(2)	(2)	(2)	621, 996	2, 072, 688
Other States [4]	147, 452	599, 078	1, 071, 120	3, 351, 631	181, 628	318, 384
Total	2, 962, 998	10, 261, 314	8, 322, 175	38, 122, 156	11, 285, 173	48, 383, 470
1953						
Alabama	149, 510	$288, 491	103, 416	$438, 948	252, 926	$727, 439
Arizona	-----------	-----------	540	1, 485	540	1, 485
Arkansas	(2)	(2)	(2)	(2)	331, 252	1, 524, 865
California	161, 975	431, 071	290, 765	1, 175, 029	452, 740	1, 606, 100
Colorado	213, 364	511, 410	151, 828	455, 286	365, 192	966, 696
Illinois	215, 132	496, 717	103, 375	221, 526	318, 507	718, 243
Indiana	287, 688	527, 229	294, 951	636, 458	582, 639	1, 163, 687
Kentucky	50, 369	228, 739	297, 990	1, 581, 249	348, 359	1, 809, 988
Maryland	8, 149	26, 393	145, 655	389, 633	153, 804	416, 026
Mississippi	(2)	(2)	(2)	(2)	43, 850	74, 209
Missouri [3]	457, 067	1, 397, 851	1, 039, 070	8, 689, 894	1, 496, 137	10, 087, 745
Montana	2, 470	12, 160	-----------	-----------	2, 470	12, 160
New Jersey	48, 341	446, 452	89, 663	377, 254	138, 004	823, 706
New Mexico	(2)	(2)	(2)	(2)	5, 367	17, 605
Ohio	802, 219	2, 203, 621	2, 005, 824	3, 976, 014	2, 808, 043	6, 179, 635
Pennsylvania	247, 837	941, 928	1, 422, 751	6, 879, 398	1, 670, 588	7, 821, 326
South Carolina	(2)	(2)	(2)	(2)	15, 208	44, 075
Texas	594	2, 548	355, 617	913, 027	356, 211	915, 575
Utah	11, 262	26, 444	24, 308	69, 020	35, 570	95, 464
Washington	6, 209	10, 471	67, 479	113, 691	73, 688	124, 162
West Virginia	(2)	(2)	(2)	(2)	677, 005	2, 213, 376
Other States [4]	73, 804	386, 262	1, 137, 891	3, 947, 290	139, 013	459, 422
Total	2, 735, 990	7, 937, 787	7, 531, 123	29, 865, 202	10, 267, 113	37, 802, 989

[1] Includes stoneware clay as follows: 1952—80,651 tons, $326,408; 1953—67,628 tons, $175,574.

[2] Included with "Other States."

[3] Includes diaspore and burley clay as follows: 1952—diaspore, 44,757 tons, $705,269; burley, 71,433 tons, $664,358; 1953—diaspore, 50,144 tons, $962,384; burley, 53,971 tons, $563,043.

[4] Includes States indicated by footnote 2 above and Georgia, Idaho, Iowa, Kansas, Michigan (1952 only), Minnesota, Mississippi (1952 only), Montana (1952 only), Nebraska (1952 only), Nevada, New Mexico (1952 only), New York (1953 only), Oregon, Tennessee (1953 only).

to large-scale open-pit mining to $12 to $15 per ton for scarce varieties requiring selective methods of underground mining. The price quotations for firebrick manufactured from fire clays, as reported in December 1954 in the E&MJ Metal and Mineral Markets are as follows: Missouri, Kentucky, and Pennsylvania, first quality, $109.00 per thousand, and high heat quality, $102.00 per thousand; Ohio fire brick, intermediate grade, $102.00 per thousand and second quality $93 per thousand. (About 4 tons of clay is required per thousand 9-inch standard fire brick.)

BENTONITE (*8*)

Bentonites are very fine grained clays that consist principally of the minerals of the montmorillonite $[(Al,Mg)_8(Si_4O_{10})_3(OH)_{10}\cdot 12H_2O]$ family and beidellite $[Al_8(Si_4O_{10})_3(OH)_{12}\cdot 12H_2O]$ associated with fragments of feldspar, biotite, gypsum, quartz, pyroxenes, zircon, and other minerals. Bentonites are derived from volcanic ash. Two classes of bentonite are recognized on the basis of physical properties: (1) Swelling—those that can absorb large quantities of water, swell greatly in the process, and produce permanent suspensions in dilute dispersions; and (2) nonswelling—the metabentonites or subbentonites, which absorb only slighlty more water than ordinary plastic clays or fuller's earths, do not swell noticeably, and settle rapidly in dilute suspensions (*10*).

The adsorptive properties of bentonite range from inactive and nearly inactivable montmorillonite clays to inactive but highly activable clays.

Bentonite and fuller's earth grade into each other, which leads to some inconsistency in production statistics. In borderline cases the significant distinction between bentonite and fuller's earth is that fuller's earth has natural highly adsorptive properties, whereas significant ability to adsorb coloring matter from oils is developed in bentonites only by chemical treatment.

Because of the great variation in the properties of bentonites from different sources, tests and specifications must be made for each specific use.

OCCURRENCE

Bentonite occurs in nearly every State west of the Mississippi River and in a belt from Kentucky to the Gulf of Mexico, but commercial production is limited to a few areas.

Over half of the domestic bentonite is produced in the Wyoming-South Dakota area, mainly from a large district in the Black Hills area of northeastern Wyoming and adjacent portions of western South Dakota and a smaller district near Greybull, Wyo. This production is mainly the swelling type of bentonite.

Mississippi is the second largest producer, supplying nonswelling bentonite.

In recent years the balance of the production has come from Arizona, California, Colorado, Idaho, Montana, Nevada, Texas, and Utah.

USES

In 1953 oil refining, drilling mud, and foundry-molding-sand bonding accounted for nearly 94 percent of the bentonite consumed (table 1).

The sodium bentonites are preferred for drilling mud, and the calcium bentonites are used in oil refining. Both types are used as a bond in molding sands, with the sodium type superior in dry strength, though distinctly inferior in green strength, to the calcium type.

Other uses include: Bonding agent for insulating and refractory bodies and cements, ore pellets and briquets, and animal-feed pellets; sealing agent in irrigation ditches, dams, and reservoirs; in polishing agents, horticultural sprays, and concrete; and for clarifying water, sewage, and industrial wastes. There are many other uses where its jelling and suspending properties are employed, as in asphalt emulsions, grease compounds, ceramic bodies and glazes, paper, pharmaceutical and cosmetic preparations, and detergents; as a base for lake pigments; and as an ingredient of porcelains, enamels, and glazes.

PRODUCTION, CONSUMPTION, AND FOREIGN TRADE

Bentonite production in 1953 totaled 1,269,971 short tons valued at $16,180,242. This was the greatest tonnage in the history of the industry. The Wyoming-South Dakota district produced 876,059 short tons or 69 percent of the total bentonite in 1953 (table 7).

TABLE 7.—*Bentonite sold or used by producers in the United States, 1951–53, by States*

State	1951		1952		1953	
	Short tons	Value	Short tons	Value	Short tons	Value
Arizona	(1)	(1)	(1)	(1)	134, 850	$651, 752
Mississippi	(1)	(1)	(1)	(1)	189, 211	2, 028, 040
Montana	----------	----------	2, 000	$24, 000	----------	----------
South Dakota	246, 585	$2, 926, 756	205, 934	2, 553, 783	205, 303	2, 700, 394
Texas	38, 425	212, 670	31, 386	584, 938	47, 887	670, 300
Utah	(1)	(1)	(1)	(1)	1, 738	20, 396
Wyoming	465, 254	5, 981, 655	692, 853	9, 168, 708	670, 756	9, 861, 321
Other States [2]	468, 604	3, 885, 564	[3] 385, 806	2, 459, 519	20, 226	248, 039
Total	1, 218, 868	13, 006, 645	1, 317, 979	14, 790, 948	1, 269, 971	16, 180, 242

[1] Included with "Other States."

[2] Includes Alabama (1953 only), Arizona (1951–52 only), California, Colorado (1951 and 1953 only), Idaho (1951 only), Louisiana (1952–53 only), Mississippi (1951–52 only), Nevada (1952 only), Oklahoma (1952–53 only), and Utah (1951–52 only).

[3] Revised figure.

The bentonite sold or used by domestic producers from 1935 to 1953 is shown in table 8. The distribution of bentonite sold or used by domestic producers for specific uses in 1953 is shown in table 1.

Data on imports and exports of bentonite in 1953 are not available. Less than 100 tons of bentonite was imported in 1952, entirely from Canada. Some domestic producers export part of their output to destinations throughout the world.

PRICES

Since 1951 bentonite prices have not been quoted in E&MJ Metal and Mineral Markets. The following quotations were given for 1951 on Wyoming bentonite: Dried, crushed, in bulk, $9 per short ton; and 200-mesh, pulverized, in 100-pound bags, $12.50. Oil-well grade was quoted at $14 per short ton. The average value per short ton, as reported by the producers to the Bureau of Mines in 1953, was $12.74 compared with $11.22 in 1952, $10.67 in 1951, and $8.79 in 1950.

TABLE 8.—*Bentonite sold or used by producers in the United States, 1930-53* [1]

Year	Short tons	Value
1930	82, 593	$827, 912
1931	52, 293	429, 842
1932	57, 743	489, 803
1933	84, 993	719, 345
1934	146, 187	977, 208
1935	157, 445	1, 047, 600
1936	177, 807	1, 367, 420
1937	194, 768	1, 500, 758
1938	192, 183	1, 373, 182
1939	219, 720	1, 702, 393
1940	251, 032	1, 919, 461
1941	354, 028	2, 451, 900
1942	374, 967	2, 548, 509
1943	480, 202	2, 997, 754
1944	546, 768	3, 605, 988
1945	573, 998	3, 770, 625
1946	601, 428	4, 361, 414
1947	763, 889	5, 949, 586
1948	921, 560	7, 136, 308
1949	867, 243	6, 938, 752
1950	973, 833	8, 560, 669
1951	1, 218, 868	13, 006, 645
1952	1, 317, 979	14, 790, 948
1953	1, 269, 971	16, 180, 242

[1] All bentonite figures before 1930 were included in Miscellaneous clays.

FULLER'S EARTH

In fuller's earth, as in bentonite, montmorillonite usually is the dominant mineral, although beidellite may predominate in some cases and zeolites are important in others. Fuller's earths, however, adsorb coloring materials from animal, vegetable, and mineral oils without activation by acids or alkalies (*2, 7, 10, 14*).

Tests and specifications of fuller's earth for oil decolorizing and other major uses are well established (*9, 16*).

OCCURRENCE

Geographically, fuller's earth is widely distributed, although the world's needs are satisfied from comparatively few producing localities. England was formerly the largest producer, but today the United States is by far the greatest producer as well as consumer.

Fuller's earth was first discovered in the United States in 1891 near Benton, Ark. It was found in 1893 at Quincy, Fla., and until 1924 Florida was the leading producing State. From 1924 to 1935 Georgia led. Since 1935 production statistics for Georgia and Florida have been combined to coneal the output of individual producers. Texas became a major producer in 1941. There are also deposits in Illinois, Colorado, Utah, Nevada, California, Mississippi, and Tennessee.

USES

Absorbent uses took 30 percent of the national consumption in 1953 compared with 24 percent in 1952, 20 percent in 1951, and 21 percent in 1950. Mineral-oil refining was the second largest consumer in 1953 (21 percent) compared with 32 percent in 1952, 36 percent in 1951, and 40 percent in 1950.

In 1953 insecticides consumed 17 percent of the national output compared with 18 percent in 1952, 1951, and 1950; rotary-drilling mud, 12 percent compared with 15 percent in 1952 and 1951 and 16 percent in 1950; and vegetable-oil refining, 4 percent compared with 4 percent in 1952 and 1951 and 5 percent in 1950. The remainder was used in other filtering and clarifying binders and unspecified uses.

SUBSTITUTES

Activated bentonite, activated bauxite, synthetic magnesium silicate, and diatomite compete with fuller's earth in mineral-oil bleaching and filtration; bentonite, clays, and other mineral fillers compete for insecticide and other uses.

PRICES

Prices quoted by industry in 1953 were as follows: 14- to 30-mesh, $20 per short ton; 30- to 60-mesh, $21.75; 100-mesh up, $10.50; and 200-mesh up, $11.50. The average value of fuller's earth sold or used, as reported to the Bureau of Mines in 1953, was $17.47, compared with $16.26 in 1952, $16.81 in 1951, and $16.42 in 1950.

TABLE 9.—*Fuller's earth sold or used by producers in the United States, 1925–53*

Year	Short tons	Value
1925	206, 574	$2, 923, 965
1926	234, 152	3, 356, 482
1927	264, 478	3, 767, 038
1928	287, 012	3, 895, 991
1929	315, 983	4, 309, 723
1930	335, 644	4, 326, 705
1931	288, 400	3, 055, 570
1932	252, 902	2, 440, 736
1933	224, 152	2, 080, 640
1934	220, 264	2, 085, 081
1935	227, 745	2, 230, 229
1936	230, 814	2, 264, 978
1937	226, 165	2, 296, 094
1938	170, 852	1, 707, 869
1939	167, 070	1, 691, 855
1940	146, 568	1, 471, 083
1941	207, 446	2, 111, 674
1942	204, 244	2, 139, 670
1943	247, 258	2, 664, 027
1944	294, 737	3, 297, 064
1945	296, 368	3, 463, 913
1946	298, 752	3, 702, 993
1947	329, 068	4, 660, 614
1948	342, 081	5, 273, 851
1949	320, 906	5, 199, 642
1950	396, 025	6, 504, 733
1951	483, 623	8, 131, 761
1952	422, 853	6, 875, 483
1953	435, 837	7, 614, 759

MISCELLANEOUS CLAYS

Clay used in producing building brick must be plastic enough to be shaped, and it must burn to a hard strong brick at about 1,800° to 1,900° F. Color and vitrification range usually are very important. Low-grade surface clays and shales are suitable for building brick. Building brick used for facing brick must fire to a pleasing color. Clay used in manufacturing building brick must not warp during drying and firing.

Clay used to produce paving brick, sewer pipe, and roofing tile must be plastic, and tough when wet, must have high dry strength, must vitrify at about 2,000° F., must have a long vitrification range, and must not warp. Clay used to produce sewer pipe should not contain free iron or lime nodules and should take a good salt glaze. Salt glaze is produced by vaporizing salt in the kiln near the end of the firing cycle. Clay used to produce roofing tile must have low absorption.

Clay used in cement manufacture provide alumina and silica to the cement charge as required.

Clay used for lightweight-aggregate production must either be natural bloating or must be amenable after blending with inexpensive materials such as fuel oil to promote bloating (*6*, *7*).

PRODUCTION

Table 10 gives the production of miscellaneous clays in the United States for 1925–53, inclusive.

TABLE 10.—*Miscellaneous clays, including shale and slip clay, sold or used by producers in the United States, 1925–53*

Year	Short tons	Value
1925	909, 122	$1, 341, 976
1926	519, 440	1, 161, 910
1927	477, 052	1, 049, 976
1928	528, 528	1, 440, 777
1929	444, 400	1, 292, 650
1930	712, 621	1, 670, 718
1931	462, 561	892, 481
1932	223, 520	738, 069
1933	202, 508	872, 588
1934	352, 464	1, 241, 504
1935	363, 069	1, 313, 741
1936	278, 965	487, 058
1937	403, 522	786, 027
1938	389, 715	861, 659
1939	409, 274	714, 205
1940	710, 515	1, 136, 740
1941	1, 210, 168	1, 402, 592
1942	1, 019, 663	1, 177, 474
1943	850, 397	1, 280, 452
1944	[1] 9, 080, 717	6, 550, 269
1945	10, 828, 686	9, 774, 854
1946	20, 190, 344	16, 022, 012
1947	21, 773, 680	17, 422, 962
1948	24, 746, 599	20, 414, 694
1949	23, 725, 565	19, 622, 568
1950	26, 400, 449	23, 241, 210
1951	27, 649, 491	29, 692, 830
1952	26, 507, 910	41, 668, 100
1953	28, 278, 307	42, 435, 964

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[1] Beginning with 1944 this table includes statistics for clays and shales (large tonnages of which are used in manufacturing of heavy clay products and portland cement) other than those discussed in preceding pages. This captive tonnage is included for the first time in 1944.

OUTLOOK

Total clay production will continue to increase with industrial expansion and growth in population, but individual classes of clay will not necessarily follow a uniform growth pattern. Because of the diversity of kaolin applications, the kaolin-production index probably will pace or exceed the Federal Reserve Board Index of Manufactures. Over 90 percent of the ball clay produced goes into products for the home, such as plumbing fixtures, floor tile, and dinnerware; it has closely followed the construction index in the past and doubtless will continue to do so in future. Fire-clay refractories probably will have increasing competition from more expensive but more serviceable nonclay refractories. The trend toward increased and deeper oil- and gas-well drilling will require the use of more bentonite in future. Bentonite demands by the foundry industries also will expand. Fuller's earth requirements of the oil-refining industry have declined slightly in recent years, but the overall tonnage has increased because of expanded use for insecticides, drilling muds, and absorbents. Insecticide and drilling-mud uses have expanded rapidly since 1946 and may be expected to continue to expand in the future. Miscellaneous clays production probably will increase slowly over a period of years. Their use as lightweight aggregate in cement-products manufacture will increase, and it is fair to assume that the face brick and some other heavy clay products demands will increase. For many years structural clay products production has not paced industrial growth. The index for the value of structural clay products (based on 100 for 1947–49) was only 110 in 1953 compared with the index of 124 for all manufacturing industries.

One factor that makes it difficult to evaluate trends for individual classes of clay is substitution of one class for another, something that has taken place frequently in the past. For example, recent trends have been toward increased use of kaolin in refractories because of the shortage of high-alumina fire clays, increased use ot fuller's earth instead of bentonite for insecticides and drilling muds, increased use of activated bentonite instead of fuller's earth in mineral-oil refining, and the use of large quantities of refractory clays in heavy clay products.

PROBLEMS

The problems of the clay industry are as diverse as the properties and uses of the clays themselves.

Diaspore clay is in very short supply and consequently has, in large measure, been replaced by imported bauxite since World War II. Flint clay of superduty quality is not abundant, and in some producing districts its shortage has restricted the output of refractories that use it. Nationally, no other types of clay are in short supply, with the possible exception of catalytic-grade halloysite; however, many regional and local problems would be resolved by the establishment of indigenous supplies.

The problems of clays that regionally are in short supply may become acute through high incoming freight rates that limit the importation from other regions of all but high-unit-value clays or through outgoing freight rates that limit the marketing radius of the finished-clay products. For example, lack of an adequate local supply of whiteware clay has discouraged establishment of electrical-porcelain and sanitary-ware plants in the Pacific Northwest, in spite of a large and rapidly expanding demand. Freight rates are a governing factor in developing and marketing all kinds of clays as such or in end products.

Interindustry relationships may encourage or inhibit the establishment or continuance of nonceramic industries that are heavy consumers of ceramic materials. For example, a commercially satisfactory source of paper filler and coating clay in the Pacific Northwest would find an immediate market in existing paper mills and would encourage the establishment of additional facilities in that area to process locally into finished high-grade papers a larger part of the great paper-pulp output of Washington, Oregon, and southeastern Alaska. In the Rocky Mountain States and in California, known resources of superduty fire clays are inadequate to sustain superduty refractory production of the magnitude required by the expanding metallurgical, glass, and other industries in some Western States. The added cost of shipping large quantities of refractories from distant points adds materially to the cost of manufactured products in these areas. Even in the older refractories-production areas in the Central and Southeastern States, dwindling supplies of certain types of clay (for example, diaspore in Missouri) have necessitated intensified search for new deposits or for substitute materials.

New processes in metallurgical and chemical industries and more exacting requirements in older processes constantly demand refractories to withstand higher temperatures, greater thermal shock, and more corrosive gases or liquids. Some of these new requirements will be met by improved refractories of customary types; others will require development of new materials.

Comprehensive industrial surveys, coupled with field and laboratory studies, are needed in a number of areas to determine the requirements for types of clay in local short supply and to catalog occurrences of promising materials that lie within economic transportation distances from consuming centers.

In some instances there is a need to develop beneficiation processes to produce salable products from offgrade materials, to utilize materials now being wasted or to prepare products to meet special requirements.

The growth and expansion of the chemical and petroleum industries and the development of new chemical processes constantly create new problems for filtering, decolorizing, absorbent, catalytic, carrier, and diluent clays. Constant research on the properties and processing of clays is required to keep abreast of new developments in these fields and to provide new products to meet specific requirements. For example, as regards mineral catalysts, there is little information on the relationship between the fundamental properties of the material and its performance characteristics. It is to the national interest that scientific, technologic, and economic data on these materials be accumulated and disseminated to the public.

The cost and time lag in evaluating clays for industrial uses is a major problem of both producers and consumers of clays. In the ceramic industries a long series of laboratory and semicommercial tests, followed by a full-scale plant run, is required to determine the suitability of a new clay as a substitute for one that is already established. Accelerated, economical, small-scale tests are needed, especially for plasticity, spall, and high-temperature load tests.

There is a shortage of suitable local material for making lightweight aggregate in scattered sections of the United States. The establishment of local supplies of lightweight aggregate in some of these areas may depend upon utilization of clays and shales, in lieu of pumice, perlite, slags, and other aggregates. If natural bloating clays are not available tests are needed to determine what constituents must be added and what processing is required to bloat the local clays economically to prepare them for use as lightweight aggregates.

Scientific control is now standard practice for drilling mud in the oil- and gas-well-drilling industry. In many instances specially prepared bentonites and fuller's earths are being shipped considerable distances to the point of consumption. Lack of knowledge of the properties of local material and of economical processes for treating them often has handicapped utilization of indigenous resources.

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COBALT

By

Hubert W. Davis[1]

COBALT occupies an important position among the alloying metals owing to its ability to withstand high temperatures, its cutting efficiency, and its magnetic properties. Because of progress in metallurgical research on extractive methods, only in recent years has it been possible to produce cobalt in quantity adequate to assure supplies for expanded applications. Reserves are sufficient to sustain new and increased uses.

Summary

The principal uses of cobalt, quantitywise, are in high-temperature alloys and permanent-magnet alloys, but it is important in many other applications. Use of the element has increased spectacularly since World War I. Belgian Congo is the largest producer of cobalt ore and cobalt metal. The United States, the largest consumer of cobalt, has been a smaller producer of cobalt ore but may become the second largest in 1956; however, the United States is a large refiner of crude cobalt and the largest importer of crude cobalt, metal, and oxide.

World output of cobalt will increase progressively as new deposits now being developed are brought into production. This expanded output, plus the quantity to be accumulated in the National Stockpile, probably will suffice for eventual full mobilization requirements.

After the stockpile objective is fulfilled, world production will substantially exceed peacetime requirements. Vigorous research will be required to find new uses and expanded applications for present uses. However, because of the importance of cobalt in many applications, the outlook for expanding present uses and developing new ones is most encouraging.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

SIZE OF DOMESTIC INDUSTRY

Despite the fact that the United States is the largest consumer of cobalt in the world, less than 10 percent of its requirements has been furnished by domestic ore. Table 1 shows production and shipments through 1953.

TABLE 1.—*Cobalt ore produced and shipped in the United States through 1953*

[Pounds of contained cobalt]

Year	Production	Shipments
Previous to 1921 (partly estimated)	730, 000	730, 000
1921–32 (partly estimated)	9, 300	5, 000
1933–39	11, 493	----------
1940–47	5, 198, 452	5, 095, 920
1948	687, 464	580, 703
1949	521, 656	673, 773
1950	809, 328	660, 025
1951	902, 629	755, 631
1952	1, 363, 251	836, 372
1953	1, 258, 924	1, 775, 489
Total	11, 492, 497	11, 112, 913

Several companies have made attempts to establish a regular production of cobalt at Fredericktown, Mo. The results of many years of research have led to a process that may be successful for separating cobalt and nickel, and under an agreement between Defense Materials Procurement Agency and National Lead Co. a 50-ton plant (head feed) was completed in April 1954. Production of cobalt is expected to total about 1,386,000 pounds annually.

Cobalt had long been known to occur as a minor constituent of the iron ore at Cornwall, Pa., but only since 1940 has there been any commercial recovery.

Although the Blackbird district of Idaho was first prospected about 1893 and cobalt minerals were noticed a few years later, no attempt was made to mine cobalt until 1918, when a carlot of concentrate averaging 17.74 percent cobalt was produced. No further production of cobalt was made until 1951. In 1945 a large block of ground in the Blackbird district was acquired by Calera Mining Co. (subsidiary of Howe Sound Co.), which has developed a mine and completed a concentrator of 1,000 tons daily capacity. The concentrator began production in June 1951. The concentrates will be refined at a refinery built by Calera at Garfield, Utah. The refinery, which was completed in August 1952, has a potential capacity of 3,300,000 pounds annually.

As a consequence of these developments, the United States will have capacity to produce about 5,300,000 pounds annually from domestic ore and when the new refineries are placed on a commercial basis will be far less dependent on foreign sources.

Although the United States has been only a small producer of cobalt ore, it is important in the international cobalt trade as a large refiner of crude cobalt and the largest importer of crude cobalt, metal, and oxide. This country is also a large producer of cobalt oxide, salts, and driers.

Table 2 shows the production of cobalt products in the United States for 1948 to 1953.

GEOGRAPHIC DISTRIBUTION OF INDUSTRY

The cobalt industry is now centered chiefly in Africa, where the Union Minière du Haut-Katanga and Rhokana Corp. have their operations. The Union Minière is by far the largest producer of cobalt in the world. In Belgian Congo it has mines, washing and concentrating plants, and rotary and electric furnaces for producing white alloy containing 42–45 percent cobalt, 15 percent copper, and 39 percent iron from ores rich in cobalt as well as an electrolytic refinery for producing metal from copper ores carrying about 0.5 percent cobalt. The crude cobalt or white alloy is refined chiefly to metal and oxide in Belgium in the Oolen plant of the Société Générale Métallurgique de Hoboken and at Niagara Falls, N. Y., in the plant of the African Metals Corp.

At Nkana, Northern Rhodesia, the Rhokana Corp. has a concentrator and reverberatory and electric furnaces for producing white alloy from the cobalt contained in the copper ore. The company completed an electrolytic refinery for producing cobalt metal in January 1952.

In Ontario, Canada, Deloro Smelting & Refining Co. and Cobalt Chemicals, Ltd., have smelting and refining works at Deloro and Cobalt, respectively. The International Nickel Co. of Canada, Ltd., began commercial production of electrolytic cobalt at its Port Colborne refinery in October 1954.

United States refineries are at Niagara Falls, N. Y.; Wilmington, Del.; Garfield, Utah; La-

TABLE 2.—*Cobalt products produced and shipped in the United States 1948–53*

[Pounds]

Product	Production		Shipments	
	Gross weight	Cobalt content	Gross weight	Cobalt content
Metal:				
1948	1, 771, 516	1, 742, 081	2, 135, 444	2, 100, 632
1949	1, 800, 614	1, 772, 519	1, 556, 198	1, 531, 997
1950	1, 850, 145	1, 817, 550	2, 280, 321	2, 240, 834
1951	1, 989, 952	1, 955, 145	2, 022, 560	1, 987, 023
1952	2, 065, 447	2, 028, 964	1, 932, 608	1, 898, 871
1953	2, 887, 487	2, 818, 859	2, 535, 896	2, 480, 840
Oxide:				
1948	547, 393	385, 774	540, 270	380, 462
1949	439, 150	310, 521	387, 654	274, 724
1950	522, 666	371, 215	570, 394	404, 618
1951	637, 456	457, 618	638, 896	458, 455
1952	745, 934	539, 467	708, 674	512, 581
1953	579, 457	415, 974	575, 209	413, 600
Hydrate:				
1948	399, 110	154, 049	420, 245	162, 700
1949	419, 248	167, 033	410, 432	165, 682
1950	262, 479	107, 771	271, 076	110, 917
1951	242, 264	98, 444	244, 216	97, 477
1952	244, 656	96, 326	244, 914	98, 046
1953	255, 386	107, 444	238, 271	99, 858
Salts:				
1948	699, 193	180, 263	821, 704	209, 631
1949	816, 041	208, 995	831, 618	214, 991
1950	1, 292, 491	331, 030	1, 238, 247	322, 300
1951	1, 038, 526	261, 791	1, 104, 562	277, 261
1952	1, 044, 102	256, 223	970, 252	233, 792
1953	1, 212, 007	309, 862	1, 246, 592	320, 672
Driers:				
1948	9, 869, 595	590, 684	10, 008, 193	597, 893
1949	8, 301, 277	490, 360	8, 284, 863	491, 395
1950	12, 471, 700	766, 712	12, 450, 974	764, 839
1951	8, 739, 612	541, 541	8, 801, 734	532, 141
1952	7, 924, 714	480, 616	7, 409, 610	449, 269
1953	9, 140, 138	567, 756	8, 995, 651	554, 297

trobe, Pa.; and Fredericktown, Mo. The United States is also a large producer of cobalt salts and driers, and there are processing plants for manufacturing these products in 16 cities. The largest producing plants, however, are at Cleveland, Ohio, and Elizabeth and Gloucester, N. J.

There is a refinery at London, England, and in France there are refineries at Paris and Savoie.

Refining facilities for cobalt are at three localities in Germany; and at Kristiansand, Norway, a refining unit was completed in July 1952 for treating the cobalt recovered from copper-nickel matte produced in Canada.

GEOLOGY

Cobalt is a relatively rare component of the earth's crust and is estimated as making up only 0.001 percent as compared with 0.02 percent for nickel.

The important cobalt minerals are sulfides, arsenides, and oxidized compounds. The principal sulfide minerals are carrollite ($CuCo_2S_4$) and linnaeite (Co_3S_4). The principal arsenides are smaltite ($CoAs_2$), safflorite ($CoAs_2$), skutterudite ($CoAs_3$), and cobaltite ($CoAsS$). The principal oxidized minerals are asbolite (an impure mixture of manganese and other oxides), heterogenite (a hydrated oxide usually containing copper and occasionally nickel and iron), and sphaerocobaltite ($CoCO_3$), and erythrite ($3CoO{,}As_2O_5 \cdot 8H_2O$). In addition, there are a large number of less known cobalt minerals. Seldom are any of them found in sufficient quantity to be mined for cobalt alone. Consequently, the production of cobalt is chiefly a byproduct or coproduct of other metals, mainly copper.

The most important geologic processes in the concentration of cobalt are: (1) Magmatic differentiation and segregation (perhaps aided by hydrothermal processes); (2) hydrothermal activity leading to the formation of veins and replacement deposits; (3) weathering, with attendant oxidation and leaching, helped in places by the movement of ground water; and (4) deposition of small amounts of cobalt in various sedimentary rocks, followed by further concentration by weathering, or by the activity of hydrothermal solutions or ground water. The relationships of the processes and the importance of associated metals in concentrations of economic importance have been discussed in much detail by Vhay (*3*),[2] who has also described the geology of the Blackbird district, Idaho, which will probably become an important producer of cobalt.

MINING

As virtually all cobalt produced is a byproduct or coproduct of other metals, chiefly copper, a detailed description of mining practice as applied to various metal deposits has been omitted. However, a brief description of the mining practice of Calera Mining Co. at its mine in the Blackbird district, Idaho, has been presented by Huttl (*6*).

METALLURGY

According to Young (*13*):

> Few base metals exhibit, in their processing from ore to refined metal, such diversity of metallurgical operations as does cobalt. Depending upon the original form of the cobalt, whether arsenide, oxide, or sulfide, one may encounter gravity, magnetic, and flotation concentrating operations; blast furnace, reverberatory furnace, and electric furnace practice; operations of roasting, leaching, chemical precipitation, electrolysis, pyrometallurgical and hydrometallurgical oxidation and reduction, with accompanying purification and filtration steps.

Therefore, as the metallurgy varies to accommodate the diverse characteristics of individual deposits of cobalt and embraces many types of processes, descriptions of the metallurgical practice as applied to only a few deposits are included.

In Belgian Congo the high-grade ore from the Musonoi mine, the lower grade ores from it and the Kolwezi mine which are enriched by concentration, and the mixed copper-cobalt ore from the Luiswishi mine are smelted and reduced in electric furnaces. The cobalt and copper obtained from smelting these cobalt-rich ores and concentrates blend with other metals contained in the charge to form two alloys—a white alloy rich in cobalt (42–45 percent) and iron but containing about 15 percent copper, and a red alloy rich in copper and poor in cobalt (6–7 percent). The red alloy is treated in rotary furnaces to yield cobaltiferous slag, which is returned to the electric furnaces to produce white alloy.

The greater part of the reserves of cobalt of Union Minière consists of oxidized copper ores containing also less than 0.5 percent cobalt. The cobalt is recovered from these ores through two distinct methods—by an electrolytic process and by smelting in electric furnaces. The ores or concentrates poorest in cobalt are sent to the electrolytic copper plant, where they are submitted to special treatment to recover the copper and the cobalt. A cobalt precipitate from the electrolytic copper plant is supplied to the cobalt electrolysis section. Deposition is achieved by electrolysis of a pulp containing cobalt hydrate in suspension. The hydrate acts as a neutralizing agent for the acid that is liberated during the electrolysis.

The cathodic cobalt is refined in a 1,600-kv.-a. triphase arc furnace. This refining, similar to the operation applied in the production of extramild basic steels, permits a slightly oxidizing fusion, which scorifies the manganese, and a desulfurization by a reducing slag at a high lime content. The zinc, largely volatilized at the beginning of the operation, is eliminated by a poling operation. The liquid cobalt is then granulated in water and polished for export. The granules average about 99 percent cobalt.

The treatment of white alloy has been described by Young (*13*) as follows:

> The alloy is dissolved in hot 20-percent sulfuric acid in copper baskets. Copper in the alloy does not go into solution, unlike cobalt and iron under these conditions, and the copper residue is shipped to a smelter. The solution of cobalt and iron is treated with sodium chlorate and soda ash, filter pressed and washed. The cake is dissolved in acid and reprecipitated to set free the occluded cobalt. The final precipitation of cobalt from solution is done with sodium hypochlorite to give cobalt hydrate. Reduction occurs in two stages. Cobalt hydrate goes to Co_2O_3, Co_3O_4, CoO, etc., the first heating driving off some oxygen. The next stage is the action of carbon monoxide on lower oxides with eventual reduction to metallic cobalt. Slow heating and reduction are essential to avoid cracking in the rondels. The mesh size of cobalt hydrate and the pelletizing practice also have an influence on cracking. The reduced rondels are cooled under charcoal in drums, passed over a magnetic separator, screened, inspected, and packed in small drums for sale. The final product is usually 97 to 99 percent cobalt.

The ore from the Calera mine is treated in a flotation mill, which has been described (*10*).

USES

The principal applications of cobalt, according to quantity, are in cobalt-chromium-tungsten-molybdenum alloys and permanent-magnet alloys, which combined consumed 71 percent of the total cobalt used in 1953. The cobalt-chromium-tungsten-molybdenum alloys, which are designed to maintain strength and resist wear, corrosion, and erosion at operating temperatures of 1,200° to 1,600° F., are used for various components of gas turbines, as well as in jet aircraft engines and turbosuperchargers. These alloys are also employed in cutting and wear-resisting materials. Permanent-magnet

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

alloys are utilized in motors and generators, control devices, communications equipment, meters and instruments, mechanical devices, and games and novelties. Smaller but important uses of cobalt are in high-speed and low-cobalt alloy steels, alloy hard-facing rods, cemented carbides for cutting tools and armor-piercing projectiles, ground-coat frit for porcelain enamel, pigments, catalysts, electroplating, lacquers, varnishes, paints, inks, stock feed, cobalt-deficient soils, and glazes and as decolorizers. Elemental Cobalt–60 is used for radiographic, therapeutic, and instrument purposes.

SUBSTITUTES AND ALTERNATES

In general, most of the substitutes for cobalt represent downgrading, not only in alloy content but also in performance. According to a report (*7*) of the International Materials Conference, there are relatively few feasible opportunities of replacing cobalt by an element other than nickel.

The properties imparted to high-speed steels by cobalt make it most difficult to recommend substitutes or alternates for that element. The performance of almost any efficient high-speed steel is improved by the addition of cobalt and is approximately proportional to the cobalt content. Low-cobalt, less efficient, or ordinary high-speed steel may be used, with loss in cutting speeds and production rates.

Although less efficient, nickel can be used as an alternate for cobalt as a binder in sintered-carbide tools and armor-piercing projectiles, and some cobalt is saved in these applications by using part nickel.

A nickel-base alloy with low cobalt content can be used as an alternate for some cobalt-base high-temperature alloys.

In the permanent-magnet field, the use of iron-powder magnets, developed in France, appears attractive as a conservation measure but would require extensive equipment, with consequent high capital investment. For many of the less essential uses of magnets, it should be possible to return to the older magnet steels, which make smaller demands on cobalt. It has been clearly demonstrated, however, that substitution of permanent magnet alloys of the Alnico III type, containing no cobalt, for the Alnico V type, containing about 24 percent cobalt, would represent false economy, as the difference in the magnetic characteristics is based on units per gram of nickel of the order of 0.7 to 5.0. Thus, elimination of cobalt requires a sevenfold increase in nickel consumption.

A new magnet alloy—bismanol—was developed by the Naval Research Laboratory. In the magnet, bismuth, manganese, and powdered iron are substituted for cobalt and nickel. However, on the basis of known characteristics of bismanol and in view of the present markets, bismanol does not appear to have a very large potential as a substitute material at this time.

A new class of ferrite permanent-magnet materials has been developed. The material is being manufactured in Europe under the name of Ferroxdure and is soon to be produced in the United States under the names of Magnadur and Indox. This material is an oxidic ceramic of the approximate composition $BaFe_{12}O_{19}$.

Cobalt driers show the most rapid drying times of all the metal driers used in the paint and varnish industry. Manganese, lead, and iron can be used as substitutes.

In case of necessity, cobalt oxide as a stain for ceramic bodies and as underglaze and overglaze ceramic stains could be dispensed with.

PRODUCTION

World output of cobalt before World War II was relatively small and, consequently, was insufficient to permit much expansion in use of this metal. During World War II, however, the capacity of mines to produce cobalt ore and of refining plants to provide metal, oxide, and other forms of cobalt was expanded phenomenally; as a result, industrialists interested in new applications were assured adequate supplies. During the Korean conflict there was further expansion in the capacity of mines, concentrators, and refineries.

New production will soon be forthcoming from Manitoba, Northern Rhodesia, and Uganda and possibly from Cuba.

Sherritt Gordon Mines, Ltd., plans to begin production of cobalt from its nickel-copper property at Lynn Lake, Manitoba, in 1955. Annual production of cobalt is estimated at 300,000 pounds.

Chibuluma Mines, Ltd., in Northern Rhodesia, has reserves of 7,300,000 tons of ore averaging 5.23 percent copper and 0.25 percent cobalt. Production is scheduled to begin late in 1955 and to reach the rate of about 16,000 tons of copper and 500,000 pounds of cobalt annually.

In Uganda the Kilembe mine is reported to have a large indicated reserve of ore carrying 3.5 percent copper and 0.13 percent cobalt. Facilities are planned for producing 2,000 to 3,500 tons of ore daily. Production probably would not begin before 1956.

If present metallurgical research on the recovery of cobalt from the lateritic ore at Nicaro, Cuba, is successful, production of about 700,000 pounds of cobalt annually from present opera-

TABLE 3.—*Mine production of cobalt in chief producing countries, 1939, 1947–53*

[Metric tons of contained cobalt]
(Compiled by Berenice B. Mitchell)

Country	1939	1947	1948	1949	1950	1951	1952	1953
Belgian Congo	1,213	3,590	4,323	4,403	5,148	5,715	6,831	8,278
Canada	332	260	701	281	265	432	645	796
Morocco, French	680	212	221	209	420	680	1,000	600
Northern Rhodesia	1,598	420	367	402	670	678	585	677
United States (shipments)	--------	307	263	306	299	343	379	805
All other	510	238	226	300	400	500	600	800
Total	4,300	5,000	6,100	5,900	7,200	8,300	10,000	12,000

tions is anticipated, possibly beginning in 1956. Lateritic ores containing cobalt are known to occur in Brazil, New Caledonia, Celebes, Venezuela, and the Philippines; if the metallurgical research at Nicaro is successful, these countries will be potential sources of cobalt.

The copper ore produced by Boleo Copper Co., Santa Rosalia, Baja California, contains 0.02 to 0.86 percent cobalt and converter slag 1 to 4 percent cobalt. Possibly 600,000 pounds of cobalt might be recovered annually if a successful metallurgical extraction process is developed.

Numerous occurrences of cobalt have been reported in Mexico, but no information descriptive of the deposits is available. The deposits near Alamos, Sonora, and Culiacan, Sinaloa, give some promise of becoming potential producers of cobalt.

Despite the fact that cobalt has been reported produced in about 17 countries, 5 of them—Belgian Congo, Canada, French Morocco, Northern Rhodesia, and United States—will, as in the past, continue for many years to supply the bulk of the production.

SECONDARY COBALT

Substantial quantities of cobalt scrap are generated at plants of certain consumers, and in normal times little effort has been made to utilize it. Because of the shortage of cobalt during recent years, much scrap has been recovered and used successfully. For example, 1,541,000 pounds of cobalt in purchased scrap was consumed in 1953.

The quantity of cobalt potentially available in the United States in scrap in various forms, such as high-temperature alloy scrap and permanent-magnet alloy scrap, together with discarded scrap from obsolete, damaged, or surplus parts or engines, represents a substantial reserve of cobalt. Probably enough secondary scrap material containing cobalt is available to meet all nonessential civilian requirements during an emergency.

RESERVES

The reserves of cobalt in the United States have been estimated by the Bureau of Mines and the Geological Survey, as shown in table 4.

The class 1 deposits include primarily those at Cornwall, Pa., Fredericktown, Mo., and the Blackbird district, Idaho. The Cornwall deposit constitutes the largest class 1 reserve; it contains only about 0.05 percent cobalt, which is intimately associated with pyrite. However, despite the large reserve, recovery of cobalt is governed by the rate of mining iron ore. The copper-cobalt ores of Blackbird, Idaho, carry about 0.6 to 0.8 percent cobalt, about twice as much copper, and a little nickel and gold. The complex lead-copper-nickel deposits at Fredericktown, Mo., contain about 0.3 percent cobalt.

Although the class 2 deposits contain the largest reserve of cobalt, the problem of recovering the cobalt by present methods of treating these ores has not been solved. Moreover, some of the ores—for example, the sulfide ores in the Appalachian region—contain only a few hundredths to 0.1 percent cobalt.

Belgian Congo, on which the United States has depended for most of its supply of cobalt, at present has the largest commercial reserves in the world. On the basis of a rate of production of 6,000 metric tons annually, the Union Minière du Haut-Katanga reports reserves of cobalt adequate for about 40 years and anticipates that these will increase as a result of further development of its copper deposits.

CONSUMPTION

The growth in consumption of cobalt in the United States has been spectacular. Before the World War of 1914–18 the consumption in the United States was small and consisted chiefly of cobalt oxide used in the ceramic industry. Before 1914 scarcely any cobalt metal was produced. During that war, however, the demand for metal for high-speed steel, high-temperature alloys, and magnets grew rapidly, yet total consumption of cobalt in all forms was only a

TABLE 4.—*Estimated reserves of cobalt in the United States as of 1943*
[Short-tons]

Availability	Measured and indicated		Inferred		Total	
	Crude ore	Cobalt content	Crude ore	Cobalt content	Crude ore	Cobalt content
Class 1	45, 350, 000	30, 000	64, 300, 000	59, 000	109, 650, 000	89, 000
Class 2	43, 000, 000	32, 500	125, 000, 000	75, 000	168, 000, 000	107, 500

few hundred thousand pounds. Following the Armistice, the requirements for cobalt dropped abruptly. During 1921–25 the average annual consumption of cobalt in the United States was only about 310,000 pounds, about 45 percent of which was in the form of metal. During 1926–30 the average jumped to about 832,000 pounds annually, of which about 61 percent was used as metal. Although the average annual consumption of cobalt during 1931–35 declined to about 718,000 pounds, of which about 45 percent was metal, it is noteworthy that the drop in consumption of cobalt metal was much less than for the other metals. During 1936–40 consumption advanced to an average of about 1,700,000 pounds annually, of which about two-thirds was metal. Because of the greatly increased demand for cobalt for high-temperature alloys, magnets, high-speed and other steels, ground-coat frit for porcelain enamel, and other applications, consumption rose to nearly 3,000,000 pounds in 1941, exceeded 5,000,000 pounds in 1948, was slightly less than 10,000,000 pounds in 1951, and was nearly 11,000,000 pounds in each of 1952 and 1953.

The combined consumption of cobalt in all Free World foreign countries in 1950 was 23 percent less than in the United States. The largest foreign consumer of cobalt is United Kingdom, which consumed 2,251,000 pounds in 1950. West Germany and France are the second and third largest consumers of cobalt in Europe and Japan is the largest in Asia.

FOREIGN TRADE[3]

Being the largest consumer of cobalt and dependent chiefly on foreign countries for its supplies, the United States is also the largest importer.

Table 6 shows the spectacular growth in imports since 1925.

Exports of cobalt from the United States are relatively small and during recent years have comprised largely cobalt scrap and alloys.

Belgian Congo, Northern Rhodesia, and French Morocco, which are among the largest producers of cobalt, are also the largest exporters, because no cobalt is consumed in any of these countries.

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[3] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

TABLE 5.—*Cobalt consumed in the United States, 1949–53, by uses*
[Pounds of cobalt]

Use	1949	1950	1951	1952	1953
Metallic:					
High-speed steel	283, 496	235, 227	316, 064	223, 203	217, 652
Other steel	162, 638	252, 885	79, 885	115, 761	162, 185
Permanent-magnet alloys	1, 194, 920	2, 834, 040	2, 052, 042	1, 664, 842	2, 336, 889
Soft-magnetic alloys	42, 965	37, 552	58, 652	18, 727	11, 559
Cobalt-chromium-tungsten-molybdenum alloys	1, 238, 083	2, 226, 199	4, 899, 591	6, 408, 537	5, 321, 689
Alloy hard-facing rods and materials	82, 965	260, 371	575, 268	505, 367	591, 909
Cemented carbides	118, 522	136, 935	297, 751	610, 750	359, 125
Other metallic	116, 344	208, 574	276, 222	132, 917	233, 428
Total metallic	3, 239, 933	6, 191, 783	8, 555, 475	9, 680, 104	9, 234, 436
Nonmetallic (exclusive of salts and driers):					
Ground-coat frit	424, 051	683, 358	448, 983	309, 167	374, 158
Pigments	188, 606	262, 441	50, 073	85, 262	102, 612
Other nonmetallic	84, 336	43, 826	60, 462	42, 960	84, 293
Total nonmetallic	696, 993	989, 625	559, 518	437, 389	561, 063
Salts and driers: Lacquers, varnishes, paints, inks, pigments, enamels, glazes, feed, electroplating, etc. (estimate)	765, 000	1, 102, 000	818, 000	701, 000	953, 000
Grand total	4, 701, 926	8, 283, 408	9, 932, 993	10, 818, 493	10, 748, 499

TABLE 6.—*Cobalt imported for consumption in the United States*
[Pounds]

Year	Gross weight					Total	
	White alloy	Ore	Metal	Oxide	Sulfate and other compounds	Gross weight	Cobalt content (estimated)
1925		34, 782	198, 669	287, 265	13, 256	533, 972	408, 000
1929		434, 443	806, 640	475, 928	64, 782	1, 781, 793	1, 212, 000
1935	378, 848	419, 110	563, 866	557, 083	80, 554	1, 999, 461	1, 167, 000
1940	7, 843, 828	2, 653, 891	130, 321	756, 759	11, 468	11, 396, 267	4, 200, 000
1945	8, 397, 145	859, 940	946, 475	120, 672	224	10, 324, 456	4, 615, 000
1947	3, 751, 452	751, 438	6, 035, 153	752, 150	530	11, 290, 723	8, 206, 000
1948	4, 879, 413	8, 167, 545	5, 266, 521	790, 300	1, 374	19, 105, 153	8, 821, 000
1949	3, 691, 051	109, 009	5, 588, 327	360, 318	359	9, 749, 064	7, 458, 000
1950	3, 979, 088	164, 188	6, 706, 875	904, 650	4, 649	11, 759, 450	9, 095, 000
1951	4, 083, 541	537, 309	8, 119, 326	436, 517	3, 157	13, 179, 850	10, 338, 000
1952	6, 113, 102	215, 572	12, 014, 920	386, 935	12, 759	18, 743, 288	15, 031, 000
1953	5, 249, 781	445, 063	14, 431, 894	610, 054	273, 286	21, 010, 078	17, 237, 000

Although no cobalt ore is produced in Belgium, the country is important in the international cobalt trade. It is the largest European refiner of crude cobalt and the largest European exporter of cobalt metal and oxide.

TARIFF

Cobalt metal and ore enter the United States duty free. The duty on cobalt oxide is 5 cents a pound, sulfate 2½ cents a pound, linoleate 5 cents a pound, and salts and compounds 30 percent ad valorem.

SELF-SUFFICIENCY

Before 1940 the United States depended wholly on foreign sources, chiefly Belgian Congo, for its supply of cobalt, and since 1940 it has depended on foreign sources for 90 to 95 percent of its supply. However, the hope of the United States to become less dependent on foreign sources will materialize when the Calera Mining Co. and National Lead Co. attain capacity production of cobalt metal at Garfield, Utah, and Fredericktown, Mo., respectively. Currently, the Pyrites Co. produces several thousand pounds annually of cobalt metal and other cobalt products at Wilmington, Del. Accordingly, the United States will have a potential capacity to produce cobalt from domestic ore at an annual rate of 5,300,000 pounds, which is greater than domestic consumption in any year before 1950. However, even this anticipated production falls far short of meeting requirements of industry and the National Stockpile. Following fulfillment of defense requirements, the demand for cobalt will probably decline substantially. Nevertheless, because of the increasing importance of cobalt in many applications, an annual peacetime consumption of 10,000,000 pounds within the next 10 years is envisioned; the United States would still depend on foreign sources for about 47 percent of this supply.

PRICES

The price of cobalt metal remained unchanged at $1.50 a pound from October 1939 to July 1, 1947, when it was advanced to $1.65. Subsequently, four price increases have been made, chiefly because of higher production costs.

The prices of cobalt metal (97–99 percent, in kegs of 500–600 pounds) f. o. b. Niagara Falls or New York, N. Y., and of ceramic-grade oxide east of the Mississippi River during recent years have been as follows:

TABLE 7.—*Prices per pound of cobalt metal and cobalt oxide in the United States, 1940–53*

Year	Cobalt metal f. o. b. Niagara Falls or New York, N. Y.	Cobalt oxide east of Mississippi River (gross weight)
1940–46	$1. 50	$1. 16
1947	1. 50–1. 65	1. 16–1. 27½
1948	1. 65	1. 27½
1949	1. 65–1. 80	1. 27½–1. 38
1950	1. 80	1. 38
1951	2. 10–2. 40	1. 60–1. 82
1952	2. 40	1. 82
1953	2. 60	1. 96

DEFENSE PROGRAM

Under amended Order M–10, the National Production Authority, after January 31, 1951, prohibited the use of cobalt in the manufacture of specified products that are relatively less essential or in which substitute materials may be effectively used; established limits on additions to inventory; and placed cobalt under allocation by prohibiting (subject to limited exceptions) any deliveries not covered by allocation authorization to be issued monthly by the NPA.

Removal of allocation control on cobalt was announced June 30, 1953, by NPA, through issuance of amendments or revocations affecting Order M-80 and its schedules.

At the request of Defense Minerals Administration, the Calera Mining Co. agreed to increase the milling plant at its mine from a rated capacity of 600 tons to 1,000 tons of ore per day and to proportionately enlarge the refinery so that its rated capacity will exceed 3,000,000 pounds of cobalt annually.

Under an agreement between Defense Materials Procurement Agency and the National Lead Co., a cobalt-nickel-copper separation plant at Fredericktown, Mo., was expected to begin operation in 1955 and was expected to produce cobalt at an annual rate of 1,386,000 pounds. The metal will be processed from rejected iron concentrates from the production of lead at the company Fredericktown concentrator.

A number of applications for loans to finance exploration for cobalt have been received by Defense Minerals Exploration Administration. Three applications for exploration in Idaho, 1 in Missouri, and 1 in Tennessee were approved.

RESEARCH

New ways of extracting copper, nickel, cobalt, and other important metals from ores by chemical methods rather than by the usual smelting and refining have been announced by the Chemical Construction Corp. Some details of the process, which is described as basically new and revolutionary, have been given in Mining Engineering (*9*). The new process is the result of research by the company, a subsidiary of American Cyanamid Co., and of research and pilot-plant work by Sherritt Gordon Mines, Ltd. The new process is used by the Calera Mining Co. and will be used by Cobalt-Nickel Reduction Co.

OUTLOOK

Despite the fact that production of cobalt in the United States is expected to reach an annual rate of 5,300,000 pounds by the end of 1955 and that world output will expand progressively for several years, the total supply will fall short of meeting requirements in case of early full mobilization. Following fulfillment of the National Stockpile objective, the potential world production of cobalt will greatly exceed world requirements.

As a result of extensive research in the past, the use of cobalt has expanded spectacularly as additional supplies were made available. Because of the fact that industrialists interested in new applications of cobalt will be assured adequate supplies, continuing research will result in developing new uses. Also, because of the importance of cobalt in many applications, the outlook for expanding the present uses of cobalt is most encouraging. Because of their outstanding characteristics, the future potentialities of cobalt-base high-temperature alloys appear promising, and because of the importance of permanent magnets in modern industry many new applications will be found.

An important use of cobalt, although only a relatively small quantity is consumed at present, is in preventing and curing certain deficiency diseases of cattle and sheep. Insufficient cobalt in soils is correspondingly reflected in the forage crops grown thereon, and the herbivorous animals suffer various types of anemia and malnutrition in consequence. These can be prevented by adding cobalt salts to the soil and cured by including them in the feed. A more widespread application of cobalt to unhealthy pastures and in stock feed would result in a substantial outlet for the mineral.

All of the most promising known deposits of cobalt ore in the United States have now been developed, and the problem of refining the ores will probably soon be solved.

The most promising source for finding commercial deposits of cobalt ore in the United States is doubtless the Blackbird district of Idaho. Three companies have been doing exploration in this district under loans from DMEA.

Additional production will soon be forthcoming from newly developed deposits in Manitoba, Northern Rhodesia, and Uganda. Recovery of cobalt from the lateritic ore of Cuba is promising. Improvement in cobalt recovery at operations in Belgian Congo and Northern Rhodesia and a metallurgical extraction method to recover the cobalt contained in the copper ore at Santa Rosalia, Baja California, would add substantially to the world supply.

When the National Stockpile objective is attained and if high-level industrial activity is continued, the demand for cobalt will decline substantially. However, it is believed that the pre-Korean peacetime uptrend will project itself into the postemergency period. In 1948, for the first time, consumption of cobalt in the United States exceeded 5,000,000 pounds. Because of the importance of cobalt in its many applications it is anticipated that the peacetime demand will expand greatly, that eventually consumption in the United States alone might reach 10,000,000 pounds annually, and that usage abroad will also expand. Nevertheless, for many years the peacetime demand for cobalt will be substantially less than potential production.

PROBLEMS

The industry will face a shortage of cobalt in case of early full mobilization and a surplus after the National Stockpile objective is fulfilled.

The low recovery of cobalt at the operations in Africa and the loss of cobalt contained in certain ores, slag, and scrap waste an important resource.

Cobalt occurs with a number of other metals, including copper, nickel, silver, and manganese. Since the other metal is the principal recovery at most operations, the extractive metallurgy of cobalt figures in as many processes as the primary metals with which it occurs. At present the cobalt metallurgy associated with copper refining dominates the scene. There is very little flexibility in production of cobalt to meet wide fluctuations in demand.

The world's largest potential source of cobalt is the laterite ores of Cuba, yet there has been no output because of lack of an economic and feasible process for its recovery. The Cuban ores also contain nickel, chrome, and iron, but thus far only nickel has been recovered; commercial recovery of all of the valuable minerals would probably make long-term operation at the Nicaro plant more attractive.

The high cost of recovering cobalt by present processes restricts its use and forces the employment of less desirable substitutes. For example, cobalt is superior to nickel for certain uses; but, because it costs four times more, efforts are directed toward maximum use of nickel. Thus, a major problem in the case of cobalt is its high recovery cost and resultant high price.

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COLUMBIUM AND TANTALUM

By

Horace T. Reno[1]

COLUMBIUM and tantalum are two rare metals important to United States economy in peacetime and vital in wartime. Columbium is essential in the manufacture of special high-temperature alloys, and tantalum is equally important in the manufacture of electronic equipment. Ninety-nine percent of the columbite and tantalite processed in the United States is imported, and the world supply has seldom been adequate since before World War II.

Summary

Columbium (also called niobium) and tantalum occur in certain complex minerals that have been divided into five groups—pyrochlore, fergusonite, columbite, samarskite, and eschynite. Two of these, the columbite and pyrochlore groups, are the commercial source of columbium and tantalum. The other three are of interest only because they include minerals that contain rare-earth elements. Columbite-tantalite minerals occur in granite pegmatite rocks or alluvium derived from pegmatites; the pyrochlore minerals occur in alkalic rocks and alluvium. Most columbium and tantalum have been obtained from columbite-tantalite minerals in the columbite group, but in 1953 a process was developed whereby columbium could be extracted from pyrochlore.

The principal columbite-producing countries are Nigeria, Belgian Congo, and the Federation of Malaya. Tantalite is produced principally in Brazil, Australia, and Southern Rhodesia. In the United States, South Dakota, New Hampshire, Idaho, Connecticut, and Maine are sources of columbite-tantalite ore.

The Electro Metallurgical Division of Union Carbide & Carbon Corp. is the major processor of columbite ore, and Fansteel Metallurgical Corp. the major processor of tantalite ore.

Columbium and tantalum are extracted together from their ores by fusing the ore with caustic soda, treating the melt with hydrochloric acid, and then dissolving it in hydrofluoric acid. The metals are selectively precipitated from this solution as tantalum potassium fluoride and columbium-potassium oxyfluoride by adding potassium fluoride. Pure tantalum is made by electrolysis of the fused potassium fluoride and pure columbium by reducing the oxide with the carbide.

The primary use of columbium has been as ferrocolumbium, a stabilizer used in manufacturing stainless steel. Tantalum is used mostly in chemical equipment and corrosion-resistant tools. Substitutes have been developed to replace columbium and tantalum in many applications. Requirements for these metals were decreased by the development of low-carbon stainless steels to replace columbium-stabilized steels.

A shortage of columbium and tantalum during World War II caused the Government to place restrictions on their use until late in 1944. During the Korean War a shortage again developed that continued until November 1953. The Defense Materials Procurement Agency began buying domestic and foreign columbium and tantalum ores on May 28, 1952; its successor agency, General Services Administration, was authorized to continue buying foreign ores until December 31, 1956, and domestic ores until 2 years later or until 15,000,000 pounds of contained combined pentoxide has been bought.

The outlook for the columbium-tantalum industry is one of change. The cost of production probably will decrease, more ore will be mined in the United States, and demand and supply will increase.

High prices, periodic shortages, and almost complete dependence upon imports cause most of the problems of the columbium-tantalum industry.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

MINERALOGY

As already stated, columbium and tantalum occur in complex minerals that have been divided into five groups by Dana—pyrochlore, fergusonite, columbite, samarskite, and eschynite (*2*).[2] The columbite group was the only commercial source of columbium and tantalum until 1953, when a process was developed in Norway for treating ore containing pyrochlore minerals. Minerals in the fergusonite, samarskite, and eschynite groups are of interest because some of them contain rare-earth elements.

Columbite and tantalite, the principal minerals of the columbite group, form a continuous iron-manganese isomorphous series (FeMn) $(Cb\text{-}Ta)_2O_5$. Both columbium and tantalum are usually present in a member of the series. If the member contains more columbium than tantalum it is called columbite; conversely, if it contains more tantalum than columbium, it is called tantalite. Columbite-tantalite is usually iron-black, opaque, hard, and brittle. It has a submetallic or often a brilliant subresinous luster, a specific gravity of 5.3 to 7.3, and a subconchoidal to uneven fracture. No pure mineral of the group is susceptible to magnetic attraction.

Pyrochlore, a complex columbate of titanium, thorium, fluorine, and cerium metals and calcium or other bases, is the principal mineral of the pyrochlore group. It grades from brown to reddish or blackish brown and is subtranslucent to opaque, hard (5 to 5.5), and brittle. It has a specific gravity of 4.2 to 4.36, a conchoidal fracture, and a vitreous or resinous luster.

SOURCE

The columbite group of minerals commonly occurs in granite pegmatite and in alluvium derived from pegmatites. Most columbite and tantalite are obtained as byproducts of tin (cassiterite) placer mining. The remainder is mined from pegmatite lodes, usually as coproducts of beryl, spodumene, mica, or cassiterite.

Pyrochlore minerals occur in alkalic rocks. Low-grade deposits have been known since 1918 but were not mined until 1953 when the Norwegians began to exploit the Sove ore fields near Ulefoss, Norway. In the United States the bauxites derived from columbium-bearing alkalic rocks have been investigated as a source of columbium. Columbium is associated in bauxite with ilmenite (iron titanium oxide). The Federal Bureau of Mines, working closely with industry, is investigating methods of concentrating the ilmenite and separating the columbium from the concentrate (*5*).

TABLE 1.—*Columbium-tantalum concentrates shipped from mines in the United States, 1944–48 (average) and 1949–53* [1]

Year	Pounds	Value
1944–48 (average)	4, 880	$11, 300
1949	1, 020	1, 785
1950	1, 000	2, 150
1951	925	1, 528
1952	5, 385	16, 723
1953 [2]	14, 867	29, 779

[1] Includes columbite, tantalite, and microlite.
[2] Containing 3,264 pounds of columbium and 2,906 pounds of tantalum.

The principal columbite-producing countries of the world, in the order of their contribution to the total produced in 1952, are: Nigeria, Belgian Congo, Federation of Malaya, Mozambique, Uganda, Madagascar, United States, Brazil, and French Equatorial Africa.

The principal tantalite-producing countries of the world, in the order of their contribution to the total produced in 1952, are: Brazil, Australia, Southern Rhodesia, Union of South Africa, South-West Africa, and Nigeria.

During 1948–53, 23,197 pounds of columbite-tantalite concentrates was produced in the United States, principally in South Dakota, New Hampshire, Idaho, Connecticut, and Maine.

RESERVES

Columbite-tantalite minerals are usually ore reserves only in those deposits where they can be recovered as byproducts. Pyrochlore mineral deposits at present are ore only in Norway, although deposits are being investigated in Canada and Africa. Reserve figures are not available for publication.

INDUSTRY

Electro Metallurgical Division, Union Carbide & Carbon Corp., Niagara Falls, N. Y., which produces ferrocolumbium and ferro-

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

TABLE 2.—*Columbite imported for consumption in the United States, 1944–48 (average) and 1949–53, by countries*

[Pounds]

(Compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce)

Country	1944–48 (average)	1949	1950	1951	1952	1953
Australia						25, 119
Belgian Congo	23, 584	198, 585	400, 868	177, 273	354, 732	580, 232
Belgium-Luxembourg [1]	5, 425					
Bolivia	[2] 1, 574				14, 678	10, 375
Brazil	2, 928	8, 568	10, 981	6, 377	5, 017	41, 005
British Guiana					800	2, 324
India	294					
Japan [1]			31, 835			
Korea, Republic of						2, 000
Malaya					20, 264	101, 967
Mozambique	4, 409	1, 200		17, 082	21, 205	57, 894
Nigeria	2, 986, 443	1, 349, 126	1, 280, 930	1, 336, 041	1, 450, 787	3, 167, 344
Norway						26, 233
Portugal			2, 103			68, 121
Southern Rhodesia						20, 460
Spain						4, 410
Sweden						30, 847
Uganda [3]	11, 397				4, 622	19, 891
Union of South Africa	364				6, 030	34, 472
United Kingdom [1]	240					
Total: Pounds	3, 036, 658	1, 557, 479	1, 726, 717	1, 536, 773	1, 878, 135	4, 192, 694
Value	$953, 710	$561, 945	$752, 926	$1, 362, 393	$2, 368, 769	$6, 905, 497

[1] Presumably country of transshipment rather than original source.
[2] Classified by U. S. Department of Commerce as from Chile, some of which is believed to be the country of transshipment only.
[3] Classified by U. S. Department of Commerce as British East Africa.

TABLE 3.—*Tantalite imported for consumption in the United States, 1944–48 (average) and 1949–53, by countries*

[Pounds]

(Compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce)

Country	1944–48 (average)	1949	1950	1951	1952	1953
Anglo-Egyptian Sudan	20					
Argentina	1, 861					
Australia	8, 082			1, 467	1, 590	20, 541
Belgian Congo	297, 372	38, 086	211, 433	210, 402	236, 701	507, 282
Belgium-Luxembourg [1]	640		85, 683	20, 876		
Brazil	137, 519	63, 478	13, 378		49, 813	39, 532
Canada	140					
French Guiana						10, 987
India	488					
Japan [1]			10, 691			
Malaya					2, 087	3, 639
Mozambique	950					
Netherlands [1]		29, 500				
Nigeria	14, 417	4, 480	7, 543	5, 700	2, 273	
Portugal					35, 428	154, 323
Southern Rhodesia	9, 320				233	8, 163
Spain					741	
Sweden						4, 242
Uganda [2]	3, 725					2, 050
Union of South Africa	909	1, 120				2, 036
Total: Pounds	475, 443	136, 664	328, 728	238, 445	328, 866	752, 795
Value	$384, 949	$237, 292	$244, 205	$190, 383	$398, 849	$1, 208, 134

[1] Presumably country of transshipment rather than original source.
[2] Classified by U. S. Department of Commerce as British East Africa.

tantalum-columbium, is the major processor of columbite ore.

Fansteel Metallurgical Corp., North Chicago, Ill., which produces tantalum and columbium metals and compounds, is the major processor of tantalum ore.

Kennametal Inc., Latrobe, Pa., processes columbite-tantalite ore and manufactures columbium-tantalum-bearing carbides.

TECHNOLOGY

The English chemist, Hatchett, discovered a new element in 1801 while analyzing a mineral from New London, Conn.; he called it columbium after "Columbia," its country of origin. The following year Ekeberg, a Swedish chemist, discovered tantalum, which he named after the mythical god, Tantalus, because he had such a "tantalizing" time dissolving the mineral in acids. In 1844 Rose announced discovery of an element similar to tantalum, which he called niobium after Niobe, the daughter of Tantalus. For some time a few investigators thought that columbium, niobium, and tantalum were identical. It was later shown, however, that tantalum was a different element and that only columbium and niobium were identical. The name niobium has been adopted by the American Chemical Society (*3*).

Columbium is a steel-gray, lustrous metal about as hard as wrought iron. It is malleable and ductile and has the lowest work function (the energy required to remove an electron from the surface) of any pure refractory metal. The specific gravity of columbium is 8.4 at 20° C.; melting point, 2,500° C.; boiling point, 3,300° C.; atomic weight, 92.91; and atomic number, 41. The metal, which will not tarnish in air at room temperatures, when heated in air becomes coated with an oxide film that retards further oxidation. Columbium is insoluble in hydrofluoric acid but completely soluble in concentrated sulfuric acid.

Tantalum, a silvery metal about as soft as copper, is remarkably ductile, malleable, and tough. Its specific gravity is 16.6; melting point, 2,996° C.; boiling point, above 4,100° C. (some investigators have found the boiling point to be 5,300° C.); atomic weight, 180.9; and atomic number, 73. Pure tantalum is similar to glass in resisting corrosion and chemical attack. It may be dissolved, however, by fused alkalis or by a mixture of nitric and hydrofluoric acids (*4*).

The chemical similarity between columbium and tantalum makes their separation difficult and their quantitative determination time consuming and often inaccurate.

Columbium and tantalum are extracted together from their ores. The method used by the Fansteel Metallurgical Corp. is as follows: The ore is fused with caustic soda at red heat until it decomposes to form sodium columbotantalate. This fused mixture is washed by decantation to remove the excess caustic soda and then digested with hot concentrated hydrochloric acid to form tantalocolumbic acid and to dissolve impurities, such as iron and manganese, which are removed by decantation. The tantalocolumbic acid, a solid, is dissolved in hydrofluoric acid, and enough potassium fluoride is added to make tantalum-potassium fluoride and columbium-potassium oxyfluoride. The resulting solution is pumped through a filter press to remove impurities and allowed to cool. Tantalum potassium fluoride crystallizes and is collected on a filter. The remaining columbium-bearing solution is concentrated to a point just short of crystallization and treated with alkali to yield crude columbium oxide.

Pure tantalum is obtained by electrolysis of the fused potassium fluoride. A cast-iron pot in which the potassium fluoride is held is used for the cathode and a graphite rod for the anode. The metal collects on the cathode as a crystalline powder, which is formed into ingots by subjecting it to high pressure in a vacuum furnace at sintering, but not melting, temperature.

Pure columbium is obtained by heating a stoichiometric mixture of columbium oxide and columbium carbide in a vacuum. Carbon monoxide is evolved, leaving a residue of pure columbium (*7*)

USES

Columbium has been used principally as ferrocolumbium or ferrocolumbium-tantalum in the manufacture of stabilized stainless steels. However, columbium-bearing high-temperature alloys, unsurpassed for certain applications in jet engines and gas turbines, may now require more columbium than the stainless steels. Other uses are as a getter in vacuum tubes and as the carbide in the manufacture of tungsten carbide cutting tools. It is used also in certain nuclear reactors.

High mechanical strength combined with extremely high resistance to chemical attack make tantalum a valuable material for constructing chemical equipment and fabricating corrosion-resistant tools, such as those used in dentistry and surgery. It has been used for cranial plates, suture wire, and bone screws. Tantalum will rectify ordinary alternating current when serving as an anode because it is quickly coated with a nonconducting oxide film that allows current to flow in only one direction. It is used, therefore, in battery-charging rectifiers such as the familiar trickle charger and in a device for eliminating "B" batteries in radio sets. Tantalum is used also in electronics as anode and grid material for transmitting tubes. Porous tantalum compacts and tantalum foil are used in electrolytic condensers, the capacities of which are governed by the area of tantalum exposed to the electrolyte (*4*).

Tantalum substituted for columbium in high-temperature alloys generally provides equal strength and creep resistance but does not provide satisfactory resistance to intergranular corrosion. Tantalum carbide is used with tungsten carbide to make fine-wire-drawing dies and cutting tools because it is extremely hard and has a melting point of about 4,000° C. Tantalum-potassium fluoride was used extensively during World War II as a catalyst in synthetic rubber plants.

SUBSTITUTES AND CONSERVATION

Inasmuch as the supply of columbium and tantalum since 1940 has seldom been adequate, industry has actively sought ways of conserving these metals. Titanium, vanadium, and tantalum have been investigated as substitutes for columbium. Type 321, 18–8 stainless steel made with titanium is a satisfactory substitute for columbium-stabilized stainless steel, except that its use increases the percentage of fabrication scrap. Ferrocolumbium-tantalum made from low-grade ore may be used to replace ferrocolumbium in the manufacture of high-temperature alloys and stabilized stainless steel. The greatest saving of columbium, however, was accomplished by developing maximum 0.03 percent carbon, 18–8 stainless steel.

Tantalum can be replaced in some applications by zirconium, titanium, tungsten, molybdenum, and platinum. Zirconium-coated molybdenum, platinum-coated molybdenum, and a mixture of platinum and graphite can replace tantalum in some types of radio tubes.

WAR-ECONOMY SUPPLY-DEMAND PATTERN

During World War II columbium and tantalum became strategic metals, and because the demand exceeded the supply Government supervision of the industry became necessary. Ore imports were controlled after April 8, 1942; tantalum in all forms was allocated to consumers after May 22, 1942; and ferrocolumbium was allocated to consumers after March 24, 1943. In 1943 two-thirds of the tantalite and some of the columbite imports arrived in the United States via air. The shortage ended in 1944. All restrictions on the use of tantalum ended on December 6, 1944, and restrictions on the use of ferrocolumbium on July 9, 1945.

During the Korean War columbium and tantalum were again controlled by the Government. The National Production Authority on October 20, 1950, announced that the production, distribution, and use of ferrocolumbium and ferrocolumbium-tantalum were to be governed by priorities, that these products could be bought only by the Department of Defense and the Atomic Energy Commission, that the use of ferrocolumbium steels was prohibited where ferrocolumbium-tantalum steels could substitute, and that the use of either type was prohibited if substitutes were available.

On March 19, 1951, the National Production Authority placed columbium and tantalum on the list of materials allocated to defense production and limited the percentage of columbium that could be used in making stainless steel. In December 1951, however, Order M–80 was issued, permitting the use of columbium and tantalum in the manufacture of products required by the Department of Defense and the Atomic Energy Commission.

The Defense Materials Procurement Agency began buying domestic and foreign columbium and tantalum ores on May 28, 1952, paying an incentive bonus of 100 percent of the specified price to actual producers. The General Services Administration, successor to the Defense Materials Procurement Agency, was authorized to continue buying foreign ores until December 31, 1956, and domestic ores until December 31, 1958, or until 15,000,000 pounds of contained combined pentoxides (Cb_2O_5 plus Ta_2O_5) of both domestic and foreign origin has been bought.

On November 1, 1953, Order M–80, which regulated the distribution and use of columbium-tantalum steels, was revoked.

OUTLOOK

The high price of columbite and tantalite ore probably will drop sharply when the Government purchasing program for foreign ores ends on December 31, 1956. Lower prices should stimulate research designed to broaden the use of columbium and tantalum, and many old markets lost because of past conservation and high prices will be regained.

In the United States placer deposits, a possible source of columbium and tantalum, will be investigated as a result of the discovery and exploitation of columbium-tantalum-bearing alluvial gravels in Valley County, Idaho. It is reasonable to believe that new deposits will be found.

The Federal Bureau of Mines laboratories are investigating processes for recovering columbium from the columbium-bearing titanium ores of Arkansas and columbium and tantalum from the placer deposits of Idaho. This work, including investigation of ore-dressing methods, has been partly successful on a laboratory scale; a liquid-liquid extraction process has been developed, the commercial application of which may solve the problem of extracting columbium and tantalum from complex ores.

PROBLEMS

High prices, periodic shortage of columbium and tantalum ore, and almost complete dependence upon imports cause most of the problems of the columbium-tantalum industry.

High prices, resulting from an expensive extraction process, limit the use of columbium and tantalum. The cost of extraction is not the limiting factor during times of ore scarcity; but, when supplies of columbium and tantalum ore increase, finding a less expensive extraction process is the primary problem that must be solved to permit wider application of the two metals.

The world ore supply, of which the United States consumes 85 percent, is ordinarily adequate to satisfy demand; but, during World War II and the Korean War and the ensuing defense buildup, demand for columbium and tantalum in the United States was much greater than the world supply. Consequently, manufacturers looked for substitutes, engineers designed away from these metals, and research investigators ceased to look for new applications. Now that columbium and tantalum are available for nondefense uses, industry has the problem of changing the views of manufacturers, engineers, and research investigators so that they will again consider using columbium and tantalum.

The United States imports 99 percent of the columbite and tantalite that it consumes. This dependence presents no special hardship except during war, when sea lanes are usually difficult to keep open. In a future war the United States might be cut off from African, Malayan, and South American supplies. Anticipating this possibility, the Government is stockpiling columbite and tantalite ores; but, because of the difficulties of predicting the requirements, a stockpile might not provide all the columbium and tantalum needed. Another problem is, therefore, to find ways to decrease the almost complete dependence of the United States on imports and stockpiling.

Many thousands of tons of columbium are present in low-grade deposits, such as the columbium-bearing titanium deposits of Arkansas and the pyrochlore-type mineral deposits of Canada and Africa. These low-grade deposits present a challenge to develop new ore-dressing procedures, for making a satisfactory concentration of the valuable minerals and developing economic separation and extraction processes to treat the concentrates.

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COPPER

By

Helena M. Meyer[1]

A HIGH STANDARD of living and national security depend upon adequate supplies of the red metal—copper. With the onset of World War II, sharply expanded consumption caused the United States, principal copper producing and consuming country in the world, to become dependent in part on foreign metal, a dependence expected to continue and possibly increase in the foreseeable future.

Summary

Copper is the most important nonferrous metal, both in quantity and value of world output; among all metals it is surpassed only by iron. Copper-bearing bronze was the first alloy in wide use by man. So extensively was it substituted for stone in making weapons, tools, and utensils that an early period has come to be known as the bronze age. Copper, essential for large-scale electrification, also made possible "the electric age."

The widespread use of copper stems from its excellent conductivity of electricity and heat, its resistance to corrosion, and its properties of ductility, malleability, and strength. Nearly half is used by the electrical industry in virtually pure metallic form and most of the remainder in the manufacture of brass, bronze, and other alloys, for ultimate use in buildings, automobiles, ships, and other applications. During times of mobilization and war, copper's most important use is in ammunition and in communications, although there are many other vital uses. Steel has replaced copper for some ammunition purposes, and aluminum is acceptable for some electrical uses. On the whole, however, copper is expected to be consumed in increasing quantities in future, following much the same growth pattern as in the past.

The United States is the world's largest producer and consumer of copper, producing nearly one-third of the total and consuming nearly one-half of it. For many years the United States produced much more copper than it consumed and exported the surplus. Production, however, reached at least a temporary peak of about 1,000,000 tons in 1929; but consumption continued to grow, and by World War II a condition of dependence on imports to meet part of total requirements developed. The return of the United States to a position of self-sufficiency is not likely under foreseeable conditions.

The largest copper-producing companies in the United States are vertically integrated, having mining, smelting, refining, fabricating, and marketing interests. The Kennecott Copper Corp., Phelps Dodge Corp., and Anaconda Copper Mining Co. usually mine over three-fourths of the total United States output and with the American Smelting & Refining Co., smelt and refine a substantial part of the primary materials in the United States. Some of the largest fabricators are affiliated with the foregoing four large companies.

Most of the United States production comes from ores lower grade than are

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[1] Assistant chief, Branch of Base Metals, Bureau of Mines.

worked elsewhere. The large size of domestic deposits, the fact that three-quarters of the total comes from open-pit operations, and the high degree of mechanization in mining, milling, and other stages of processing and manufacturing make it possible to utilize low-grade material at a profit. United States smelting, refining, and fabricating facilities have excess capacity, which is used to treat foreign materials.

Foreign production, like that in the United States, comes chiefly from large production units. Chile, which usually ranks next to the United States in production, has only 3 important mines; Northern Rhodesia outranked Chile, temporarily at least, in 1953, with its output coming from 4 mines; 70 percent of Canada's comes from 3; and Belgian Congo's from a number of mines operated as a unit. Of these countries that rank next to the United States in mine output, only in Canada is there important consumption. Canada still exports over half of its output, although the proportion available for export is dropping as consumption advances. Efforts are being made in a number of countries of the world to become independent of United States or any other country's smelting, refining and fabricating facilities.

Because United States production is inadequate for all needs, copper is designated as one of the strategic materials to be stockpiled for United States protection in case of war.

After the Defense Minerals Administration was established, under provisions of the Defense Production Act of 1950, copper was placed high on the list to receive production expansion assistance. Together with the National Production Authority, also created under this act, the Defense Production Administration, in its effort to solve supply-requirement imbalances that followed Korean hostilities, reinstituted the Controlled Materials Plan, used effectively in World War II, for steel, copper, and aluminum, the three materials covered by that major war device of the earlier period. The emergency period passed in 1953, but the demand for copper continued large, as industrial activity remained at high levels.

Equipping a new copper mine requires large capital expenditure and considerable time. None of the production-expansion projects begun in 1951 got into production before November 1953, and others were to follow thereafter.

BACKGROUND

HISTORY

The early history of copper marked an important chapter in human development. Through its use man advanced from the stone age to the bronze age; the latter represented a stage of civilization, rather than a definite chronological period.

TABLE 1.—*Salient statistics of the copper industry, 1919–53*

[All figures in short tons, except price and tenor of ore]

Year	Mine production	Average tenor of copper ores (percent)	Refinery production (primary) from—			Imports (refined)[1]	Exports (refined)[1]	Apparent consumption of new copper [2]	Quoted price at New York [3] (cents per pound)	World production (smelter)	Production from scrap as metal and in alloys		
			Domestic materials	Foreign materials	Total						Old scrap	New scrap	Total
1919	606, 167	1. 65	716, 743	168, 341	885, 084	17, 569	219, 080	457, 236	18. 90	1, 095, 696	152, 600	134, 590	287, 190
1920	612, 275	1. 63	591, 212	171, 871	763, 083	54, 372	275, 613	526, 919	17. 50	1, 057, 200	168, 960	143, 500	312, 460
1921	233, 095	1. 70	304, 707	170, 682	475, 389	34, 625	298, 059	305, 494	12. 65	614, 600	131, 990	85, 310	217, 300
1922	482, 292	1. 74	452, 335	175, 423	627, 758	51, 572	326, 333	448, 317	13. 56	952, 400	202, 800	133, 100	335, 900
1923	738, 870	1. 58	732, 083	257, 835	989, 918	80, 356	364, 690	650, 237	14. 61	1, 341, 500	270, 900	140, 000	410, 900
1924	803, 083	1. 59	837, 107	292, 931	1, 130, 038	72, 955	504, 812	677, 371	13. 16	1, 493, 600	266, 200	122, 100	388, 300
1925	839, 059	1. 54	841, 448	260, 839	1, 102, 287	49, 887	484, 033	700, 506	14. 16	1, 546, 500	291, 010	129, 200	420, 210
1926	862, 638	1. 46	865, 649	295, 594	1, 161, 243	85, 283	428, 062	785, 068	13. 93	1, 608, 300	337, 300	142, 500	479, 800
1927	824, 980	1. 41	859, 476	303, 406	1, 162, 882	51, 640	461, 233	711, 480	13. 05	1, 673, 300	339, 400	150, 800	490, 200
1928	904, 898	1. 41	895, 899	347, 905	1, 243, 804	42, 365	474, 737	804, 269	14. 68	1, 880, 500	365, 500	170, 900	536, 400
1929	997, 555	1. 41	991, 366	378, 690	1, 370, 056	67, 007	411, 227	889, 293	18. 23	2, 098, 800	404, 350	222, 200	626, 550
1930	705, 074	1. 43	695, 612	382, 918	1, 078, 530	43, 105	297, 057	632, 509	13. 11	1, 760, 000	342, 200	125, 000	467, 200
1931	528, 875	1. 50	537, 303	213, 418	750, 721	87, 225	202, 698	451, 032	8. 24	1, 536, 000	261, 300	85, 700	347, 000
1932	238, 111	1. 83	222, 539	117, 895	340, 434	83, 897	110, 977	259, 602	5. 67	1, 027, 000	180, 980	67, 200	248, 180
1933	190, 643	2. 11	240, 669	130, 120	370, 789	5, 432	124, 582	339, 350	7. 15	1, 143, 000	260, 300	77, 800	338, 100
1934	237, 401	1. 92	233, 029	212, 331	445, 360	27, 417	262, 366	322, 638	8. 53	1, 448, 000	310, 900	66, 500	377, 400
1935	386, 491	1. 89	338, 321	250, 484	588, 805	18, 071	260, 735	441, 371	8. 76	1, 681, 000	361, 700	87, 200	448, 900
1936	614, 516	1. 54	645, 462	177, 027	822, 489	4, 782	220, 390	656, 179	9. 58	1, 895, 000	382, 700	101, 900	484, 600
1937	841, 998	1. 29	822, 253	244, 561	1, 066, 814	7, 487	295, 064	694, 906	13. 27	2, 585, 000	408, 900	123, 200	532, 100
1938	557, 763	1. 34	552, 574	239, 842	792, 416	1, 802	370, 545	406, 994	10. 10	2, 254, 000	267, 300	92, 500	359, 800
1939	728, 320	1. 25	704, 873	304, 642	1, 009, 515	16, 264	372, 777	714, 873	11. 07	2, 396, 000	286, 900	212, 800	499, 700
1940	878, 086	1. 20	927, 239	386, 317	1, 313, 556	68, 337	356, 431	1, 008, 785	11. 40	2, 734, 000	333, 890	198, 156	532, 046
1941	958, 149	1. 15	975, 408	419, 901	1, 395, 309	346, 994	103, 602	1, 641, 550	11. 87	2, 905, 000	412, 699	313, 697	726, 396
1942	1, 080, 061	1. 09	1, 064, 792	349, 769	1, 414, 561	401, 436	131, 406	1, 608, 000	11. 87	3, 076, 000	427, 122	500, 633	927, 755
1943	1, 090, 818	1. 04	1, 082, 079	297, 184	1, 379, 263	402, 762	175, 859	1, 502, 000	11. 87	3, 038, 000	427, 521	658, 526	1, 086, 047
1944	972, 549	. 99	973, 852	247, 335	1, 221, 187	492, 395	68, 373	1, 504, 000	11. 87	2, 847, 000	456, 710	494, 232	950, 942
1945	772, 894	. 93	775, 738	332, 861	1, 108, 599	531, 367	48, 563	1, 415, 000	11. 87	2, 436, 000	497, 095	509, 421	1, 006, 516
1946	608, 737	. 91	578, 429	300, 233	878, 662	154, 371	52, 629	1, 391, 000	13. 92	2, 067, 000	406, 453	397, 093	803, 546
1947	847, 563	. 90	909, 213	250, 757	1, 159, 970	149, 478	147, 642	1, 286, 000	21. 15	2, 513, 000	503, 376	458, 365	961, 741
1948	834, 813	. 92	860, 022	247, 424	1, 107, 446	249, 124	142, 598	1, 214, 000	22. 20	2, 580, 000	505, 464	467, 324	972, 788
1949	752, 750	. 91	695, 015	232, 912	927, 927	275, 811	137, 827	1, 072, 000	19. 36	2, 600, 000	383, 548	329, 595	713, 143
1950	909, 343	. 89	920, 748	319, 086	1, 239, 834	317, 363	144, 561	1, 447, 000	21. 46	2, 915, 000	485, 211	492, 028	977, 239
1951	928, 330	. 90	951, 559	255, 429	1, 206, 988	238, 972	133, 305	1, 304, 000	24. 37	3, 095, 000	458, 124	474, 158	932, 282
1952	925, 359	. 85	923, 192	254, 504	1, 177, 696	346, 960	174, 135	1, 360, 000	24. 37	3, 115, 000	414, 635	488, 562	903, 197
1953	926, 448	. 85	932, 232	360, 885	1, 293, 117	274, 777	109, 510	1, 435, 000	28. 92	3, 275, 000	429, 388	529, 076	958, 464

[1] Imports and exports may include some refined copper produced from scrap. Categories not wholly comparable from year to year. Copper is also imported in crude form and shows up as refinery production from foreign ore. Exports, on the other hand, take place also in forms beyond the refined stage.

[2] Adjusted for changes in stocks.

[3] American Metal Market price for electrolytic copper in New York; f. o. b. refinery through August 1927, New York refinery equivalent thereafter.

Utilization of copper preceded the bronze age by at least a thousand years, but it was not until the hardening effects of a tenth part of tin was discovered that the resulting bronze grew to such importance as to christen an era. The step from working native copper to smelting copper ores was extremely significant, but when or where it first occurred is unknown. By 3500 B. C. copper ornaments and simple tools and utensils were in common use by peoples of the Near East and adjacent areas. Bronze was discovered and came into common use in the same area by 2200 B. C.

Copper was first produced in the American Colonies in 1709 at Simsbury, Conn. It was not until discovery and operation of the rich ore deposits of the Northern Peninsula of Michigan in the early 1840's, however, that production in the United States exceeded a few hundred tons a year. Thereafter, smelter output from domestic ores increased from 728 tons in 1850 to 30,240 in 1880.

The development of rich gold placers in Montana in the 1860's, followed by silver mining in the seventies, led to intensive prospecting, many copper discoveries, and a new era in the world copper industry. Copper was discovered in this area at the Anaconda mine, Butte, Mont.

Another great copper camp was slowly rising in the Southwest at the same time. The Bisbee operation, acquired by Phelps Dodge Corp. in the 1880's, formed the basis for the great mining operations of Arizona. In Utah, the Bingham Canyon property, first of the larger low-grade, disseminated ore bodies was brought into production, in the first years of the 20th century, by the Utah Copper Co., later absorbed by the Kennecott Copper Corp.

Large-scale electrification in the current era led to its being called "the electric age." In response to demand to fill needs of the electrical industry, other industries, and those related to war, United States copper mine-production was expanded to a record high of 1,100,000 tons in 1942 and 1943 (World War II) and averaged over 900,000 tons in 1950–53.

SIZE AND ORGANIZATION OF COPPER INDUSTRY

Copper is commonly conceded to be the most important nonferrous metal, from both industrial and mobilization viewpoints. In value it ranks first in the nonferrous group and stands second only to iron among all metals. The value of refined copper produced at primary refineries from domestic, foreign, and secondary sources in 1953 was $851,000,000 compared with $124,000,000 for lead and $210,000,000 for zinc. Aluminum produced by primary producers was valued at $494,000,000. The value of pig iron was $3,700,000,000 and of semifinished and finished steel products $10,900,000,000; an inbetween point would be a proper base for comparison with copper, lead, zinc, and aluminum.

The primary copper industry of the United States is composed of approximately 200 firms engaged in producing and selling copper. The largest producers are vertically integrated and have mining, smelting, refining, fabricating, and marketing interests. The principal producing companies, with their 1953 output, were as follows:

Company [1]	Short tons	Percent of total United States
Kennecott Copper Corp	429,000	46
Phelps Dodge Corp	224,000	24
Anaconda Copper Mining Co	74,000	8
Inspiration Consolidated Copper Co. (Anaconda holds 28 percent of issued stock)	40,000	
Miami Copper Co. (including Castle Dome Copper Co., Inc.)	47,000	
Magma Copper Co	25,000	3
Calumet & Hecla, Inc	20,000	2
Total above companies	859,000	93
Total United States	926,000	------------

[1] Individual company figures from Yearbook of the American Bureau of Metal Statistics, 1953.

MINING

There were over 300 active copper-producing mines in the United States in 1953, most of them relatively small. The 25 largest mines produced 98 percent of the total copper. The mines are listed in table 6 (p. 8).

SMELTING

The primary copper-smelting companies in 1953, their approximate capacities in terms of charge (according to the Yearbook of the American Bureau of Metal Statistics), and the percentages of the total represented, are as follows:

Company	Annual capacity, tons of material	Percent of total capacity (charge)
American Smelting & Refining Co	[1] 2,883,000	34
Phelps Dodge Corp. & Phelps Dodge Refining Corp.	2,650,000	32
Anaconda Copper Mining Co	1,000,000	12
Kennecott Copper Corp	840,000	10
International Smelting & Refining Co.[2]	360,000	4
American Metal Co., Ltd	265,000	3
Magma Copper Co	250,000	3
Tennessee Copper Co	70,000	1
Lake smelters:		
Calumet & Hecla, Inc	100,000	1
Quincy Mining Co	12,000	------------
Total	8,430,000	------------

[1] The greater part of the capacity (1,608,000 tons) of the smelter at Garfield, Utah, and of the capacity (300,000 tons) of the smelter at Hayden, Ariz., is used in treating concentrates from the Utah Division and the Ray Division, respectively, of the Kennecott Copper Corp.

[2] Owned by Anaconda.

REFINING

The copper-refining capacity of primary producers in the United States in 1953, according to the American Bureau of Metal Statistics, aggregated about 1,896,000 tons. The copper-refining companies and their approximate percentage of the total are listed in order of magnitude of available facilities.

Company	Annual capacity, tons	Percent of total capacity
American Smelting & Refining Co	[1] 486,000	26
Phelps Dodge Refining Corp	405,000	21
Kennecott Copper Corp	264,000	14
International Smelting & Refining Co.[2]	240,000	13
American Metal Co., Ltd	200,000	10
Anaconda Copper Mining Co	150,000	8
Calumet & Hecla, Inc	100,000	5
Inspiration Consolidated Copper Co.[3]	39,000	2
Quincy Mining Co	12,000	1
Total	1,896,000	------------

[1] Part used for refining copper produced by Kennecott.

[2] Owned by Anaconda.

[3] 28 Percent of stock owned by Anaconda.

About 10 percent of the primary refined copper produced from domestic materials in the United States is recovered by fire refining in

Michigan, New Mexico, and Texas from crude materials produced in Michigan, New Mexico, and Arizona.

FABRICATION

Fabricators are the principal customers of the primary copper producers. It is in the fabricating plants that the bulk of the new copper is put into semifinished forms—wire, rods, extruded and rolled shapes, etc.—which constitute the raw materials for many other industries.

About 30 companies in the United States are generally recognized as important fabricators of raw copper. Many of the largest are owned by or associated with the great copper mining, smelting, and refining companies, giving them integrated operations from the mines to the finished brass and copper products. A list of the fabricating companies affiliated with copper-producing companies follows:

Fabricating companies of principal copper producers:

Fabricating company	Parent company or company having part stock ownership
Chase Brass & Copper Co	Kennecott Copper Corp.
Kennecott Wire & Cable Co	Do.
American Brass Co	Anaconda Copper Mining Co.
Anaconda Wire & Cable Co	Anaconda Copper Mining Co. (owns 70 percent of stock).
Phelps Dodge Copper Products Corp.	Phelps Dodge Corp.
Revere Copper & Brass, Inc	American Smelting & Refining Co. (owns 36 percent of stock).
General Cable Corp	American Smelting & Refining Co. (owns 42 percent of stock).
Wolverine Tube Division	Calumet & Hecla, Inc.
C. G. Hussey & Co	Copper Range Co.
New Haven Copper Co	Tennessee Corp. (parent company of the Tennessee Copper Co.).
Titan Metal Manufacturing Co	Consolidated Coppermines Corp. (owns 64 percent of stock).

The more important independent fabricators not affiliated with the major producers include the following:

Bridgeport Brass Co.
Bristol Brass Corp.
Chicago Extruded Metals.
Lewin Metals Division, Lewin Mathes Co.
Olin Mathieson Chemical Corp.
Mueller Brass Co.
Reading Tube Co.
J. A. Roebling's Sons Corp.
Rome Cable Corp.
Scovill Manufacturing Co.
Triangle Wire & Cable Co., Inc.
Volco Brass & Copper Co.

GEOGRAPHIC DISTRIBUTION OF COPPER INDUSTRY

Copper occurs so widely in nature that almost every country has some copper-ore deposits; 21 countries each mined over 10,000 tons of recoverable copper in 1953, and some 16 other nations reported some output. In spite of this wide distribution, most of the world mine production is made in but a few places. Concentration mills are found almost always at the mines, although some mills receive custom ores from short distances. Smelting facilities are usually within short distances of mines and mills, and absence of such facilities retards development of new areas of production. Smelter products frequently must be shipped long distances for refining. The smelter products are of such high purity that little, if any, saving in transportation costs would result from shipping refined instead of smelted copper to consumption localities. The scrap supply is chiefly in the industrial areas.

RESOURCES (*5*) [2]

About 90 percent of unmined world copper resources is in five regions—south-central Africa, Chile, the western United States, eastern Ontario and southern Quebec in Canada, and Kazakhstan, U. S. S. R. Table 2 lists 12 districts or mines containing 85 percent of the world copper resources. This list includes both developed reserves that are surely economic under present conditions and partly explored semieconomic deposits that are so large they probably will be important for the future. Deposits not known to contain copper reserves in quantities greater than 3 million tons of copper metal have been omitted from the list.

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[2] Italicized numbers in parentheses refer to items in the Bibliography at the end of this chapter.

TABLE 2.—*Twelve districts containing 85 percent of the world's copper resources, 1950* [1]

Deposits	Country	Major ownership	Nationality
"Mine series"	Northern Rhodesia	Selection Trust, Ltd	British.
		Anglo-American Corp., Ltd	Do.
Chuquicamata	Chile	Anaconda Copper Mining Co	United States.
"Mine series"	Belgian Congo	Union Minière du Haut Katanga	Belgian.
Butte, Mont	United States	Anaconda Copper Mining Corp	United States.
Braden (El Teniente)	Chile	Kennecott Copper Corp	Do.
Bingham, Utah	United States	do	Do.
Keweenaw, Mich.[2]	do	Copper Range Co. and Calumet & Hecla, Inc.	Do.
Morenci, Ariz	do	Phelps Dodge Corp	Do.
Sudbury, Canada [3]	Canada	International Nickel Co. of Canada, Ltd.	Canadian.
	do	Falconbridge Nickel Mines, Ltd	Do.
San Manuel, Ariz.[2]	United States	Magma Copper Co. (Newmont Mining Co.)	United States.
Kazakhstan	U. S. S. R	State-owned	U. S. S. R.
Urals region	do	do	Do.

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[1] Bureau of Mines and Geological Survey, Copper: NSRB Materials Survey, 1952.

[2] Large reserves are considered marginal awaiting production tests. (NOTE: Subsequently it has been decided to bring the deposit into production—anticipated in 1956).

[3] Copper is a byproduct of nickel production.

Coverage is raised to 93 percent by the addition of 13 districts, as follows:

United States—Ely-Kimberly, Nev.; Ray, Ariz.; Central, N. Mex.; Ajo, Ariz.; Bisbee, Ariz., Yerington, Nev.; Miami-Inspiration, Ariz.

Mexico—Cananea.

Chile—Potrerillos (Andes Copper Co.) Aquirre-Africana (Santiago), Rio Blanco.

Peru—Toquepala-Quelleveco, Cerro de Pasco.

The remaining 7 percent of the world reserves is distributed among 34 countries.

NOTE.—Since preparation of the report from which the foregoing data were taken further exploration work in three large porphyry deposits in Peru—Toquepala, Quelleveco, and Cuajones—has proved additional substantial reserves which altogether are estimated to approximate 1 billion tons, averaging about 1 percent copper.

PRODUCTION

Mine.—Most of the copper produced in the United States comes from Western States. Tables 4 and 5 give the sources of mine production by States and districts. Table 6 lists the leading copper-producing properties in the United States.

In the United States, for many years after important production was begun (1845), Michigan was by far the leading source of copper. Montana began a long term in the leading position shortly before 1890 but was displaced in 1907 by Arizona, which, except for 1909, has stood first ever since. Production from the beginning of operations through 1953 and in 1942–53 is shown in table 4.

TABLE 3.—*World mine production of copper, by principal countries, 1949–53*

[Short tons]

(Compiled by Pauline Roberts

	1949	1950	1951	1952	1953
North America:					
Canada	263, 000	264, 000	270, 000	258, 000	252, 000
Cuba	19, 000	23, 000	22, 000	20, 000	18, 000
Mexico	63, 000	68, 000	74, 000	65, 000	66, 000
United States	753, 000	909, 000	928, 000	925, 000	926, 000
Total	1, 098, 000	1, 264, 000	1, 294, 000	1, 268, 000	1, 262, 000
South America:					
Chile	409, 000	400, 000	418, 000	446, 000	400, 000
Peru	30, 000	33, 000	36, 000	34, 000	38, 000
Other	6, 000	6, 000	5, 000	5, 000	5, 000
Total	445, 000	439, 000	459, 000	485, 000	443, 000
Europe:					
U. S. S. R. [1] [2] [3]	223, 000	249, 000	283, 000	330, 000	334, 000
Yugoslavia [3]	38, 000	44, 000	35, 000	36, 000	34, 000
Other	76, 000	75, 000	78, 000	90, 000	81, 000
Total [1] [2]	337, 000	368, 000	396, 000	456, 000	449, 000
Asia:					
Japan	36, 000	44, 000	47, 000	59, 000	64, 000
Other	59, 000	65, 000	69, 000	75, 000	70, 000
Total [1] [2]	95, 000	109, 000	116, 000	134, 000	134, 000
Africa:					
Belgian Congo [3]	157, 000	194, 000	212, 000	226, 000	236, 000
Northern Rhodesia	286, 000	328, 000	352, 000	363, 000	411, 000
Union of South Africa	34, 000	37, 000	37, 000	39, 000	39, 000
Other	13, 000	14, 000	15, 000	18, 000	16, 000
Total	490, 000	573, 000	616, 000	646, 000	702, 000
Australia	15, 000	17, 000	19, 000	21, 000	40, 000
World total [1]	2, 480, 000	2, 770, 000	2, 900, 000	3, 010, 000	3, 030, 000

[1] Approximate production.
[2] Output from U. S. S. R. in Asia included with U. S. S. R. in Europe.
[3] Smelter production.

TABLE 4.—*Mine production of recoverable copper, by States, 1942–53 (total), and cumulative production from earliest record to end of 1953*

[Short tons]

State	1942–53	Total production from earliest record to end of 1953
Alaska	97	685, 905
Arizona	4, 440, 061	13, 886, 856
California	39, 540	632, 110
Colorado	26, 168	270, 462
Idaho	25, 346	123, 105
Michigan	354, 778	4, 987, 340
Missouri	29, 352	[1] 37, 832
Montana	965, 108	7, 002, 565
Nevada	678, 507	2, 144, 001
New Mexico	811, 482	1, 828, 926
Oregon	196	12, 419
Pennsylvania	51, 816	(2)
South Dakota	2	106
Tennessee	85, 166	435, 451
Texas	427	1, 383
Utah	3, 046, 806	6, 693, 332
Vermont	29, 516	(2)
Virginia	494	(2)
Washington	62, 285	109, 349
Wyoming	2	16, 327
Undistributed	2, 516	234, 400
Total	10, 649, 665	[3] 39, 101, 869

[1] Small quantity for Wisconsin included with Missouri.
[2] Not available, included in undistributed.
[3] For States east of the Mississippi except Michigan, largely smelter production.

TABLE 5.—*Mine production of copper in the principal districts*[1] *of the United States, 1949–53, in terms of recoverable copper*

[Short tons]

District or region	State	1949	1950	1951	1952	1953
West Mountain (Bingham)	Utah	196, 101	277, 655	270, 183	282, 098	268, 511
Copper Mountain (Morenci)	Arizona	141, 934	154, 689	143, 921	124, 882	123, 789
Globe-Miami	do	80, 189	84, 688	90, 225	93, 079	86, 478
Summit Valley (Butte)	Montana	55, 945	53, 897	56, 826	61, 557	77, 520
Central (including Santa Rita)	New Mexico	[2] 53, 276	63, 694	71, 526	74, 008	69, 869
Ajo	Arizona	58, 350	64, 400	63, 093	63, 808	64, 730
Robinson (Ely)	Nevada	37, 533	52, 087	56, 198	57, 148	60, 557
Mineral Creek (Ray)	Arizona	18, 595	36, 442	50, 580	49, 274	47, 574
Warren (Bisbee)	do	9, 840	13, 345	27, 271	27, 440	29, 344
Pioneer (Superior)	do	21, 616	22, 636	17, 662	17, 716	25, 093
Lake Superior	Michigan	19, 506	25, 608	24, 979	21, 699	24, 097
Eureka (Bagdad)	Arizona	7, 906	10, 673	9, 087	9, 228	10, 072
Ducktown	Tennessee	6, 489	6, 851	7, 069	7, 620	7, 829
Orange County	Vermont	2, 986	3, 504	3, 774	3, 774	3, 947
Chelan Lake	Washington	[3] 5, 249	[3] 4, 904	3, 932	[4] 4, 273	[4] 3, 614
Lebanon (Cornwall mine)	Pennsylvania	3, 974	4, 142	5, 297	3, 485	3, 027
San Juan Mountains	Colorado	1, 974	2, 582	2, 712	3, 157	2, 376
Southeastern Missouri	Missouri	3, 670	2, 982	2, 422	2, 576	2, 374
Coeur d'Alene	Idaho	1, 171	1, 896	1, 874	1, 862	2, 100
Lordsburg	New Mexico	1, 934	2, 061	1, 521	1, 475	1, 988
Cochise	Arizona	689	498	1, 350	1, 838	1, 849
Pima (Sierritas, Papago, Twin Buttes)	do	348	282	334	1, 090	1, 353
Blackbird	Idaho			148	[5] 1, 214	([6])
Verde (Jerome)	Arizona	17, 215	13, 291	9, 742	4, 524	626
Burro Mountain	New Mexico	([2])		([7])		([7])

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[1] Districts producing 1,000 short tons or more in any year of the period 1949–53.

[2] Burro Mountain included with Central to avoid disclosing individual company operations.

[3] Includes Peshastin Creek and Wenatchee to avoid disclosing individual company operations.

[4] Includes Ferry to avoid disclosing individual company operations.

[5] Includes Spring Mountain and Texas to avoid disclosing individual company operations.

[6] Figures withheld to avoid disclosing individual company operations.

[7] Less than 0.5 ton.

TABLE 6.—*Twenty-five leading copper-producing mines in the United States in 1953, in order of output*

Rank	Mine	District	State	Operator	Source of copper
1	Utah Copper	West Mountain (Bingham)	Utah	Kennecott Copper Corp	Copper ore.
2	Morenci	Copper Mountain (Morenci)	Arizona	Phelps Dodge Corp	Do.
3	Butte Mines	Summit Valley (Butte)	Montana	Anaconda Copper Mining Co	Copper, lead-zinc ores.
4	Chino	Central	New Mexico	Kennecott Copper Corp	Copper ore.
5	New Cornelia	Ajo	Arizona	Phelps Dodge Corp	Do.
6	Ray	Mineral Creek (Ray)	do	Kennecott Copper Corp	Do.
7	Inspiration	Globe-Miami	do	Inspiration Consolidated Copper Co.	Do.
8	Ruth Pit	Robinson (Ely)	Nevada	Kennecott Copper Corp	Do.
9	Copper Queen	Warren (Bisbee)	Arizona	Phelps Dodge Corp	Copper, lead-zinc ores.
10	Miami	Globe-Miami	do	Miami Copper Co	Copper ore.
11	Magma	Pioneer (Superior)	do	Magma Copper Co	Do.
12	Castle Dome	Globe-Miami	do	Castle Dome Copper Co., Inc	Do.
13	Calumet & Hecla, Inc	Lake Superior	Michigan	Calumet & Hecla, Inc	Copper ore and tailings.
14	Morris Brooks Pit	Robinson (Ely)	Nevada	Consolidated Coppermines Corp.	Copper ore.
15	Bagdad	Eureka (Bagdad)	Arizona	Bagdad Copper Corp	Do.
16	Ruth Pit Extension	Robinson (Ely)	Nevada	Consolidated Coppermines Corp.	Do.
17	Kimbley Pit	do	do	Kennecott Copper Corp	Do.
18	Burra Burra, Calloway, Mary, Eureka, Boyd.	Polk County	Tennessee	Tennessee Copper Co	Copper-zinc ore.
19	Elizabeth	Orange County	Vermont	Vermont Copper Co., Inc	Copper ore.
20	Holden	Chelan Lake	Washington	Howe Sound Co	Copper-zinc ore.
21	Cornwall	Lebanon County	Pennsylvania	Bethlehem Steel Co	Magnetite-pyrite-chalcopyrite ore.
22	Champion	Lake Superior	Michigan	Copper Range Co	Copper ore.
23	Quincy	do	do	Quincy Mining Co	Copper-ore tailings.
24	Republic & Mammoth	Cochise	Arizona	Coronado Copper & Zinc Co	Copper-zinc ore.
25	Treasury Tunnel-Black Bear	Upper San Miguel	Colorado	Idarado Mining Co	Do.

Smelting and Refining Areas.—The Bureau of Mines is not at liberty to publish smelter and refinery production by States because to do so would reveal individual company information. A list of copper smelter plants in the United States is given in table 8.

TABLE 7.—*World smelter production of copper, by countries, 1949–53*

[Short tons]

(Compiled by Pauline Roberts)

Country	1949	1950	1951	1952	1953
North America:					
Canada	226, 000	238, 000	245, 000	196, 000	236, 000
Mexico	54, 000	53, 000	65, 000	56, 000	57, 000
United States [1]	860, 000	1, 009, 000	1, 037, 000	1, 025, 000	1, 048, 000
Total	1, 140, 000	1, 300, 000	1, 347, 000	1, 277, 000	1, 341, 000
South America:					
Chile	386, 000	381, 000	397, 000	422, 000	372, 000
Peru	23, 000	26, 000	27, 000	23, 000	26, 000
Total	409, 000	407, 000	424, 000	445, 000	398, 000
Europe:					
Germany, West [2]	160, 000	223, 000	235, 000	215, 000	241, 000
U. S. S. R. [3] [4]	223, 000	249, 000	283, 000	330, 000	334, 000
Yugoslavia	38, 000	44, 000	35, 000	36, 000	34, 000
Other	72, 000	65, 000	75, 000	80, 000	92, 000
Total [3] [4] [5]	493, 000	581, 000	628, 000	661, 000	701, 000
Asia:					
Japan	42, 000	41, 000	48, 000	54, 000	70, 000
Other	25, 000	28, 000	36, 000	40, 000	43, 000
Total [3] [4]	67, 000	69, 000	84, 000	94, 000	113, 000
Africa:					
Belgian Congo	157, 000	194, 000	212, 000	226, 000	236, 000
Northern Rhodesia	290, 000	308, 000	346, 000	350, 000	406, 000
Union of South Africa	33, 000	37, 000	36, 000	38, 000	39, 000
Other	------------	2, 000	1, 000	2, 000	1, 000
Total	480, 000	541, 000	595, 000	616, 000	682, 000
Australia	11, 000	17, 000	17, 000	22, 000	40, 000
World total [3]	2, 600, 000	2, 915, 000	3, 095, 000	3, 115, 000	3, 275, 000

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[1] Smelter output from domestic and foreign ores, exclusive of scrap Production from domestic ores only, exclusive of scrap, was as follows: 1949, 758,000; 1950, 911,000; 1951, 931,000; 1952, 927,000; and 1953, 943,000.

[2] Includes scrap.

[3] Approximate production.

[4] Output from U. S. S. R. in Asia included with U. S. S. R. in Europe.

[5] Belgium reports a large output of refined copper which is believed to be produced principally from crude copper from Belgian Congo and is not given here, as that would duplicate output reported under the latter country

TABLE 8.—*Copper smelters in the United States*

[Plants that treat primary crude materials exclusively or chiefly]

Location	Company
Arizona:	
Ajo	Phelps Dodge Corp., 40 Wall St., New York 5, N. Y.
Morenci	Do.
Douglas	Do.
Hayden	American Smelting & Refining Co., 120 Broadway, New York 5, N. Y.
Miami	International Smelting & Refining Co., 25 Broadway, New York 4, N. Y.
Superior	Magma Copper Co., Superior, Ariz.
Montana: Anaconda	Anaconda Copper Mining Co., 25 Broadway, New York 4, N. Y.
Nevada: McGill	Kennecott Copper Corp., 161 E. 42d St., New York 17, N. Y.
New Jersey: Carteret	American Metal Co., Ltd., 61 Broadway, New York 6, N. Y.
New Mexico: Hurley	Kennecott Copper Corp., 161 E. 42d St., New York 17, N. Y.
New York: Laurel Hill	Phelps Dodge Refining Corp., 40 Wall St., New York 5, N. Y.
Tennessee: Copperhill	Tennessee Copper Co., 61 Broadway, New York 6, N. Y.
Texas: El Paso	American Smelting & Refining Co., 120 Broadway, New York 5, N. Y.
Utah: Garfield	Do.
Washington: Tacoma	Do.

Slightly over half of the United States electrolytic refining capacity is on the Atlantic seaboard in metropolitan New York and New Jersey and Baltimore, Md. Cheap power, so important in electrolytic refining, and large nearby markets, together with ocean transportation, have combined to produce this concentration.

Table 9 gives North American refinery capacity by countries, and table 10 lists foreign refineries outside of North America.

Fabrication Areas.—Major copper-fabricating countries, except the United States and the U. S. S. R., are not important producers of copper ore. The other large consuming countries are United Kingdom, France, and Germany. Canada is becoming an increasingly large user of copper and resembles the United States and U. S. S. R. in that it is also a large producer. Unlike the latter two countries, however, Canada exports as much copper as it consumes.

TABLE 9.—*Annual capacity (in short tons) of primary refineries in the United States, Canada, and Mexico, in 1953* [1]

	Electrolytic	Lake	Fire refined
United States:			
American Metal Co., Ltd., Carteret, N. J.	144, 000		56, 000
American Smelting & Refining Co.:			
Baltimore, Md.	198, 000		
Perth Amboy, N. J.	168, 000		
Tacoma, Wash.	120, 000		
Anaconda Copper Mining Co., Great Falls, Mont.	150, 000		
Calumet & Hecla, Inc., Hubbell, Mich.		100, 000	
Inspiration Consolidated Copper Co., Inspiration, Ariz.	39, 000		
International Smelting & Refining Co., Perth Amboy, N. J.	240, 000		
Kennecott Copper Corp.:			
Hurley, N. Mex.			72, 000
Garfield, Utah	192, 000		
Phelps Dodge Refining Corp.:			
Laurel Hill, N. Y.	140, 000		
El Paso, Tex.	240, 000		25, 000
Quincy Mining Co., Hancock, Mich.		12, 000	
	1, 631, 000	112, 000	153, 000
Canada:			
Canadian Copper Refiners, Ltd., Montreal, East, Quebec	140, 000		
International Nickel Co. of Canada, Ltd., Copper Cliff, Ontario	168, 000		
	308, 000		
Mexico: Cobre de Mexico, S. A., Atzcapotzalco, D. F.	43, 000		

Casting capacity	United States	Canada	Mexico
1. Electrolytic (including scrap)	1, 739, 000	308, 000	43, 000
2. Lake	112, 000		
3. Fire refined (in addition to capacity reported under item 1)	153, 000		
Total	2, 004, 000	308, 000	43, 000

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[1] Taken from American Bureau of Metal Statistics, 1953 Yearbook.

TABLE 10.—*Foreign copper refineries, exclusive of Canada and Mexico* [1]

Company	Location of plant
South America:	
Andes Copper Mining Co.[2]	Potrerillos, Chile.
Braden Copper Co.[3]	Calentos, Chile.
Chile Exploration Co.[2]	Chuquicamata, Chile.
Cerro de Pasco Corp	Oroya, Peru.
Europe:[4]	
Soc. Gen. Mét. de Hoboken [5]	Oolen, Belgium.
Cie. Gén. d'Électrolyse du Palais	Le Palais, France.
Outokumpu Oy	Pori, Finland.
Kupferwerk Ilsenburg A. G.[5]	Ilsenburg (Harz), Germany (East).
Hüttenwerke Kayser A. G.[5]	Lunen, Germany.
Hochofenwerk Lubeck A. G	Herrenwyk, Germany.
Mansfeldscher Kupferschieferbergbau A. G.[5].	Hettstedt, Germany (East).
Unterharzer Berg-u. Hüttenwerke.	Okser, Germany.
Norddeutsche Affinerie [5]	Hamburg, Germany.
Zinnwerke Wilhelmsburg	Do.
Actid, Ltd	Blantyre, Glasgow, Scotland.
Birmingham Battery & Metal Co., Ltd.[3]	Birmingham, England.
Thos. Bolton & Sons, Ltd.[3]	Widnes, Lancs., and Froghall, North Staffs., England.
British Copper Refiners, Ltd.[3]	Prescot, Lancs., England.
Enfield Copper Refining Co., Ltd.[3]	Brimsdown, England.
Elkington & Co., Ltd.[5]	Walsall., Staffs., England.
McKechnie Bros., Ltd	Widnes, Lancs., England.
I. C. I. Metals, Ltd.[3]	Birmingham and Swansea, England.
Murex, Ltd.[3]	Rainham, Essex, England.
Falconbridge Nikkelverk A/S	Kristiansand, Norway.
Röros Kobberverk [3]	Röros, Norway.
Soc. Española de Construcciones Electro-Mecanicás and Subsidiaries.[5]	Cordoba, Spain.
Sociedad Industrial Asturiana [5]	Lugones, Spain.
Reymersholms Gamla Industri [3]	Hälsingborg, Sweden.
Bolidens Gruv Aktiebolag	Rönnskar, Sweden.
Rudnici Bakra i Topionice	Bor, Yugoslavia.
Africa:	
Union Miniére du Haut Katanga [2]	Jadotville-Shituru, Belgian Congo.
Do [3]	Lubumbashi, Belgian Congo.
Messina (Transvaal) Dev. Co.[3]	Messina, Transvaal.
Rhodesia Copper Refineries, Ltd	Nkana, Rhodesia.
Mufulira Copper Mines, Ltd	Mufulira, Northern Rhodesia.
Asia:[4]	
Indian Copper Corp.[3]	Moubhandar, Chota, Nagpur, India.
Sumitomo Metal Mng. Co., Ltd	Niihama, Ehime-ken, Japan.
Dowa Mng. Co., Ltd	Kosaka, Akita-ken, Japan.
Furukawa Denki Kogyo K. K	Nikko, Tochigi-ken, Japan.
Mitsui Mng. & Smelting Co., Ltd.	Takehara, Hiroshima-ken, Miike, Omuda, Fukuoka-ken, Japan.
Nihon Mng. Co., Ltd	Hitachi, Ibaragi-ken, Saganoseki, Oita-ken, Japan.
Mitsubishi Metal Mining Co., Ltd.	Osaka, Japan.
Ergani Bakir Isletmesi Mudurlugu.[3]	Ergani-Maden, Turkey.
Murgul Bakir Isletmesi [3]	Murgul, Turkey.
Australia:	
Mount Lyell Mining & Railway Co.	Mount Lyell, Tasmania.
Electrolytic Refining & Smelting Co.[5]	Port Kembla, N. S. W.

[1] Taken from 1953 Yearbook of American Bureau of Metal Statistics. Electrolytic except as otherwise stated; so far as possible plants treating only secondary material have been excluded.

[2] Direct from ore.

[3] Fire refining.

[4] Copper refineries in Russia are omitted owing to absence of accurate data. In China electrolytic refineries at Chungking and Kunming (Yunnan) are reported to be producing over 2,000 tons of copper annually.

[5] Does fire as well as electrolytic refining.

Satisfactory data covering consumption in major United States areas of fabrication are not available. An analysis of figures for November 1954 indicates the following rough breakdown:

Estimated geographic distribution of consumption of refined copper in the United States

	Percent of total
Connecticut and Rhode Island	14
Pennsylvania, Delaware, and Maryland	7
New York and New Jersey	46
Michigan and Ohio	10
Indiana and Illinois	14
Other	9
	100

New Production and Expansion Projects.—*United States.*—Increases in copper production as a result of various types of assistance permitted by the Defense Production Act; that is, Government loans, Government purchase contracts and tax-amortization benefits, or combinations of the three, and expected increases at later dates are discussed in the following paragraphs. Some expansion now in progress is independent of these Government aids. A review of copper-production expansion projects submitted to the Government, several of which are now in production and others in course of construction, indicates that new mining, milling, and smelting capacity in the United States now costs $1,500 to $2,000 per ton of annual copper-production capacity. Costs abroad are lower, and under favorable circumstances (that is, where milling or smelting facilities already exist), production can be expanded at much lower unit costs than in the United States. Nevertheless, it is significant that a 10-percent expansion in domestic capacity would cost on the order of $200,000,000. It should be noted also, in this connection, that major copper projects require 3 to 4 years for completion.

The following paragraphs summarize the major programs for production expansion:

The "Greater Butte project" of the Anaconda Copper Mining Co., Montana, announced in 1947, was designed to bring into production low-grade ore above the 3,400-foot level in the Butte Hill mines. It was to cost about $27,000,000, make available over 130,000,000 tons of ore (averaging 1 percent copper), and augment normal Butte production 45,000 tons annually. Improvements to make this output possible include construction at Anaconda of a leach-precipitation-float plant, a sponge-iron plant for treating the oxidized portion of the ore, storage bins, and expanded concentrator capacity. Ore production of 12,000 tons a day, at the end of 1953, was expected to rise to 15,000 tons a day in 1954.

The Yerington mine, Anaconda Copper Mining Co., Lyon County, Nev., began to produce in November 1953; it was the first major copper property to begin production as a result of Defense Production Act stimulation. The oxide ore body, containing an estimated reserve of 35,000,000 tons averaging 0.97 percent copper, is mined by open-pit methods and the copper recovered by leaching. An estimated reserve of at least 15,000,000 tons of sulfide ore of similar grade underlies the oxide ore body. Development and equipment of the property, including reopening the Leviathan mine, Alpine County, Calif., as a source of sulfur for the sulfuric acid needed for the operation, were expected to require about $38,000,000; about $35,000,000 had been spent to the end of 1953. An annual rate of 33,000 tons of copper was expected. Copper precipitates are shipped to Anaconda, Mont., for smelting.

The American Smelting & Refining Co. brought the Silver Bell unit, 40 miles northwest of Tucson, Ariz., into production in April 1954. Production was expected to be 18,000 tons annually. Ore reserves were estimated at 32,000,000 tons, assaying 0.9 percent copper, in 2 separate ore bodies, which can be mined by open-pit methods. Total expenditures for bringing the property into production were expected to reach $18,000,000.

The Phelps Dodge Corp. started production in July 1954 at the Bisbee East (Lavender pit) ore body, Warren (Bisbee) district, Arizona. Production was expected to be at an annual rate of about 38,000 tons of copper and to continue for 11 or 12 years. The cost was estimated at $25,000,000.

The Copper Cities Mining Co., a wholly owned subsidiary of the Miami Copper Co., began production at the Copper Cities property, Globe-Miami district, Gila County, Ariz., in August 1954. Movement of the mill at the Castle Dome mine, also owned by Miami, to the Copper Cities mine began in December 1953 coincidental with exhaustion of the former mine.

The White Pine Copper Co., wholly owned subsidiary of the Copper Range Co., plans to bring the White Pine mine, Lake Superior district, Michigan, into production late in 1954. The annual target was 36,000 tons of copper. Reserves were estimated at 309,660,000 tons of ore averaging 21.3 pounds of copper per ton. Milling and smelting facilities are included in the plans for this project. A Government loan of $57,185,000 was authorized for this property.

The Kennecott Copper Corp. is preparing to mine by underground methods an undeveloped ore body near the company open-pit operation at Ruth, Robinson (Ely) district, White Pine County, Nev. A very small margin of profit was expected, based on the 1951 price-cost relationship. This area was expected to be in production in March 1955.

The San Manuel Copper Corp., Pinal County, Ariz., wholly owned subsidiary of the Magma Copper Co., plans eventually to produce 70,000 tons of copper and 6,000,000 pounds of molybdenum annually at its mine now under development. Reserves are more than 460,000,000 tons, averaging 0.782 percent copper. Magma obtained a $94,000,000 loan from the Reconstruction Finance Corp., part of a requested loan of $111,288,000, to develop and equip the property, having already invested $10,000,000 in the property. The remainder of the funds required were to be obtained privately. A mill and smelter are included in the plans. Production was to begin in 1956.

Canada.—The program of the International Nickel Co. of Canada, Ltd., to increase underground production to an eventual 13,000,000 tons to offset declines from open-pit operations approaching exhaustion is making noteworthy progress, and 11,000,000 tons of underground ore was produced in 1953. The goal of 13,000,000 tons equals 76 percent of the total copper ore produced by underground methods in the United States in 1953. International Nickel Co. reserves at the end of 1953 were 262,000,000 tons of ore containing 7,817,000 tons of nickel and copper.

The Sherridon operation of Sherritt Gordon Mines, Ltd., Manitoba Province, was worked out in 1951; and the mining plants, concentrator, and employees' houses were being moved to the Lynn Lake operation, where nickel and copper are to be produced. A new chemical metallurgical refinery, one of the first of its kind, is being constructed; pilot-plant operations were carried on in 1953. Reserves are estimated at 14,100,000 tons averaging 1.223 percent nickel and 0.618 percent copper.

The Campbell Chibougamau Mines, Ltd., is developing its property on Merrill Island in Dore Lake, Quebec, and plans to start production in the first half of 1955. It has a purchase contract with the United States Government.

The Hudson Bay Mining & Smelting Co., Ltd., is developing a deposit containing nickel-copper-cobalt ore in the Yukon Territory not far from the Alaska border. Other claims in the area have been optioned to other companies. The mountainous nature of the terrain and the relative remoteness of the area may impede development of producing properties there.

At the Gaspé property, Quebec, of Noranda Mines, Ltd., the mine and mill were expected to be in operation late in 1954 and the smelter early in 1955. Ore reserves are estimated at 67 million tons of 1.3-percent copper ore.

Chile.—At the Chuquicamata mine, Antofagasta Province, the oxide ore body has formed the basis of operations since they were begun in 1915. Production from many millions of tons of oxide and mixed ores will continue for years. In 1947 the Chile Exploration Co. began to construct a new metallurgical plant for treating the sulfide ores underlying the oxide ore body, which had been encountered in the pit. The new plant (consisting of a sulfide concentrator and a smelter) began to produce in 1952. The capacity of new plant and existing plant should exceed 250,000 tons annually compared with a range of yearly output of 172,000 to 229,000 tons in 1948–53, inclusive.

Peru.—The American Smelting & Refining Co. has carried on extensive drilling programs at the Toquepala and Quellaveco porphyry-type copper deposits, 20 miles apart, in southern Peru. Both deposits were found to contain substantial tonnages of ore susceptible to open-pit mining. At the Toquepala property 400,000,000 tons averaging slightly over 1 percent copper and at the Quellaveco 200,000,000 tons averaging slightly less than 1 percent have been proved.

The Export-Import Bank announced in November 1954 that it was prepared to extend a credit of $100,000,000, out of an anticipated total of $205,000,000, to assist in financing the United States dollar costs of bringing the Toquepala property into production. Thirty thousand tons of ore was to be mined and milled per day and up to 140,000 tons of blister copper produced annually.

Belgian Congo.—An extensive development program is reported in process at properties of the Union Minière du Haut Katanga, and the company is also engaged in plant extensions and new installations. The company produced a record tonnage of 236,000 short tons of copper in 1953. No data are available on the prospective increase above this tonnage, but a considerable increase is expected.

Northern Rhodesia.—New companies were to exploit Northern Rhodesian mineral deposits further. The Chibuluma Mines, Ltd., a subsidiary of Mufulira Copper Mines, Ltd., received a United States Government loan equivalent to about $14,000,000 to develop the Chibuluma mine. Annual production of 18,000 tons of copper and 500,000 pounds of cobalt was anticipated, and the United States was to be repaid in shipments of copper and cobalt to the National Stockpile. Production was expected to begin in 1955. Estimated ore reserves were 7,300,000 tons averaging 5.23 percent copper and 0.25 percent cobalt. The Bancroft Mines, Ltd., was to develop and equip its mine to produce about 4,000 tons of copper monthly, at an estimated cost of £12,000,000. Production was expected to start in 1957. Ore reserves were 80,000,000 tons averaging 3.6 percent copper.

Uganda.—The Kilembe Mines, Ltd., was developing a copper deposit in the foothills of the Ruwenzori Mountains near Kilembe. Proved reserves were estimated as 15,000,000 tons averaging 1.44 to 2.09 percent copper and 0.18 to 0.19 percent cobalt.

Yugoslavia.—Consideration was being given to development of Majdanpek mine in eastern Serbia. This deposit was said to contain over 100,000,000 tons of ore, averaging 0.85 percent copper, which is susceptible to mining by open-pit methods.

TECHNOLOGY

GEOLOGY

Copper ores occur in rocks of nearly all kinds and ages and in ore deposits of many types. The principal copper minerals are native copper, chalcopyrite ($CuFeS_2$), chalcocite (Cu_2S), bornite (Cu_3FeS_3), enargite ($3Cu_2S.As_2S_5$), cuprite (Cu_2O) and the hydrous copper carbonates, malachite and azurite. Copper deposits can be classified geologically into a multitude of types, but for present purposes may be grouped into two classes—disseminated "porphyry" and all others. The disseminated deposits are large masses of more or less altered and decomposed igneous rock in which copper is uniformly though sparsely distributed in the form of small particles and veinlets; the rock usually is of a granitic type and porphyritic texture, whence the term "porphyry copper." Other types of deposits include bedded deposits, vein deposits, massive replacement deposits, etc. The practical importance of the porphyry type is that its size and physical character encourage application of large-scale, low-cost mining and metallurgical methods.

MINING

Copper is mined by both underground and open-pit methods; in Michigan an appreciable production is obtained by dredging old tailings from a lake for re-treatment. Underground almost all the common and some uncommon stoping methods are used as circumstances require; block caving is the most important in the United States. In the open pits, waste and ore alike usually are removed in steplike benches, 40 to 60 feet high, and loaded by electric power shovels into trucks or standard-gage cars on an electrified rail-haulage system. If the overburden : ore ratio is too high for surface mining, porphyry deposits usually are mined by block caving.

Copper mining in the United States and Canada is well mechanized, and great tonnages of ore are moved at low cost with a minimum of labor. Wherever the physical characteristics of the ore bodies prohibit adoption of such methods, the average grade of the ore must be higher if mining is to be profitable.

Table 1 (p. 3) shows how the average grade of copper ore mined in the United States has decreased since 1919. It should be observed that the declining grade of ore mined does not solely reflect exhaustion of richer reserves but is partly accounted for by the ability to exploit lower grade material owing to technologic progress; it is perhaps due also to price increases and in part to the fact that a large and growing fraction of the domestic production comes from low-cost open pits instead of underground mines in which costs (and grade-percentage cutoffs) are necessarily higher.

MILLING

The typical copper-milling flowsheet comprises crushing, grinding, flotation, thickening, and filtering. Flotation is, and has been for 25 years, the mainstay of the treatment of copper ore. Modifications and exceptions are numerous: Gravity concentration in jigs and on tables; acid leaching with production of "cement" or electrolytically precipitated metal; ammonia leaching with the end product copper oxide; and byproduct-molybdenite recovery. The typical copper concentrate in the United States contains 25 to 30 percent copper and appreciable gold, silver and minor metals like selenium.

LEACHING

Leaching is not competitive with flotation where sulfide ores are concerned, but ores that are predominantly oxide and are associated with little or no acid-consuming gangue and oxidized ores that are present in acid-consuming gangue are more amenable to acid or ammonia leaching than to flotation. Leaching has been used for the oxide ores at such properties as the Chuquicamata in Chile and for mixed sulfide-oxide ores at properties such as Inspiration in Arizona; it is still used at these two and is also a supplementary operation at numerous other properties.

At such operations as Chuquicamata and Inspiration, the ores mined are leached directly in the leaching plant, and subsequent precipitation is by electrolysis. At many open-pit and some underground mines, the copper in "waste" dumps or in specially segregated dumps of leaching ore is recovered by heap leaching. The water applied to the dump, from whatever source—mine drainage, deep well, or surface runoff—may be acidified with sulfuric acid or may pick up ferric sulfate from the oxidation products of the ore minerals and thus become capable of dissolving copper from the ore. The copper-bearing solution is passed over scrap iron, whereupon the iron in the scrap replaces the copper in the solution, and the latter metal is precipitated as a fine "cement" that is washed, drained, and shipped to the smelter. Considerable copper is thus produced very cheaply.

As mine workings deepen and near-surface oxide minerals are replaced by the sulfides, flotation is required for economic operation. At Chuquicamata the ore occurs in three general subdivisions—oxide, sulfide and a mixture of both. Only oxide was treated from 1915 to 1952. To supplement its production of copper from oxide ores the company began to construct a new sulfide plant in 1947, and the current construction phase was completed in 1953.

Leaching in place in a mine, consisting of percolating leaching solutions through unmined ores, may be considered a mining operation. The leaching solutions usually comprise mine-drainage water but may be acidified surface water. The copper is precipitated on scrap iron.

SMELTING AND REFINING

Copper blast furnaces for producing copper matte became nearly obsolete when, with general adoption of flotation, fine-size copper sulfide concentrate had to be smelted. Most copper ores and concentrates are now smelted, with or without roasting, in reverberatory furnaces. Copper matte and slag are produced. The liquid-copper matte (a copper-iron sulfide containing the precious metals and some impurities) is treated in a copper converter, where siliceous flux is added and air is blown through the molten bath. The iron sulfide is oxidized by the air blast to ferrous oxide, which unites with the silica, producing a ferrous silicate slag, which is poured off. The copper sulfide reacts with oxygen to yield metallic copper and sulfur dioxide. Some volatile impurities are also eliminated in the converter as gaseous products and some are slagged off; others remain with the copper, which at this stage is called "blister" copper. The name "blister" comes from the rough surface caused by the escaping gases of the cooling copper castings. Blister copper cannot be used as such, as it still is too impure. It may be transferred from the converter to small holding or casting furnaces, where it can be cast into blister cakes or anodes for shipment to the refinery; otherwise, it may be transferred molten to reverberatory furnaces for fire refining before casting into commercial intermediate shapes, such as cakes, billets, or ingots. Most copper enters consumption channels in electrolytically refined form for electrical and other uses.

The anodes are taken to the electrolytic tankhouse, where they are suspended with thin sheets of copper (starting sheets) placed alternately between in a solution comprised mainly of dilute sulfuric acid and copper sulfate. The starting sheets act as cathodes. An electric current is passed through the tank, causing the anode to dissolve and the copper to build up in pure form on the cathode; the insoluble impurities, such as gold, silver, etc., sink to the bottom of the tank as a sludge or slime or adhere to the undissolved anodes, which are later washed free. Some cathodes are cut to specified sizes and sold to consumers in this form, but most of the electrolytic copper is cast into other shapes—that is, wire bars, billets, cakes, ingots, and ingot bars.

SECONDARY PRODUCTION

Copper recovered from copper scrap, copper-base alloy scrap, and other copper-bearing scrap materials as metal, as copper alloys without separation of the copper, or as copper compounds is known as secondary copper. Secondary copper is produced from new and from old scrap, the two general classes of scrap copper. "New scrap" is defined as waste produced during the manufacture of articles for ultimate consumption, including defective finished or semifinished articles that must be reworked. Typical examples of new scrap are defective castings, clippings, punchings, turnings, borings, skimmings, drosses, and slag. From a long-range viewpoint recovery from new scrap is not regarded as part of the total supply of copper; however, for short-range purposes, including allocation of copper by the United States Department of Commerce in emergency periods, new scrap generated by durable-goods manufacturers is considered part of the total supply. Bureau of Mines statistics on copper recovered from new scrap cover only purchased new scrap and not "home scrap" (scrap generated in a milling or foundry plant and consumed in the same plant). If the same type of material is sold it is "purchased scrap." "Old scrap" consists of metal articles that have been discarded after serving a useful purpose. Such articles may be worn out, obsolete, or damaged. Items of old scrap include discarded trolley wire, radiators, fired cartridge cases, used pipe, and lithographers' plates. These articles were subtracted from the available supply when first manufactured and therefore are an addition to it when they return in wornout condition for reprocessing.

Secondary copper is also classified by the form of recovery, that is, as unalloyed copper and as alloyed copper. Only the unalloyed copper is part of the total supply of refined copper. This section, however, discusses total secondary production, alloyed and unalloyed, and both old and new scrap.

Secondary-copper production increased markedly during the 29-year period, 1925-53. From an average yearly recovery of 511,000 tons in 1925–29, output rose to 900,000 tons per year in 1948–52 and totaled 958,000 tons in 1953. Alloyed copper, principally brass and bronze, provided virtually the entire increase. Recovery in this form averaged 87 percent of the total in 1941–45 compared to 56 percent yearly for 1925–29. In the postwar years, 1946–49, alloyed copper comprised 72 percent and unalloyed copper 28 percent of the total recovery of scrap; in 1953 the percentages were 75 and 25, respectively.

Table 1 (p. 221) shows secondary production from new and from old scrap.

It has become a generally accepted policy in defense agencies and others that old scrap only should be taken into account in long-term studies involving supplies and requirements for copper. As already stated, for short periods such as times of emergency when supplies are allocated, new scrap is considered part of the total supply by the United States Department of Commerce. Old scrap is largely collected by several thousand scrap dealers, who, for the most part, are not affiliated with the great primary producers.

The channels through which much of the reclaimed copper returns from the scrap dealers and fabricators to use are the secondary refiners, the brass and bronze ingotmakers, and the brass mills.

The bulk of the secondary copper in the United States is recovered in ingot brass and bronze and other alloys; recovery of scrap as unalloyed copper represents less than one-third of the total annual output.

RESERVES

The supply of copper known to remain in the ground in the United States is estimated at about 25,000,000 tons of recoverable metal. The present major producing mines have 20 to 30 years of life at the production rates of 1940–50 (*5*). Beyond this, exploration campaigns of the past decade around old mines and in new areas indicate an additional 10 to 15 years' supply. If intensive exploration is continued, it is expected that this favorable position will be maintained. An important feature of the copper-reserve situation is that copper is a large-scale, mass-production resource. About 95 percent of the reserves occur in less than a dozen mines or mining areas.

Much public attention is given to the importance of small mines, but it is a plain fact that the bulk of copper comes from a few big operations. The myriad little veins, lodes, and pods scattered throughout the country have relatively little importance as potential producers of large tonnages of war-emergency copper.

The United States has more large copper mines than any other country, but the accuracy of the knowledge of the reserves in individual deposits is extremely varied. Physical measurement of ore deposits is only one part of the problem of determining economic reserves. A further difficulty is estimating the economic basis. Not only do the commercial reserve totals fluctuate with changes in the price of copper, but there is another feature that is often overlooked. Inflationary cycles can raise the cost of production and wipe out otherwise economic reserves. Mining companies are acutely aware of the fluctuating profit-or-loss features of their reserves and tend to minimize them in order to be conservative.

Ultimate copper resources are an important aspect for long-range planning, but reserves are normally considered in terms of economically profitable material. The United States reserve position in the light of current profitability ranks first, at present, among the three major copper regions of the world but may ultimately rank third. The main reason is the relatively lean quality of domestic ore. The uptrending scale of wages, not mentioned by the author of the report on reserves (*5*), is an important additional factor.

The African deposits are notable for huge reserves of 3- to 6-percent ore, whereas the United States average is less than 1 percent. The Chilean deposits have huge tonnages averaging 2 percent copper. Russian grades appear no better than those of the United States.

Currently African producers have transportation difficulties and shortages of fuel, power, and skilled labor. In South America the labor problem is intensified by the vagaries of local government policies. The threat of extreme taxation and even confiscation has deterred investments in Latin America. Political events promise to be a major factor in determining the competitive position of the world's major copper producers.

Three-fourths of American production now comes from large-scale, low-cost, open-pit deposits, in contrast to the predominance of the underground mines in South America and Africa. However, American open-pit operations face eventual termination because of increasingly adverse ratios of waste rock that must be removed to uncover copper ore. At least under current conditions, the life of American open-pit operations is limited largely to 10 to 20 years, with 30 years the exception. The life of the property can be extended if the grade of ore permits a changeover from open-pit to underground mining.

About 95 percent of the United States reserves occurs in a dozen mining areas. These deposits, including both economic and semieconomic materials, are listed in table 11.

Addition of the following mines and districts brings the coverage to 98 percent: Magma, Ariz.; Silver Bell, Ariz.; Cornwall, Pa.; Bagdad, Ariz.; Tyrone, N. Mex.; and Glacier Peak, Wash. The remaining 2 percent of reserves is distributed among some 200 present and former copper mines and districts of the United States.

TABLE 11.—*Twelve mining districts containing 95 percent of the United States copper reserves, 1950*

District:	*Principal ownership*
Butte, Mont	Anaconda Copper Mining Co.
Bingham, Utah	Kennecott Copper Corp.
Keweenaw, Mich	Copper Range Co. & Calumet & Hecla, Inc.
Morenci, Ariz	Phelps Dodge Corp.
San Manuel, Ariz	Magma Copper Co. (Newmont Mining Co.)
Ely-Kimberly, Nev	Kennecott Copper Corp. and Consolidated Coppermines Corp.
Central, N. Mex	Kennecott Copper Corp.
Ray, Ariz	Do.
Ajo, Ariz	Phelps Dodge Corp.
Yerington, Nev	Anaconda Copper Mining Co.
Miami, Ariz [1]	Miami Copper Co. and Inspiration Consolidated Copper Co.
Bisbee, Ariz	Phelps Dodge Corp.

[1] Includes Castle Dome, Copper Cities (Sleeping Beauty), Inspiration, Miami, and Globe.

The districts that contain 85 percent of the world's copper resources are listed under Geographic Distribution of Copper Industry.

Most observers agree that world reserves exceed 100,000,000 short tons of recoverable metal, with the prospect that, when conditions for production in foreign countries improve (that is, adequate power, transportation, and other facilities are supplied and political, labor, and other problems alleviated or solved), reserves may be double the quantity given. The possibilities for advancement in mining and metallurgical techniques are great, and the quantities of copper that ultimately may be made available probably will differ sharply from any of the many previous estimates.

CONSUMPTION, USES, AND SUBSTITUTES

CONSUMPTION AND USES

Copper is widely used because it has superlative electrical conductivity, high heat conductivity, and good corrosion resistance and because it is strong, yet malleable and ductile and can be alloyed readily with many other metals to acquire or impart desirable characteristics. Nearly half of it is consumed in virtually pure metallic form, primarily by the electrical industry in the manufacture of generators, motors, electric locomotives, switchboards, telephone and telegraph equipment, light and power lines, and other items. Most of the remainder is employed by the alloys industry in manufacturing brass, bronze, and other copper alloys that are used ultimately in building construction, automobile manufacture, ammunition, shipbuilding, and a wide variety of other purposes. During a war, copper's chief use is in munitions, although it is employed widely in vehicle production, airplane manufacture, naval and other ship construction, and signal equipment and ordnance manufacture.

The consumption of refined copper, by broad industry classifications, in 1949–53 is shown in table 12.

Estimates on the consumption of copper by end uses were made by an industry representative for the President's Materials Policy Commission and were quoted in its report. These figures are given in table 13.

TABLE 12.—*Refined copper consumed in 1949–53 by types of consumers*

[Short tons]

	1949	1950	1951	1952	1953
Brass mills	478, 126	675, 100	650, 967	675, 073	689, 477
Wire mills	623, 713	713, 354	710, 199	739, 487	753, 029
Secondary smelters	4, 463	6, 209	13, 744	22, 918	15, 305
Chemical plants	1, 576	3, 122	3, 223	3, 719	3, 849
Foundries and miscellaneous	21, 808	26, 649	38, 732	38, 535	32, 555
Total	1, 129, 686	1, 424, 434	1, 416, 865	1, 479, 732	1, 494, 215

TABLE 13.—*Consumption of copper, by uses, in the United States in 1950*

Use	Short tons	Percent
Electrical	303, 000	22. 0
Telephone and telegraph	87, 000	6. 0
Light and power	100, 000	7. 0
Wire cloth (screens)	10, 000	. 7
Other rod and wire	127, 000	9. 0
Ammunition (cartridge cases and rotating bands)	70, 000	5. 0
Automobiles (radiators, 47–70 lb. per car)	145, 000	10. 0
Building (plumbing, flashing, and leaders only)	135, 000	10. 0
Clocks and watches	7, 000	. 5
Copper-bearing steel	8, 000	. 5
Heating radiators (convectors)	9, 000	. 6
Radio and television	50, 000	3. 5
Railroads	25, 000	2. 0
Refrigerators	26, 000	2. 0
Ships (propellers, etc.)	22, 000	1. 6
Air conditioning	30, 000	2. 1
Exports	151, 000	10. 7
Other	95, 000	6. 8
Total	1, 400, 000	100. 0

Data on refined copper contained in shipments of copper-base mill and foundry products, by industry groups, in the fourth quarter of 1951 through the second quarter of 1953, were compiled by the Business and Defense Services Administration and published (*22*) in 1954. The figures are summarized in table 14.

TABLE 14.—*Shipments of copper-base mill and foundry products, in the fourth quarter of 1951 through the second quarter of 1953, by industry groups*

[Million pounds refined-copper content]

Industry	1951		1952		1953	
	October-December	Percent of total	January–December	Percent of total	January–June	Percent of total
Direct military	79	11. 7	392	14. 2	177	11. 4
Construction:						
Defense Electric Power Administration	42	6. 2	171	6. 2	88	5. 7
Communications	34	5. 0	127	4. 6	71	4. 6
Other	68	10. 1	301	10. 9	162	10. 5
Total construction	144	21. 3	599	21. 7	321	20. 8
Building materials	18	2. 7	74	2. 7	55	3. 6
Communications equipment	10	1. 5	40	1. 4	25	1. 6
Consumer durable goods	18	2. 7	80	2. 9	59	3. 8
Electrical equipment	106	15. 7	391	14. 1	219	14. 2
Electronics	13	1. 9	54	2. 0	37	2. 4
Engines and turbines	9	1. 3	36	1. 3	19	1. 2
General components	33	4. 9	125	4. 5	73	4. 7
General industrial equipment	18	2. 7	69	2. 5	47	3. 0
Metalworking equipment	10	1. 5	42	1. 5	19	1. 2
Motor vehicles	63	9. 3	244	8. 8	159	10. 3
Railroad equipment	12	1. 7	44	1. 6	24	1. 6
Scientific and technical equipment	11	1. 6	46	1. 7	29	1. 9
Other industries	29	4. 3	79	2. 9	46	3. 0
Maintenance, repairs, operations	56	8. 3	232	8. 4	141	9. 1
Other deliveries	47	6. 9	215	7. 8	96	6. 2
Total	676	100. 0	2, 762	100. 0	1, 546	100. 0

SUBSTITUTES

The copper industry faces competition from several metals that have made inroads into the copper-consumption pattern in the post-World War II period. Price is only one of several factors that contribute to a trend toward partial substitution in copper. The sudden increase in demand since June 1950, arising out of a defense program superimposed on high levels of industrial activity, and the comparatively inflexible short-run supply of copper caused prices to rise. Production could not be expanded rapidly, and adequate copper was not available even at relatively high prices; many major consumers were impelled to search, at least temporarily, for substitutes. The electrical industry particularly, chief copper-consuming industry, looked to aluminum. Aluminum wire with a steel core (ACSR) has displaced copper for long-distance transmission lines, and this change appears permanent. Other typical substitutions of aluminum for copper are fractional horsepower motor windings and lightbulb bases. Further, the cold-war economy gave the military establishment time to change specifications and design. An example is an accelerated shift to the use of steel for brass in shell cases. Research work indicates that titanium can become an important substitute for copper when the cost is reduced through application of production technology so that its price is low enough to come into a competitive range.

Aluminum and stainless steels have also somewhat reduced the use of copper in the building industry. The plastics have not been a strong competitor. Technology has furthered substitution for copper by the introduction of the numerous clad metals, including copper-clad, which use proportionately less metal than would solid copper. Printed electrical circuits have also come into the field as a substitute for copper wire.

A study of the use of copper and aluminum in the first quarter of 1953 compared with an average quarter of 1947, for selected product groups, was made by the Business and Defense Services Administration. Some of the major groups selected were found to have shown the following changes: Electrical equipment for internal-combustion engines, copper rose 39 percent and aluminum 205 percent; radio and radio products, copper rose 133 percent and aluminum 246; motor vehicles and parts, copper rose 18 percent and aluminum 82; metal stampings, copper dropped 42 percent while aluminum rose 57; telephone and telegraph equipment, copper rose 61 percent and aluminum 97, and the total for all groups selected, copper rose 11 percent and aluminum 95 percent.

Adjustment to the new competing materials does not involve much more than a shift in the overall consumption pattern. It will make more copper available for those uses for which it is peculiarly fitted and for which no substitute is practicable. A concentrated effort to introduce successfully an automobile radiator of a material other than copper has not proved successful. The heating, air-conditioning, and refrigeration industries do not expect substitution for copper in their fields, where copper tubing is used extensively and is unexcelled.

SOURCES AND ADEQUACY OF STATISTICAL INFORMATION

The major continuing series of Government statistics on copper are as follows:

Bureau of Mines

Monthly series

PRODUCTION
1. From mines.
2. From primary smelters and refineries (domestic and foreign primary crude materials, and scrap).
3. Copper sulfate.
4. From secondary smelters and brass mills as unalloyed and alloyed copper, showing that from old and new scrap.
5. Copper in chemicals from scrap at secondary plants.

CONSUMPTION
1. Refined copper by major industry classes.
2. Scrap, by types, from all plants.

STOCKS
1. Primary smelters and refineries finished products.
2. Scrap at secondary smelters and brass mills, by types.
3. Scrap at primary plants by grades.
4. Copper sulfate.

Annual series

PRODUCTION
1. Refinery shapes at primary plants.
2. From scrap at foundries and chemical plants showing that from old and new scrap.
3. Byproduct sulfuric acid at copper plants.

CONSUMPTION
1. Consumption of brass and bronze ingot by foundries.
2. Consumption of scrap at foundries, miscellaneous manufacturers, and chemical plants.

STOCKS
1. Scrap at foundries, miscellaneous manufacturers and chemical plants.

Sales of copper by selling agencies.

U. S. Department of Commerce

FOREIGN TRADE
1. Imports of copper by countries and classes.
2. Exports of copper by classes.
3. Exports of refined copper by country of destination.
4. Imports of copper sulfate by countries.
5. Exports of copper sulfate by country of destination.
6. Imports of copper and brass scrap.
7. Exports of copper and brass scrap.

DOMESTIC

Copper Industry Quarterly Report.

Shipments of Copper-Base Mill and Foundry Products.

The Copper Institute, the U. S. Copper Association, and the American Bureau of Metal

Statistics, all at 50 Broadway, New York, N. Y., compile monthly figures on copper production and shipments in the United States, some monthly data on foreign production, monthly totals on consumption in the United States and outside thereof, and monthly stocks of refined copper. The American Bureau of Metal Statistics publishes production for the larger mines or mining companies on an annual basis, world production and consumption by countries, plant capacities, and other items, some of which are not statistical.

There are no continuing data on consumption of copper by end uses and none showing consumption by geographic location, although geographic data would be available regularly in Bureau of Mines reports if a few companies now reporting on a company basis would give plant figures instead. Figures showing sources of scrap on a geographic basis likewise are lacking, as are adequate data on reserves.

FOREIGN TRADE[3]

United States foreign trade in copper has consisted chiefly of importing crude materials, such as ores, concentrates, matte, and blister and of exporting refined copper, fabricated copper and fabricated-copper products, and manufactured goods containing copper. For many years United States smelting and refining capacity has been excessive for the treatment of domestic materials, and this excess capacity has encouraged the importation of foreign crude materials for custom treatment. Through this means greater quantities of foreign copper were available for commerce, and the United States smelting and refining industry also benefited; however, foreign producers have made great progress in building their own reduction plants and are continuing to become independent of this United States service.

Imports of copper into the United States by countries and by classes in 1953 and by countries in 1949–53 are shown in tables 15 and 16.

Most copper exported is in advanced forms of manufacture in which the copper content is indeterminate. In recent years the copper exported in refined form and in primary fabricated shapes has been largely from imported crude materials refined or fabricated here. Because supplies of copper in the United States have been inadequate to fill requirements in most of the period since the end of World War II, copper has been subject to export controls. Exports are given in tables 17 and 18.

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[3] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

TABLE 15.—*Copper (unmanufactured) imported into the United States, in 1953, by countries*

[Short tons]

(General imports: U. S. Department of Commerce)

Country of origin	Ore (copper content)	Concentrates (copper content)	Regulus, black or coarse copper (copper content)	Unrefined black blister and converter in pigs or converter bars	Refined in ingots, plates, or bars	Old and clippings for remanufacture	Total
Australia	6	1, 038		9, 414	2, 543	40	13, 041
Belgian Congo					5, 799		5, 799
Belgium-Luxembourg					5, 540	75	5, 615
Bolivia	792	3, 163	17				3, 972
Canada	102	31, 373	1, 768	3, 494	67, 487	3, 223	107. 447
Chile	3, 234	12, 529		117, 520	147, 394	499	281, 176
Cuba	123	17, 634				449	18, 206
France						2, 160	2, 160
Germany, West					3, 570		3, 570
Malta, Gozo, and Cyprus		3, 680					3, 680
Mexico	357	8, 646	4, 310	44, 982	7, 513	10	65, 818
Northern Rhodesia				85, 264	2, 778		88, 042
Norway					4, 368	59	4, 427
Peru	746	8, 752	865		16, 157	3	26, 523
Philippines	(1)	[1] 13, 815					13, 815
Sweden				550	1, 603	64	2, 217
Turkey				11, 894			11, 894
Union of South Africa	1, 404	6, 008		166	666		8, 244
United Kingdom			56	326	1, 396	416	2, 194
Yugoslavia					7, 775		7, 775
Other countries	233	235	3		188	795	1, 454
Total	6, 997	106, 873	7, 019	273, 610	274, 777	7, 793	677, 069

[1] Some copper in "Ore" and "Other" from Republic of the Philippines is not separately classified and is included with "Concentrates."

TABLE 16.—*Copper (unmanufactured) imported into the United States, 1949–53, by countries*

[Short tons]

(General imports: U. S. Department of Commerce)

Country	1949	1950	1951	1952	1953
Australia	941	1, 307	1, 143	684	13, 041
Belgian Congo	----------	103	----------	(1)	5, 799
Belgium-Luxembourg	273	[2] 474	----------	646	5, 615
Bolivia	4, 671	5, 220	4, 449	3, 097	3, 972
Canada (including Newfoundland and Labrador)	82, 821	82, 365	54, 554	[2] 81, 932	107, 447
Chile	285, 386	292, 215	268, 359	362, 303	281, 176
Cuba	15, 849	22, 891	22, 302	19, 934	18, 206
Ecuador	812	640	----------	----------	----------
France	[2] 158	3, 801	1, 587	1, 806	2, 160
Germany	----------	44	----------	[3] 8, 932	[3] 3, 570
Japan	1, 167	54, 400	1, 908	223	----------
Malta, Gozo, and Cyprus	6, 888	6, 530	5, 556	5, 441	3, 680
Mexico	64, 706	62, 748	47, 878	50, 997	65, 818
Netherlands	234	352	47	41	175
Northern Rhodesia	[4] 27, 244	[4] 87, 300	43, 717	28, 225	88, 042
Norway	[2] 37	4, 098	----------	1	4, 427
Peru	22, 316	28, 502	10, 054	11, 317	26, 523
Philippines	7, 969	10, 129	12, 608	14, 787	13, 815
Sweden	----------	57	----------	----------	2, 217
Turkey	4, 572	3, 266	----------	3, 779	11, 894
Union of South Africa	8, 919	9, 859	7, 353	8, 588	8, 244
United Kingdom	1, 925	940	6	37	2, 194
Yugoslavia	14, 727	10, 998	6, 223	14, 833	7, 775
Other countries	[2] 1, 094	[2] 2, 150	[2] 1, 391	1, 277	1, 279
Total	552, 709	690, 389	489, 135	[2] 618, 880	677, 069

[1] Less than 1 ton.

[2] Revised figure.

[3] West Germany.

[4] Tonnages credited to Southern Rhodesia by the U. S. Department of Commerce have been added to Northern Rhodesia, inasmuch as copper of the grades reported does not originate currently in Southern Rhodesia.

TABLE 17.—*Refined copper exported from the United States, by country of destination, 1949–53*

[Short tons]

(U. S. Department of Commerce)

Country	1949	1950	1951	1952	1953
Algeria	1, 727	1, 174	76	446	----------
Argentina	1, 871	110	1, 276	----------	4, 353
Australia	----------	----------	650	166	100
Austria	2, 481	192	655	1, 356	286
Belgium-Luxembourg	1, 404	578	562	----------	560
Brazil	3, 198	1, 356	3, 621	5, 496	9, 635
Canada	50	94	37	12, 884	763
Denmark	831	1, 982	1, 372	1, 447	917
France	23, 948	18, 401	18, 626	35, 573	17, 834
Germany, West	10, 600	3, 417	10, 273	20, 447	12, 035
Greece	241	309	637	9	84
India	20, 514	8, 989	217	6, 243	1, 830
Italy	19, 914	16, 640	7, 948	17, 040	10, 971
Japan	----------	----------	----------	365	2, 350
Mexico	6	2	28	51	35
Netherlands	11, 611	6, 148	8, 190	5, 994	11, 362
Norway	495	3, 217	1, 911	1, 674	3, 266
Portugal	----------	112	----------	771	----------
Sweden	2, 239	2, 015	593	2, 242	----------
Switzerland	9, 374	5, 152	5, 415	9, 562	6, 365
Union of South Africa	3	----------	560	----------	8
United Kingdom	26, 236	74, 245	70, 161	48, 116	22, 367
Uruguay	22	124	81	95	176
Other countries	1, 062	304	416	4, 158	4, 213
Total	137, 827	144, 561	133, 305	174, 135	109, 510

TABLE 18.—*Exports of copper from the United States by classes, 1949–53*

[Short tons]

(U. S. Department of Commerce)

	Ore, concentrates etc. (copper content)	Refined (bars, ingots, etc.)	Old and scrap	Pipes and tubes	Plates, sheets, and strips	Rods	Wire (bare) [1]	Wire and cable insulated
1949	200	137, 827	8, 284	3, 344	1, 088	12, 678	7, 881	24, 888
1950	616	144, 561	9, 445	1, 988	581	10, 073	7, 009	18, 682
1951	234	133, 305	7, 701	2, 160	572	521	7, 983	14, 032
1952	648	174, 135	8, 941	2, 591	553	1, 937	7, 163	17, 070
1953	495	109, 510	34, 568	1, 622	367	321	9, 313	15, 622

[1] Due to changes in classifications, data for 1952–53 not strictly comparable to earlier years.

The sources of imported crude materials and destinations of exported refined copper and primary fabrications of copper are shown under Foreign Trade. From 1949–53, 23 percent of refined and unrefined copper imported came from Canada and Mexico, and 56 percent came by sea from other Western Hemisphere sources. Africa supplied 11 percent of the United States receipts during this period, but such copper was largely for reexportation.

STRATEGIC CONSIDERATIONS

The United States now depends on imports of copper to help fill all needs for this metal both in time of war and while experiencing a healthy peacetime economy. Evidence points to the conclusion that, in future, only under abnormal and not-to-be-desired conditions (such as a major depression) are domestic mines likely to fill total domestic demand and then only if prices remained high in relation to costs. The shift of the United States from a net exporting (exports exceeding imports) to a net importing (imports exceeding exports) nation did not take place until World War II, although the excess of United States supplies over requirements had been declining for some time. This growing dependence resulted more from the sharp expansion in United States consumption than from exhaustion of domestic reserves and was met in part by increased imports from Africa and Canada, but principally from Chile. United States production and apparent consumption of new copper are shown in table 19. The table indicates a smaller than actual export surplus for early years and a greater dependence than actually exists in later years. This condition results from counting copper taken into United States fabricating plants as consumption, whereas much copper, not measurable, leaves the country in manufactured form.

TABLE 19.—*Mine production of copper, apparent consumption of new copper and excess consumption over production, 1920–29, 1930–39, 1940–49, 1950–53*

[Short tons]

Period	Mine production	Apparent consumption [1] (new)	Excess consumption over production
1920–29	7, 298, 745	6, 498, 954	[2] 799, 791
1930–39	5, 029, 192	4, 919, 454	[2] 109, 738
1940–49	8, 796, 420	13, 642, 335	4, 845, 915
1950	909, 343	1, 447, 000	537, 657
1951	928, 330	1, 304, 000	375, 670
1952	925, 359	1, 360, 000	434, 641
1953	926, 448	1, 435, 000	508, 552

[1] Includes copper, if any, delivered by industry to the National Stockpile.

[2] Production excess.

Of the supplies received from Chile (49 percent of total United States imports), virtually all was from properties controlled by the Anaconda Copper Mining Co. and the Kennecott Copper Corp. (United States companies). The greatest part of Chilean revenue comes from copper mining, and the three mines that supply almost all of Chile's exports are owned by these United States concerns.

Western Hemisphere production of copper is more than adequate to take care of Western Hemisphere requirements; an excess of more than 150,000 tons is estimated for 1953.

Dependence of the United States on imported metal for a substantial part of requirements has led to the establishment of a large National Strategic Stockpile objective for copper. The stockpile objective was still incomplete at the end of 1954.

BUREAU OF MINES EXPLORATION PROGRAM

The Strategic Minerals Development Program was inaugurated as a result of the passage of the Strategic Materials Act of 1939 (Public Law 177, 76th Congress, June 7, 1939). Copper was not on the original list of scarce metals for investigation in 1939, as the United States was a copper-exporting nation at that time. It became apparent in 1940 that the United States would be unable to fill all of its expanded requirements by a substantial margin, and by 1942 a number of authorities said that copper had become the most critical metal of all. Bureau of Mines investigations thus began to place more stress on copper. In Bureau of Mines Report of Investigations 4647, a résumé of the results of mineral resources projects through the fiscal year 1949, Lowell B. Moon said:

> Of the 36 copper deposits where the Bureau's work has developed ore, 35 are of moderate size and moderate grade and the other is a large low-grade porphyry deposit. Fifteen of the 35 moderate-size deposits have produced some copper since the Bureau's projects were finished.
>
> A mining company has continued the development started by the Bureau on the porphyry-copper deposit—San Manuel in Arizona. Although the Bureau's work indicated 110,000,000 tons of 0.77-percent copper, the additional drilling done by the company has indicated an additional 363,000,000 tons of the same grade, making a total of 473,000,000 tons containing on the order of 3 million tons of recoverable copper. The company is sinking two shafts as the beginning of an underground development program. Production is expected to begin in about another 5 years. The deposit will be able to produce an important part of the country's copper in a future emergency.

The Bureau continued mineral-resources investigations after 1949, but on a reduced scale, and when this type of work was expanded because of defense needs, the Defense Minerals Administration was charged with the responsibility of carrying on the work, with the Bureau of Mines and the Geological Survey in advisory capacities. Further discussion is included under Defense Program.

DEFENSE PROGRAM

The Defense Production Act, empowering the President to regulate the economy to assure adequate supplies of materials for expanded defense and essential civilian requirements, became law in September 1950. Among the provisions that applied particularly to the copper industry were those that called for encouragement of expansion in supplies and curtailment in unessential consumption. The Defense Minerals Administration and its successors—the Defense Materials Procurement Agency and the Defense Minerals Exploration Administration—which are charged with encouraging production of strategic materials and with searching for new sources of production were the agencies concerned most directly with mineral production. Copper was given a high priority to receive various types of production expansion assistance—production loans, Government purchase contracts assuring a market at specified prices with escalation provisions, and certificates of necessity permitting accelerated amortization for income-tax purposes. The larger projects are discussed under Geographic Distribution of the Industry—Production. Properties that fulfilled certain specifications were granted exploration loans amounting to 50 percent of estimated costs, to be repaid from eventual production.

Soon after the National Production Authority was established late in 1950, it issued orders restricting excessive copper and copper-base-alloy inventory accumulations, reducing consumption, establishing rules directed at regulating the flow of these materials, as well as others, directing the flow of scrap, and limiting toll agreements. In 1951 the Defense Production Administration reinstituted the Controlled Materials Plan, used effectively in World War II, for distributing fabricated products made from copper, steel, and aluminum, the three materials covered by that major war device of the earlier period. Copper raw materials were placed under complete allocation control, effective August 1, 1951.

Efforts to assure equitable supplies for all "Free World" nations resulted in the placing of copper under international allocation. Quotas for the Free World were established by the International Materials Conference, beginning with the fourth quarter of 1951. The member countries voluntarily accepted restrictions upon quantities to be consumed and agreed not to dispose of copper to countries in the Soviet bloc.

Another defense measure was the placing of ceiling price restrictions on copper and other commodities in January 1951. The effects on copper are described under Prices and Costs.

Early in 1953 the situation had eased so that United States price controls and national and international allocation of copper were abandoned. Military and Atomic Energy Commission needs were still to receive preferential treatment, however.

PRICES AND COSTS

United States price quotations are usually in cents per pound for electrolytic copper delivered Connecticut Valley or the equivalent f. o. b. New York refinery and cover the ordinary forms of wire bars and ingot bars. According to the E&MJ Metal and Mineral Markets, the premiums on special shapes, in cents per pound, are: Standard ingots, 0.125; slabs, 0.375 and up; cakes, 0.425 and up, depending on weight and dimensions; and billets 1.35 and up, depending on dimensions and quality. Discounts granted on cathodes are 0.125 to 0.15 cent per pound. Early in July 1950 the Kennecott Copper Corp. announced that it would dispose of its copper on a uniform delivered price basis throughout the United States, marking a noteworthy change in the long-established practice of the copper industry. The delivered basis has since been generally adopted by the industry, but not until nearly 4 years later.

Prices for copper in 1920 to 1953, inclusive, are shown in table 20. Quoted and weighted prices are given, as well as weighted prices adjusted by the wholesale index of the Bureau of Labor Statistics.

TABLE 20.—*Average yearly quoted prices of electrolytic copper and average weighted prices of refined copper delivered in the United States (including prices adjusted by the wholesale index), 1920–53*

[Cents per pound]

Year	Electrolytic domestic f. o. b. refinery [1]	Electrolytic domestic f. o. b. refinery [2]	Electrolytic export f. o. b. refinery [2]	Refined copper	
				Weighted, f. o. b. refinery [3]	Adjusted by wholesale index [4]
1920	17.50	17.456	(5)	18.4	18.3
1921	12.65	12.502	(5)	12.9	20.3
1922	13.56	13.382	(5)	13.5	21.5
1923	14.61	14.421	(5)	14.7	22.5
1924	13.16	13.024	(5)	13.1	20.5
1925	14.16	14,042	(5)	14.2	21.1
1926	13.93	13.795	(5)	14.0	21.5
1927	13.05	12.920	(5)	13.1	21.1
1928	14.68	14.570	(5)	14.4	22.9
1929	18.23	18.107	(5)	17.6	28.4
1930	13.11	12.982	(5)	13.0	23.2
1931	8.24	8.116	(5)	9.1	19.2
1932	5.67	5.555	(5)	6.3	15.0
1933	7.15	7.025	6.713	6.4	14.9
1934	8.53	8.428	7.271	8.0	16.4
1935	8.76	8.649	7.538	8.3	16.0
1936	9.58	9.474	9.230	9.2	17.5
1937	13.27	13.167	13.018	12.1	21.6
1938	10.10	10.000	9.695	9.8	19.2
1939	11.07	10.965	10.727	10.4	20.7
1940	11.40	11.296	10.770	11.3	22.1
1941	11.87	11.797	10.901	11.8	20.8
1942	11.87	11.775	11.684	11.8	18.4
1943	11.87	11.775	11.770	11.8	17.6
1944	11.87	11.775	11.700	11.8	17.4
1945	11.87	11.775	11.700	11.8	17.1
1946	13.92	13.820	14.791	14.4	18.3
1947	21.15	20.958	21.624	20.9	21.7
1948	22.20	22.038	22.348	21.7	20.8
1949	19.36	19.202	19.421	19.7	19.8
1950	21.46	21.235	21.549	20.8	20.2
1951	24.37	24.200	26.258	24.2	21.1
1952	24.37	24.200	31.746	24.2	21.7
1953	28.92	28.798	30.845	28.7	26.1

[1] American Metal Market.
[2] E&MJ Metal and Mineral Markets.
[3] Reported by copper selling agencies.
[4] Weighted price divided by Bureau of Labor Statistics wholesale price index (1947–49=100).
[5] Not available. Export quotation established after imposition of tariff in 1932.

The price for copper in 1928 was the highest for any year since 1920, 1929 exceeded 1920, and in 1930 continued high until April. There is little doubt that one effect of these high prices was acceleration of the development of copper mines throughout the world, particularly in Africa and Canada. When the price collapsed it dropped to the lowest average for all time, or to 5.7 cents a pound for all of 1932. The average price indicated for the entire production for 87 years (1845–1931) was 16.1 cents a pound and for the 30 years (1902–31) 16.3 cents.

Prices continued low, almost without exception, until after World War II. In World War II Government controls held copper at 12 cents a pound delivered Connecticut Valley (11.87 f. o. b. refinery equivalent). Under the Premium Price Plan the Government granted premiums for copper produced from high-cost mines and from high-cost sections of other mines. Quotas were assigned by the Office of

Production Management and the Office of Price Administration, and only production over assigned quotas was eligible for premiums. During the period of premium prices (February 1942–June 1947), when demand for copper was enormous, the average price, including all premiums, was only 14.3 cents a pound.

The quoted price jumped from 11.87 cents a pound, f. o. b. refinery, before price control was ended on November 10, 1946, to 19.37 cents by the end of the year.

The postwar period was one of continuing high demand, except for part of 1949, when an industrial recession was underway. Improvement was noted in the second quarter of 1950, and there was a noteworthy upsurge of prices following the onset of the Korean emergency in mid-1950; the quotation for electrolytic copper was 24.37 cents a pound at the end of the year.

Ceiling prices for copper were established by the General Ceiling Price Regulation issued by the Economic Stabilization Agency and effective January 26, 1951. This order set maximum prices at not to exceed the highest prices received by individual producers between December 19, 1950, and January 25, 1951, inclusive. Most primary producers were selling electrolytic copper, delivered Connecticut Valley, at 24½ cents a pound, and that price became the ceiling for those producers. Some other companies, representing a relatively small aggregate tonnage, sold for substantially higher prices during the period selected, and these higher prices became their ceilings.

Foreign countries, like the United States, had mobilization plans calling for increased quantities of copper, and world consumption in 1951 was rising. Efforts of all countries to obtain supplies of metal led to increasing world prices and to speculation, causing further price increases. Rumors had it that prices up to 60 cents a pound were paid for copper on the European continent, while United States prices were frozen substantially at 24½ cents.

The situation led to an agreement between the United States and the Chilean Government in May 1951 providing for payment of an additional 3 cents over the United States ceiling for Chilean copper sold in the United States. A little later the Office of Price Stabilization permitted all copper refined in the United States from imported materials to be paid for on basis of 27½ cents a pound.

Another factor that tended to confuse the price situation was action of the Defense Materials Production Agency to avoid loss of production from mines that faced closing because of rising costs. Maintenance-of-production contracts based on production costs of the mines involved were entered into with these companies; thus additional sets of prices were established.

Prices in foreign countries continued above those for foreign copper sold in the United States (27½ cents), and Chile became dissatisfied with the foregoing arrangement. The outcome was abrogation, in May 1952, of its agreement with the United States.

On May 21, 1952, the Office of Defense Mobilization authorized importers to pay higher prices for imported copper and to pass 80 percent of increased costs on to users. Early in June it was decided that the increases were to be calculated using 24½ instead of 27½ cents as a base. In an effort to distribute the effects of these price ceilings fairly among consumers, the National Production Administration began to allocate domestic copper production and foreign entitlements to all users on an approximate 60–40 percent basis instead of just allocating copper. As a result of this move, some domestic consumers whose domestic copper-producing affiliates usually supplied ample copper had to sell some of their production to competing companies and to buy some foreign metal or to decrease consumption. Domestic fabricating affiliates of American mining companies producing in Chile, on the other hand, since May 1951 had been absorbing the 3-cent differential in costs between 24½ and 27½ cents but during that time were not receiving the higher prices permitted for their Chilean-produced metal. They continued to receive only 24½ cents for their output until July 1952 when they, with all other fabricators, were permitted to pass on to consumers 80 percent of increased costs over 24½ cents.

The improved supply-requirements relationship resulted in abandonment of price controls on copper and copper products in February 1953; and by the end of April domestic and foreign prices, except Chilean, were close together at about 30 cents a pound; the price for the Chilean outputs of the three large American mines was held, under Chilean Government direction, at 35½ cents a pound in Chile (about 36½ cents in the United States) until December 1953.

Very few summaries are available on the costs of producing copper. References to some articles on the subject are given in the Bibliography.

TARIFF

When the market for copper collapsed at the end of the 1920's, domestic producers had difficulty in disposing of their output and feared that they might even lose a large part of the domestic market. As a result of pressure at the time, an excise tax of 4 cents a pound was imposed June 21, 1932. The Government became virtually the only importer in World War II, and during the period when it held this position the tax was not applicable. The tax was reinstated for a very brief period, following which it was suspended by acts of Congress from April 30, 1947, to June 30, 1950. The suspended tax, meanwhile, was reduced, as a result of the Trade Agreement negotiations at Geneva in 1947, to 2 cents a pound, effective March 16, 1949. The 2-cent tax finally came into effect on July 1, 1950. It was, however, suspended again on April 1, 1951, and the suspension was subsequently continued to June 30, 1955. The law provides that the Tariff Commission must notify the President within 15 days after the end of any month in which the price drops below 24 cents a pound, delivered Connecticut Valley, and within 20 days thereafter the President must revoke the suspension.

OUTLOOK

During World War II and in most of the postwar years, United States requirements for copper exceeded supplies, and the world situation was almost parallel. As a consequence, rather than a search for new uses there has been more general interest in the development of new sources of supply, in attempts to make supplies of copper stretch further, and in the study of substitutes. For many years before World War II the United States produced more than enough copper for total needs and exported the surplus. The outlook now is that the United States will continue to be a net importer of copper, except for possible brief periods when demand is cut more sharply than production, and that this dependence is more likely to grow than to diminish. As a result of this trend, interest in the search for substitutes has been accelerated.

Other trends are toward greater international independence of United States smelting, refining, and fabricating facilities, in the order mentioned. Chile has a new Government smelter that is processing a large part of the ores and concentrates formerly shipped to United States plants, Mexico has a relatively new refinery that reduces the supply of foreign copper to United States plants, Northern Rhodesia is becoming increasingly self-sufficient in regard to refining facilities, and the same trend is noticed elsewhere, as in Yugoslavia, where efforts are being made to fabricate a substantial part of the copper mined.

If the international tension, with attendant semimobilization, continues for several years, if there is a continued high level of industrial activity, and if the population increases as anticipated, annual consumption of new copper in the United States may average 1,600,000 tons by 1960. This figure makes no provision for Government stockpiling but includes consumption of manufactured goods for export. United States production may be 1,000,000 to 1,100,000 tons at that time, which means, of course, that net imports of copper must average one-half million tons annually. Foreign deposits should be readily able to supply this large excess over all other world needs. If the efforts of other countries to develop higher standards of living and to become more self-sufficient in regard to smelting, refining, and fabricating facilities bear fruit, and they may be expected to do so, in part at least, this large excess of foreign production might not be available to the United States; but, by the same token, under such circumstances the consumption requirements of the United States would fall below the 1,600,000 tons indicated.

PROBLEMS

Basically the United States copper industry is on a sound footing and is a prosperous and well-balanced business. The prospects are that the demand for copper will hold the price at profitable rates for a long time to come. There may be short-term slumps; and, of course, a general, long-continued, industrial depression would affect the copper industry as seriously as any other. However, nothing of that kind is foreseen. A large share of the copper-productive machinery is integrated with the consuming industry. The industry, as a whole, is characterized by the small number and large size of its units. Measured-ore reserves are in good shape and probably are as large, in terms of metal recoverable at current prices, as they ever were. American copper companies have a large financial share and a considerable management responsibility in many of the principal foreign enterprises. The industry has supported research in the past, and losses of copper at all stages from mining through milling and smelting and refining into the consuming industries have been reduced to a degree that makes such losses a minor problem.

Some anxieties of the industry are the following:

1. Competition from foreign copper produced at lower unit labor costs.
2. The tariff.
3. Depletion of domestic ore reserves.

4. Inroads of other materials, particularly aluminum, into the copper market.
5. The future of marginal producers.
6. Growing dependence on foreign sources of supply.
7. Rise of nationalism abroad.
8. Wide price fluctuations outside of the United States.

Of all these and many others that may be named, the most serious long-term problem relates to the United States increasing dependence on foreign sources of supply. This dependence has implications that are not wholly related to military security. Stockpiling, as currently practiced, does not protect United States buyers in the world market if unfriendly or hostile interests choose to exercise arbitrary controls over critical raw materials.

As has been pointed out elsewhere, the growing reliance on foreign copper is not due to exhaustion of domestic reserves but rather to expanded domestic consumption. The discovery, exploration, development, and exploitation of large, rich, foreign ore deposits have made possible the filling of the gap between domestic production and consumption plus stockpiling.

To maintain a substantial supply from domestic mines, the problems will be to develop economic methods of mining and processing ores of lower and lower grade, of the types now being produced, as well as to develop new or improved methods of extracting copper from oxide, mixed oxide-sulfide, and complex ores. This, however, is a serious problem. The cost of production depends largely on labor costs, and each advance in these and other costs, unless accompanied by compensating factors like higher copper price or improved technology, converts a certain quantity of marginal reserves into merely mineralized rock which cannot be extracted and processed economically.

Development of improved methods of scrap recovery and utilization would help to solve the foregoing problems by supplementing the contribution from domestic mines.

Inadequate knowledge of the ways in which materials in more abundant supply can be substituted for copper without loss of efficiency is a problem. It is of particular concern to Government in times of emergency and a problem that continues to worry the industry.

Lack of data on certain aspects of the copper industry robs industry and Government of some of the tools required for most efficient handling of problems of balancing supplies and requirements in emergency periods, as well as in normal times. Some of the missing data are:

1. End-use consumption figures on a continuing basis.
2. Figures on consumption by geographic areas.
3. Figures on generation of scrap by geographic areas.
4. Complete and comprehensive data on reserves.

Most foreign countries have more serious gaps in their data.

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A large volume of literature is available on copper covering all phases of the industry. The Bureau of Mines publishes a chapter on Copper in the Minerals Yearbook (*4*) which presents complete statistical coverage plus a review of domestic and foreign events of significance for each year. Statistical information is also available in the annual volumes of the American Bureau of Metal Statistics and the American Metal Market annual, Metal Statistics. The Copper Institute and U. S. Copper Association monthly reports (the latter marked "Confidential") are published in the technical press.

The Copper and Brass Research Association and the Copper Development Association (London) distribute numerous publications on the utilization and properties of copper and its alloys. The larger copper consumers have information pamphlets for distribution on the advantageous characteristics of copper and copper products. The Bureau of Mines has issued numerous reports of investigations on domestic copper occurrences and operations.

The copper industry is well covered in Peele's Mining Engineers' Handbook and Taggart's Handbook of Ore Dressing. Other standard texts on the nonferrous metals devote considerable space to copper. Some of these are: An Outline of Metallurgical Practice, by Hayward; Nonferrous Production Metallurgy, by Bray; and Handbook of Nonferrous Metallurgy, vol. II, by Liddell.

CORUNDUM

By

Henry P. Chandler[1]

CORUNDUM, a natural oxide of aluminum is, with the exception of diamond, the hardest mineral known. It is used as an abrasive, and because of certain specialized applications is considered to be sufficiently essential to warrant strategic stockpiling.

Summary

Although occurrences of corundum are known in many parts of the United States, efforts to develop a dependable commercial domestic source have been unsuccessful in recent years, and the United States currently depends upon overseas imports. Nearly all of the world production outside of Russia is from South Africa, and United States imports for consumption are purchased by one company.

During the past 8 years three-quarters of the corundum grain sold in the United States has been ground to fine sizes and used by the optical industry for lens grinding. The remainder is sold, in coarser sizes, to the manufacturers of grinding wheels, because corundum wheels are still in demand for grinding certain types of castings.

Artificial alumina abrasive competes with corundum, and the primary need currently is for more technical and other information on these two materials to guide both industry and Government activities involving them.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

Until invention of artificial abrasives in the 1890's, natural corundum and emery (an impure corundum) were the principal abrasives used in bonded grinding wheels. Since 1900 aluminum oxide and silicon carbide have largely superseded natural abrasives in wheels, but some consumers prefer corundum wheels for a few grinding operations. Corundum also continues to hold a place in the optical industry, particularly for lens grinding.

Beginning in 1870, the North Carolina and Georgia corundum deposits supplied the abrasive for the United States requirements. Canadian production supplied the industry from about 1900 until 1915, and when the Canadian sources declined, South African and Indian corundum appeared on the market. Since 1919 the importations from India have been unimportant; and South Africa—and, to a minor extent, Madagascar— have supplied the United States with corundum. In 1953 some 95 percent of the world supply, outside of the Soviet bloc, came from Africa.

Macon and Jackson Counties, N. C., Rabun County, Ga., and Gallatin County, Mont., are areas in which corundum has been mined in the United States. By 1900 these deposits were no longer commercially workable. Production was revived in North Carolina during 1917 and 1918, but since then there has been no corundum output from that area. Some corundum was produced in Montana between 1901 and 1905 and again during World War II, but the deposits are now idle. Efforts to establish and develop domestic sources of corundum during World War II yielded negligible results.

For nearly 20 years corundum was produced at various points in Canada, the Craigmont property in Renfrew County, Ontario, being the most important. Production at this mine ceased in 1913, but during World War II the old tailing piles were reworked and produced some 2,000 tons of corundum concentrate. There has been no output of corundum in Canada since then. Several deposits of corundum are known to occur in a belt about 6 miles wide and 100 miles long in eastern Ontario; however, the deposits are small and scattered, with the corundum content rarely reaching 5 percent.

The American Abrasive Co. of Westfield, Mass., is the only firm importing and concentrating corundum in the United States. Its agent in South Africa is the American Abrasive Co. of South Africa, Ltd., Pietersburg, Transvaal. This firm is engaged in collecting corundum concentrate from the producers in that area and shipping it to the United States.

In South Africa most corundum comes from the Zoutpansburg district in northern Transvaal, with small productions from Namaqualand, Nyasaland, and Southern Rhodesia. The Transvaal deposits occupy a triangular region of about 3,000 square miles and are divided into 2 fairly distinct areas: The Plateau area from Pietersburg northward and the Low Country area eastward from the foot of the Drakensburg Mountains. In the Low Country the most productive districts are mainly east of Mica Siding; in the Plateau area they lie around Bandolierkop and Mara.

Corundum is mined in several forms. Crystal corundum consists of loose crystals found in shallow eluvial deposits formed by the disintegration of the corundum-bearing rocks. Boulder corundum is found as blocks composed of crystals held together in a matrix of weathered feldspar. Reef corundum occurs as a rule in veins or dikes a few feet wide that intrude the basic rocks. In the Plateau area, these reefs are nearly always a white, coarse-grained "plumasite" containing scattered crystals of grayish corundum. The commercially valuable reefs contain 20 to 40 percent corundum.

In the corundum-mining districts of the Transvaal, about 60 percent of the corundum production comes from eluvial deposits and 40 percent from reefs. Eluvial deposits are the result of the alteration and breakdown of the reefs. About 10 small individual mining operations are carried on; the largest has a yearly output of 800 tons of corundum concentrate. The deposits are not large, and the operators do not feel justified in much capital expenditure for plant equipment to increase their efficiency over that obtained by hand methods. Corundum-bearing material at the eluvial deposits is mined from open pits (seldom over 20 feet deep) with picks and shovels, and hand drilling when blasting is necessary. The material is screened and washed to recover the crystal. No attempt is made to save sizes that pass a ⅛-inch screen. A small tonnage is supplied by natives who pick up corundum crystals from the surface of the ground after rains. In mining the reef deposits some hand cobbing is done, as the larger crystals break away from the surrounding material. The finer sizes are recovered by washing.

Tables 1 and 2 show basic statistics of annual production, imports, and consumption of corundum.

TABLE 1.—*Corundum imports and consumption in the United States, 1944–51*

Crude corundum concentrate [1]		Corundum grain used in the United States (short tons)		
	Imports into United States, short tons	Grinding wheels	Optical	Tota
1944	6, 402	1, 032	3, 763	4, 795
1945	6, 244	929	3, 003	3, 932
1946	4, 207	880	1, 890	2, 770
1947	2, 401	622	2, 371	2, 993
1948	3, 612	701	1, 175	1, 876
1949	2, 013	264	1, 320	1, 584
1950	3, 543	412	1, 626	2, 038
1951	4, 893	1, 080	2, 750	3, 830
Total	33, 315	5, 920	17, 898	23, 818

[1] 4 tons of crude corundum concentrate is usually required to make 3 tons of commercial corundum grain.

Comparative prices of artificial aluminum oxide, corundum, and domestic and imported emery that were quoted in November 1952 are shown in the following table.

Abrasive:	*Price per pound, ton lots*
Aluminum oxide, grinding-wheel grain	12¼¢, delivered.
Corundum, grinding-wheel grain	12¼¢, delivered.
Corundum, optical grain:	
Sizes 120 and coarser	10½¢, f. o. b.
Sizes 140 and finer	11½¢, f. o. b.
Corundum optical powders (500 and finer)	31½¢, f. o. b.
Turkish emery	10¢, f. o. b.
American emery	7½¢, f. o. b.

The use of corundum by the grinding-wheel industry is confined almost entirely to the coarse size range, namely, sizes Nos. 8 through 24. The optical industry uses a wide range of sizes from No. 60 down to and including 220-mesh for coarse grinding and sizes 500 to 2600 for finer grinding.

TABLE 2.—*World production of corundum, by countries,*[1] *1944–48 (average) and 1949–53*[2]

[Metric tons]

(Compiled by Helen L. Hunt)

Country [1]	1944–48 (average)	1949	1950	1951	1952	1953
Argentina	[3] 30	(4)	(4)	(4)	(4)	(4)
Australia	2				55	
Brazil	[3] 45	(4)	(4)	(4)	(4)	(4)
Canada	405					
French Equatorial Africa	39					
India	264	1, 493	304	557	647	(4)
Kenya			2			
Madagascar	29	7				
Malaya, Federation of [3]			10	25		
Mozambique	253		16		(4)	(4)
Nyasaland	202	113	187	101	47	(4)
Southern Rhodesia	25					765
South-West Africa			10			
Union of South Africa	2, 923	2, 464	3, 201	4, 563	3, 791	1, 692
Total (estimate) [1]	7, 940	9, 000	9, 000	10, 000	10, 000	9, 000

[1] In addition to countries listed, corundum is produced in U. S. S. R., but data on production are not available, and estimate is included in the total.

[2] This table incorporates a number of revisions of data published in previous annual reviews of corundum.

[3] Estimate.

[4] Data not available; estimate by author.

The (October 1952) price being paid for clean corundum concentrate by the American Abrasive Co., Ltd., of South Africa, is £ S. A. 25 (or $70) a ton at its collection point near Pietersburg, Transvaal. The material is shipped to the United States as received, without further treatment.

The delivered cost of corundum concentrate at Westfield, Mass., in 1952 was about $110 a short ton. The loss in cleaning and grading the corundum at Westfield is about 25 percent, making the material cost of the finished product in the neighborhood of $147, to which must be added the cost of cleaning and screening into the sizes called for by the abrasive industry.

The tariff on corundum entering the United States is one-half cent a pound.

Only an export certificate is required by the South African Government for shipments of corundum from that country, for which only a nominal fee is charged.

Relatively few data are available on corundum research by private companies; however, it is apparent that such research has resulted in progressive substitution of artificial abrasives for corundum. The National Bureau of Standards has devised a testing apparatus for determining the abrasive performance of samples of corundum.

Corundum is now being stockpiled by the Munitions Board, primarily for use as a fine-grain abrasive to grind optical lenses. Inquiries have indicated that a number of industrial firms no longer consider corundum essential, but the two major consumers of optical grain still do.

OUTLOOK

For many years no new uses for corundum have been developed. Demand (including stockpiling) has been greater than supply. This resulted in unsatisfactory progress in stockpiling, but the commercial needs served by natural corundum apparently have been reasonably well satisfied. No large new sources of supply have been found, so an expanding market assisted by a price reduction cannot be expected. Corundum probably will continue to have a good market as long as its price can be maintained at about the same level as that of artificial aluminum oxide. As no major changes in either the supply or uses of corundum are anticipated, the corundum industry is expected to maintain its present course during the next few years, with tonnage remaining at about 2 percent of that of the artificial abrasives.

PROBLEMS

The principal problem at the moment is lack of enough technical facts regarding the comparative grinding properties of corundum and its substitutes. Better technical data would help to resolve the question of the essentiality of corundum. If in the end corundum is found to be so essential as to warrant continuation of the stockpiling program, special arrangements will have to be made to obtain the necessary quantities.

Although the outlook for this commodity is not now encouraging to producers, there is no conclusive evidence that the market is saturated. There apparently is an opportunity for successful development of new deposits if the product can be sold at prices competitive with those of artificial aluminum oxide. Under these conditions the development of low-cost production facilities is a problem.

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DIAMONDS

By

Henry P. Chandler [1]

ALTHOUGH the diamond has been valued as a gem stone since antiquity, it is only in recent years that its extraordinary utility as an abrasive has been recognized. The industrial preference for diamonds, costing thousands of dollars per pound, over silicon carbide and aluminum oxide, which are priced at cents per pound, illustrates their effectiveness.

Summary

The principal market for diamonds, from the standpoint of value, is in jewelry, but quantitywise, industrial uses predominate. Its distribution and price structure are carefully controlled by producers. Gem diamonds are very important to the jewelry trade throughout the world; however, industrial diamonds now occupy a far more important place in industry as a whole, as they have become essential in maintaining the high-speed, precision methods of production that characterize modern manufacturing.

To meet the principal industrial requirements, diamonds are employed in three main forms: (1) In rough, uncut, and unpolished state; (2) cut and polished to some particular shape; and (3) in the form of grit or powder. In the first two forms, the stones are carefully sorted by quality, size, and shape for the purpose for which they will be best suited; the inferior material is crushed for smaller sizes and powder (*15*).[2]

In industry the term bort (also spelled boart and bortz) is commonly applied to diamond material that is used in a crushed or fragmented form (crushing bort). However, the term also is used commercially to designate any diamonds that are unsuitable for use as gem stones because of color, size, or other imperfections.

United States import statistics divide industrial diamonds into the following categories: (1) Carbonado and ballas (carbonado being a closely knit aggregate of very small diamond crystals and ballas being a globular mass of diamond crystals); (2) diamond dust or powder; (3) crushing bort, including bort that has been crushed or is suitable for crushing; (4) other industrial diamonds, which includes miners' diamonds, not elsewhere specified, and glaziers' and engravers' diamonds, not set. It also includes diamond bort, manufactured (diamond dies).

Experience has shown that certain materials, such as carbides, can be ground efficiently with diamond grinding wheels. Carbides can be rough-ground with silicon carbide wheels; but close tolerances, acceptable surface finishes, and keen cutting edges can best be attained with a diamond wheel. Performance advantages have reduced the significance of the high initial price

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[1] Commodity-industry analyst, Bureau of Mines.

[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

of diamonds. It is on this basis that industrial diamonds are being increasingly used in modern industry, in spite of their advancing price. During 1953 some 20 million carats of diamonds (including 16.4 million industrials) was mined in the world.

The United States has always depended completely on foreign sources (at present, principally the Belgian Congo) for its supply of industrial diamonds. No commercial deposits are known in the United States. Development of domestic mines adequate to meet any substantial part of the United States demand is considered to be unlikely; however, the recent discovery of a method for synthesizing diamonds opens new possibilities for eventual development of domestic sources of supply.

Problems of the diamond industry include those concerned with supply, conservation, more efficient utilization, salvage, and substitutes.

BACKGROUND

Before 1920 diamond production came largely from South and South-West Africa. After 1920 important diamond discoveries were made in other parts of Africa, notably in Belgian Congo, Angola (Portuguese West Africa), Gold Coast, and Sierra Leone. These came into production between 1920 and 1933 and are especially rich in industrial diamonds. These regions now constitute the principal sources of world supply of that material; Belgian Congo is the most important.

Before 1920 the South African diamond producers had formed an association and agreed to limit production to meet the demands of the trade and to sell their diamonds, both gem stones and industrials, exclusively to a single group of London diamond merchants, commonly known as the "Diamond Syndicate." The syndicate was organized in 1893. DeBeers Consolidated Mines, Ltd., organized in 1888, was the most important African producer.

To maintain control over the world market for diamonds of all kinds, especially the United States market, the Diamond Syndicate agreed to market the production of new fields as they became important; arrangements went into effect in Angola in 1923, Gold Coast in 1925, Belgian Congo in 1926, and Sierra Leone in 1935. Confiscated Russian jewels appeared on the European market in 1920 and were purchased to prevent a price decline.

Rich alluvial deposits were discovered in parts of the Union of South Africa in 1926 and 1927; but they were rapidly developed, and by 1930 their production had declined. To maintain prices, the Diamond Syndicate acquired a large part of this production.

The depression of 1929 made it difficult for the syndicate to maintain prices and to finance its extensive purchase contracts. To relieve the syndicate and avoid an uncontrolled market, the South African producers purchased half of the syndicate's inventory. To assist in financing this purchase the Diamond Corp. was formed in 1930; it also took over the syndicate's exclusive purchase agreements with the producers. DeBeers subscribed to 80 percent and the Consolidated Diamonds of South-West Africa to 20 percent of the stock in this new corporation.

In 1934 the Diamond Trading Co. was formed (DeBeers and the Diamond Corp. each took 50 percent of its capital stock) to be the exclusive selling agency for all diamonds produced by the African mines formerly having sales contracts with the syndicate. This was some 99 percent of the African production and 97 percent of the world production.

In 1938 the Diamond Trading Co. made an agreement with the Producers Association to limit production whenever it seemed desirable. In 1946 the Industrial Distributors (1946), Ltd., was formed to handle the sale of industrial diamonds exclusively. The Diamond Trading Co. has renewed and maintained in effect exclusive sales contracts until the present.

Before 1940 the annual quantity of industrial diamonds mined was greater than the market absorbed, even though the demand for industrials was expanding. The low prices asked for industrials, especially bort, encouraged increased use. However, from 1940 until the end of the Korean conflict the world sales of industrials of all kinds exceeded production. This demand, which included some material for the National Strategic Stockpile, was met in part from the large stocks that had been accumulated by the Diamond Corp., its subsidiaries, and others. These stocks are now believed to be depleted. Industrial diamond dealers and users that buy from Industrial Distributors (1946), Ltd., at the London sales must take what is offered, as there is no other major primary agency from which they can buy.

INDUSTRIAL DIAMONDS

TECHNOLOGY

Industrial diamonds are produced from two types of deposits: (1) "Pipes" of kimberlite and (2) placers, or alluvial deposits. Ninety percent of the African production is from placer deposits and the rest of the world production is entirely so. Methods similar to those used in placer-gold mining for stripping the overburden and mining the pay gravel are employed in alluvial diamond mining. Operations are now almost completely mechanized, hand labor being mostly used in cleaning the bed rock and other types of mining that cannot be done mechanically. At the better equipped mines of Belgian Congo, the top soil is stripped with bulldozers and wheel scrapers, and the diamond-bearing gravel is treated in washing plants.

The final separation is made with grease tables, electromagnetic separators, and hand picking. In South America simple hand methods of mining and concentrating are still in use.

Underground mining methods are used in the "pipe" mines of South Africa, followed by crushing and various concentrating methods, including the use of the heavy-media process, concentrating pans and jigs. At the Premier Mine no pans are used following the heavy-media process and jigs are only used to concentrate the -10 mesh material. At Kimberley the pans precede the heavy-media process, and jigs are used only on the -10 mesh fraction. The final separation is accomplished by grease tables, electrostatic and electromagnetic separators, and hand picking.

Before diamonds are offered for sale the selling agencies of the Diamond Trading Co. sort the material into selling categories or "series" in which they are offered to the market.

The industrial diamonds sold through Industrial Distributors (1946), Ltd., usually are sent to Johannesburg and the cuttable stones to Kimberley for sorting and evaluating, then to London, where they are sold at what are known as "sight" sales, held several times a year. Only selected dealers and large consumers are invited to the sales; the customers use their purchases or distribute them to smaller dealers and consumers.

SUBSTITUTES

No adequate substitute or synthetic material has yet been developed to replace industrial diamonds completely; however, several processes are being developed that might be substituted for diamond-grinding-wheel operations or have possibilities for machining and grinding hard materials. These techniques include: (1) Electrodischarge processes, (2) electrolytic (or electrolytically assisted) processes, (3) ultrasonic abrasive grinding, (4) silicon carbide belt grinding and (5) silicon carbide wheel grinding.

1. In electrodischarge processes the stock is removed by sparking or arcing between the work and an electrode. This type of process is being used in shaping and finishing dies.
2. Electrolytic grinding is usually done by a metal-disk cathode or a metal-bonded diamond wheel revolving close to an anodic workpiece while an electrolyte is applied to the disk and workpiece in the same manner as a coolant is applied to a grinding wheel and work.
3. In the ultrasonic process stock removal is accomplished by employing a finely divided abrasive in water flowing over a tool vibrating at ultrasonic frequencies.
4. Silicon carbide belt grinding has some application in the final finishing operation on single-point tools.
5. Research on silicon carbide wheel grinding is directed toward improvement in bonds and grinding techniques.

RECLAMATION

For conservation and cost-reduction reasons the larger users of industrial diamonds have begun to save sludges from diamond-grinding operations, worn drill bits, tools, and wheel stubs. Several plants are now producing commercially industrial-diamond material reclaimed from such waste by various methods. Estimates indicate that perhaps 10 percent of the quantity of industrial diamonds used could be recovered for reuse.

RESERVES

The largest known reserves of industrial diamonds are in Belgian Congo, and the most important one is at the Bakwanga mine. Other diamond fields in west and central Africa have substantial reserves, and prospecting is being conducted in promising localities.

The "pipe" mines of South Africa and Tanganyika also have relatively large reserves; they are prospective producers for many years to come.

Reserves in Brazil and elsewhere in South America are unknown, but it is believed that the present production can be maintained for many years.

STATISTICAL INFORMATION

The Bureau of the Census compiles information on imports and exports of industrial diamonds.

The National Production Authority collected data from the users of crushing bort and diamond powder regarding the use of these materials in diamond grinding wheels and other applications in industry, but this has been discontinued.

Publications of the Industrial Diamond Information Bureau, 32–34 Holborn Viaduct, London, England, give statistical data on the production and use of industrials throughout the world.

The Federal Bureau of Mines and the National Museum compile information on diamond production throughout the world.

PRODUCTION, IMPORTS, AND CONSUMPTION

World production of diamonds in 1953 is shown in table 1.

Belgian Congo supplied 73 percent of the world production of industrial diamonds in 1953 on the basis of quantity. In that year, industry in the United States used some 11 million carats of industrial diamonds of all types, or approximately two-thirds of the world production.

Neither the United States, nor other North American countries produce any significant number of diamonds, either of gem or industrial quality. The world production for the past 25 years is shown in table 2.

TABLE 1.—*World production of diamonds in 1953*

[Metric carats]

	Total production	Industrial diamonds	
		Production	Percent of total
Africa:			
Angola	729, 377	307, 000	42
Belgian Congo:			
Beceka	12, 016, 198	11, 750, 000	98
Kasai	564, 058	250, 000	45
French Equatorial Africa	140, 144	92, 000	66
French West Africa	[1] 180, 000	120, 000	66
Gold Coast:			
European companies	917, 429	780, 000	85
African producers	1, 249, 935	735, 000	60
Sierra Leone	472, 934	322, 000	68
South-West Africa	617, 411	123, 000	20
Tanganyika	170, 679	73, 000	43
Union of South Africa:			
Premier mine	1, 269, 925	978, 000	77
DeBeers group	1, 127, 830	564, 000	50
Other "pipe" mines	117, 181	59, 000	50
Alluvial	[1] 180, 000	[2] 90, 000	50
Other regions:			
Brazil	[1] 200, 000	100, 000	50
British Guiana	35, 306	21, 000	60
Venezuela	84, 790	60, 000	71
India, Borneo, Australia, U. S. S. R	[1] 5, 000	3, 000	60
Total	20, 100, 000	16, 400, 000	82

[1] Estimate, subject to revision.

[2] Including State-owned mines in Namaqualand.

TABLE 2.—*World production of industrial diamonds*

[Carats]

1929	4, 000, 000	1942	8, 300, 000
1930	5, 300, 000	1943	7, 000, 000
1931	5, 500, 000	1944	9, 800, 000
1932	5, 100, 000	1945	12, 200, 000
1933	3, 300, 000	1946	8, 000, 000
1934	5, 600, 000	1947	7, 700, 000
1935	5, 600, 000	1948	7, 900, 000
1936	6, 700, 000	1949	11, 800, 000
1937	7, 600, 000	1950	12, 800, 000
1938	9, 500, 000	1951	14, 000, 000
1939	10, 500, 000	1952	15, 800, 000
1940	11, 200, 000	1953	16, 400, 000
1941	7, 600, 000		
		1929–53	219, 300, 000

In the 10 years 1941–50, despite a major increase in output the world demand for industrial diamonds exceeded production by over 50 million carats. This difference was handled by drawing on accumulated stocks of industrial diamonds in the hands of producers, dealers, and consumers throughout the world. The factors chiefly accountable for the greatly increased demand were wartime requirements, postwar industrial growth, and strategic stockpiling.

United States imports of industrial diamonds during the period of rapid growth in their use are shown in table 3.

TABLE 3.—*Imports of industrial diamonds into the United States*

[Carats]

1929	46, 901	1942	11, 203, 704
1930	145, 862	1943	12, 084, 133
1931	224, 970	1944	12, 614, 507
1932	163, 704	1945	10, 729, 869
1933	263, 484	1946	4, 625, 282
1934	526, 007	1947	3, 999, 119
1935	954, 589	1948	10, 421, 207
1936	1, 166, 094	1949	6, 279, 096
1937	1, 885, 670	1950	11, 039, 036
1938	1, 396, 247	1951	12, 120, 647
1939	3, 568, 730	1952	13, 452, 819
1940	3, 809, 071	1953	12, 765, 495
1941	6, 882, 248		

In the United States during 1953, according to information supplied by the United States Department of Commerce and the Industrial Diamond Association, the estimated industrial diamond requirements were as shown in table 4.

TABLE 4.—*Estimated consumption of industrial diamonds in the United States in 1953*

[Carats

Bort and powder, mostly for grinding wheels	9, 000, 000
Stones for diamond drilling	1, 300, 000
Stones for grinding wheel dressers	600, 000
Stones for shaped tools, etc	50, 000
Stones for wire drawing dies	50, 000
Total	11, 000, 000

Over three-quarters of the bort and powder is consumed in making diamond grinding wheels for sharpening cemented carbide tools, rock drills, and wheels for cutting concrete and stone. The use of diamond wheels for glass grinding, such as producing glass windows for automobiles and generating curves on lenses, is expanding.

Diamond bits now are used for exploratory core drilling, quarrying, and drilling blast holes and oil wells. An experimental use is in drilling ventilation holes in mines. Improvements in diamond drills and in drilling techniques have contributed to growth in this phase of the diamond industry (*10, 11, 12*).

Rock-drill bits and grinding-wheel dressers take the larger, more perfect stones. Wire-drawing dies require those of particularly good quality.

The use of crushing bort in 1953 was estimated by the National Research Council as follows (*19*):

	Percent
1. Sharpening milling cutters, broaches, and fluted tools of cemented carbides	26
2. Surface, cylindrical, and internal grinding of cemented carbides	18
3. Sharpening single-point tools of cemented carbides	15
4. Chip-broaches grinding of cemented carbides	10
5. Shaping and finishing dies	8
6. Glass-industry grinding operation	8
7. Miscellaneous	15
	100

STRATEGIC CONSIDERATIONS

The United States is now stockpiling industrial diamonds of various classifications. Transportation is not a serious factor for this commodity, as a year's supply of industrial diamonds could easily be flown into the United States on one flight of a transport plane. On the other hand, the final treatment plants in Africa are vulnerable to sabotage; should they be damaged, the supply would be curtailed seriously.

PRICES

At the end of 1941 crushing bort was selling in London at approximately 92 cents a carat. After negotiations with representatives of the United States Government, the Diamond Corp. reduced the price of crushing bort in London to 52 cents a carat and maintained it at that price for the duration of the war. Immediately after the war the price reverted to the prewar level. The price movement of crushing bort in London during recent years has been reported as follows:

	Per carat
Before 1942	$ 0. 92
January 1942	. 52
September 1946	. 92
March 1948	1. 02
December 1948	1. 28
January 1949	1. 50
January 1950	1. 50
March 1951	1. 79
January 1952	[1] 2. 19
March 1952	[1] 2. 21
December 1954	[1] 2. 22

[1] From January 1952 to December 1954 crushing bort was priced at 15/9 in London.

Basic prices for all classifications of industrial diamonds now are established in London by Industrial Distributors (1946), Ltd.

Table 5 shows the average value of industrial diamonds imported into the United States, 1948–53, inclusive.

TABLE 5.—*Average value of industrial diamonds imported for consumption in the United States, 1948–53.*

[Excluding manufactured bort and diamond dust]

	Per carat		Per carat
1948	$3. 13	1951	$3. 82
1949	2. 77	1952	3. 79
1950	3. 33	1953	3. 67

TAXES AND TARIFF

Industrial diamonds are not subject to luxury taxes, and there is no import duty on them. The import duty on cut gem diamonds is 10 percent; if from Cuba, 8 percent. There is no import duty on rough, cuttable gem material. The import duty on diamond wire-drawing dies is 15 percent, if from Communist countries 30 percent.

CONSERVATION PRACTICES

Improvements in the manufacture of industrial-diamond products and grinding machinery and development of better grinding methods are raising grinding efficiency and thereby, in effect, conserving diamond supply. In core drilling the setting of diamonds in the bits so that the hardest face of the diamond is exposed (referred to as orientation) as a cutting medium improves diamond life and consequently reduces drilling costs.

Eight firms now process diamond-containing sludge and swarf and other materials, and the diamond so recovered is an important addition to the domestic supply. Many diamond-wheel users have found that, by setting up a separate collection device for sludge, they can reduce contamination with other abrasives, and the diamond content of the sludge is thereby increased. By saving sludges many diamond-wheel users are reducing their costs.

Between 1951 and 1953 the National Production Authority conducted a diamond conservation and salvage program, during which users of industrial diamonds were urged to get the most effective use from that material and to save grinding sludges containing diamond.

RESEARCH

Several methods for sharpening cemented carbide tools by electrolytically assisted grinding and other processes are now being investigated. It seems probable that, for certain operations, the new processes have value, and may even prove important. These methods have not yet been developed sufficiently for full appraisal of their commercial potential but certainly merit thorough investigation.

On February 15, 1955, the General Electric Research Laboratory at Schenectady, N. Y., announced that it had successfully produced synthetic diamonds under conditions of high pressure and temperature. The diamonds produced were large enough to meet the size requirements of some industrial uses. The company indicated, however, that production costs were relatively high; much additional research and process development will therefore be required before synthetic diamonds can compete with natural.

GEM DIAMONDS

World production of gem diamonds totaled 3,600,000 carats in 1953. The United States continued to be the largest importer; during 1953 cut but unset diamonds imported into the United States totaled 444,247 carats, and rough or uncut gem material imported totaled 733,630 carats.

There are about 300 diamond-cutting establishments in the United States; they employ about 1,700 cutters and polishers. The industry is mostly located in New York City.

A survey of the retail-jewelry business in the United States for N. W. Ayer & Son, Inc., indicated a slight decline in the sale of diamond jewelry in 1953 compared with 1952. Only a few jewelers reported sales of diamonds for investment purposes. An improved supply of all sizes of diamonds was noted by many jewelers, the only shortage being for top-quality gems in the larger sizes. Most jewelry firms reduced their diamond inventories during 1953. The DeBeers Co. is conducting an extensive advertising campaign to encourage diamond sales in America.

OUTLOOK

The Industrial Diamond Information Bureau of London each month lists the new applications of diamonds in all branches of industry and all patent applications throughout the world in which diamonds are included. The number of these new applications indicates the importance attached to the abrasive qualities of the diamond and its expanding use in industry.

The trend in the use of industrial diamonds has followed the production of cemented carbide tools and similar appliances that require diamond grinding wheels to machine them. These wheels consume about 80 percent of the total imports of crushing bort. The use of tools requiring the larger sizes of industrials has also gained in the past few years. The value of industrial diamonds is so well known that diamond suppliers believe an increase in available supply at attractive prices would expand its application in industry in major new markets. It is apparent, therefore, that the market for industrial diamonds is growing.

The present trend of output is up, and a major addition to the supply of industrial diamonds probably can be achieved by increasing production at the Bakwanga mine in Belgian Congo; this mine now produces nearly three-quarters of the world supply of industrials and has ample reserves. A new hydroelectric plant began operations in 1953 and is substantially reducing mining and concentrating problems and costs.

Previous methods of concentration have allowed large quantities of small diamonds to be wasted. New methods of concentration are improving recovery. This may increase the production from present operations, bring formerly submarginal deposits into production, and possibly permit reworking of accumulated tailing piles.

Prospecting for diamonds is continuing both in Africa and South America, and discovery of new fields and extensions of the older ones is expected. Reopening of more of the older "pipe" mines also is being considered.

Even though substitutes are receiving increasing attention, the combination of substantial reserves and growing requirements make the market outlook for this commodity seem bright.

PROBLEMS

The increasing use of industrial diamonds and the decreasing producers' stocks have reached a point where consumer demand (according to information received from producers) must be filled from current production. For the past 14 years production has lagged behind demand(which has included both requirements for current consumption and for stockpiling). As indicated in the Outlook section, both diamond supply and demand are increasing. It is noteworthy that recently the availability of supply has improved substantially, and consumers' demands apparently are being adequately filled.

The industry has many technical and economic problems, such as rate of development, recovery processes, distribution, pricing, and use technology; information is needed to solve them.

The most serious problem from the viewpoint of the United States is its complete dependence upon distant foreign sources of supply. Not only are the sources distant, but the bulk of the supply of industrials comes from one mine. This creates a serious strategic material problem, and as one step toward its solution a stockpile of industrial diamonds is being accumulated by the United States Government. There are also a number of other means whereby the position of this commodity can be improved.

Further exploration, both in Africa and South America, is a promising means of increasing and broadening the supply. Major discoveries in South America obviously would improve the situation in the Western Hemisphere. More information is needed on potential sources of industrials and on the best employment of assistance to companies.

Substantial quantities of usable diamond material are lost in grinding sludges. Improvement in recovery practices is an industry problem. It has been estimated that as much as 10 percent of the industrial diamonds used can be recovered.

Conservation is a problem which is most acute during periods of shortage. The development and application of improved grinding and drilling techniques can materially increase efficient use of diamonds.

The substitute problem includes development of hard compounds, such as carbides, nitrides, and borides, and grinding methods that require little or no diamond.

Now that a method for synthesizing diamonds has been discovered, one of the most urgent problems of the industry is further development of this process and its commercial evaluation.

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DIATOMITE

By

Henry P. Chandler[1]

MODERN INDUSTRY has breathed new life into diatomite, the skeletal remains of teeming aquatic life, and its use is growing rapidly.

Summary

The essential constituents of diatomite or diatomaceous earth (also known as infusorial earth and kieselguhr and by a number of other designations and trade names) are the fossil remains of diatoms. This material is largely opaline silica. The purest varieties of diatomite are chalklike in appearance, lightweight, porous, and friable.

The uses of diatomite are many. The principal use categories are: As a filtration medium (60 percent of sales), as a filler (25 percent), for heat and sound insulation (10 percent), as an abrasive (less than 1 percent), and for other uses (4 percent).

Production of diatomite in the United States in 1954 exceeded 300,000 short tons annually, with a value at the mine of more than 10 million dollars.

A feature of the diatomite industry in the United States has been its ability to develop new uses for the products of its western mines. The trade has come to expect uniform grade and high-quality standards in diatomaceous products.

The principal problems concern development of new uses and servicing of the major eastern markets far from the western mines.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

HISTORY

For many years diatomite was used only in small quantities, principally as a polishing medium, but in 1860 certain German deposits were developed as a source of material for absorbing nitroglycerin and also for the silica used in waterglass. About 1900, large deposits in California were first mined commercially, but production remained low until 1915, when more rapid development began. New uses were found for diatomite when the dynamite industry turned to other absorbents. The field of filtration, particularly for sugar refining, was studied, and efficient diatomaceous filter aids were developed.

The Celite Corp. pioneered in development of the deposits at Lompoc, Calif. In the late 1920's the Johns-Manville Corp. purchased the Celite Corp. Subsequently several other diatomite companies became established as competitors for one or more of the important portions of the market.

Since 1930 the production and average price of diatomite at the mine in the United States, in short tons, as reported to the Bureau of Mines, have been as follows:

Period	3-year production	Average per year	Average price
1930–32	248, 273	82, 758	$15. 72
1933–35	244, 342	81, 447	14. 81
1936–38	279, 645	93, 215	15. 65
1939–41	360, 502	120, 167	15. 94
1942–44	524, 872	174, 957	18. 85
1945–47	640, 764	213, 588	20. 17
1948–50	722, 670	240, 890	25. 55
1951–53	908, 448	302, 816	29. 97

Current prices are reported to range from about $28 for certain filler uses up to about $50 for filter-aid grades.

The two largest producers of diatomite in 1953 were Johns-Manville Corp., Lompoc, Calif., and the Great Lakes Carbon Co. at Walteria, Calif.; Basalt, Nev.; and Terrebonne, Oreg. The third largest is the Eagle Picher Co., operating near Reno, Nev. A relatively small production was contributed by the following firms: Chick Bed Co., Fernley, Nev.; Tri-O-Lite Products Co., Carlin, Nev.; Kenite Corp., Quincy, Wash.; and James H. Rhodes Pumice Co., Espanola, N. Mex.

Diatomite is widely distributed over the world. As far as is known, the United States leads the world in production. Germany, Algeria, and Brazil are important producers, and Russia is said to have large deposits in the Caucasus Mountains.

California leads the States in production, followed in order by Oregon, Nevada, and Washington. The best known and most extensively worked deposits in the United States are near Lompoc in northern Santa Barbara County, Calif. Here beds (of a known thickness of 700 feet in places), cover several square miles. Commercial deposits are also worked in Los Angeles County, and occurrences are noted in Monterey, Shasta, Inyo, Orange, and San Luis Obispo Counties.

In Oregon important deposits of fresh-water diatomite are now being worked near Terrebonne, Deschutes County. These have a thickness up to 67 feet. Several other counties have occurrences of diatomite.

Nevada has commercial fresh-water deposits in Churchill, Elko, Esmeralda, and Storey Counties, the most productive being in Storey.

In Washington the larger known occurrences are in Kittitas and Grant Counties.

Other States where diatomite occurrences are known, and in some instances have been worked, are: Arizona, Connecticut, Florida, Idaho, Maryland, Massachusetts, Nebraska, New Hampshire, New Jersey, New Mexico, New York, South Carolina, South Dakota, Utah, Vermont, and Virginia.

TECHNOLOGY

In general, diatomite deposits may be divided into three types: First, those occurring as ancient marine beds, compact and more or less massive, but sometimes with seams and partings of clay; second, deposits adjacent to rivers and lakes, probably for the most part made from fresh-water diatoms of more recent age than those in the marine beds; and third, deposits in lakes and swamps, where the process of accumulation is even now going on. Deposits of the third type usually contain some organic matter, which has not completely decomposed and which in some places is in the form of a mantle of peatlike material.

Most commercial deposits occur at or near the surface, but in a few localities they are mined to considerable depths.

Overburden is usually removed with power shovels, bulldozers, and wheeled scrapers. The formation often is soft enough to be mined by open-pit methods without drilling and blasting. Underground mining, though uncommon, has been used, room-and-pillar and shrinkage stope methods being employed.

Solid diatomite bricks and blocks may be cut directly from the quarry. When such bricks are produced they are inspected, dried at 500° F. in kilns, resized with grinding wheels, again inspected, and packed for shipment.

Pressed brick, made from crushed diatomite, with a binder, is formed in brick presses, dried, and fired in kilns.

Powders for both heat and sound insulation are made by dry-grinding crude diatomite, either with or without calcining before grinding.

The milling of diatomite for filter aid is done by dry-grinding in impact mills, classifying in air classifiers, and calcining. Salts may be mixed with the powdered feed to improve quality of product. Variations in grades are produced by the regulation of grain size, by methods and temperatures of calcination, and by the use of chemical treatments. In grinding for this use, an effort is made to separate the individual diatoms with a minimum breakage of each.

From wet, bog-type deposits, diatomite may be recovered by dredges or dragline scrapers; dewatered by draining or pressing; dried either in drying racks or kilns; calcined; ground; and air-classified.

For nearly all uses, freedom from impurities is important. For filter aid, the type, size, species, and shape of diatoms and the absence of broken and very small diatoms are very important factors. Careful grading of material having the most advantageous shapes and sizes of diatoms is necessary.

For insulation, important properties are low thermal conductivity, low water content, low apparent density, and strong physical structure.

For abrasive and filler uses, particle size must be carefully controlled. Whiteness is important for some filler uses, such as paper and paint, but not for others.

In concrete admixtures, chemical purity, fineness, workability, and water retention are most important.

USES

The value of diatomite for most uses depends upon its physical structure. The hollow cells of the diatoms form dead air spaces, the reason for its low apparent density and its value as an insulating medium for both heat and sound. The great number and the exceedingly small size of these cells or pores are the reasons for its value as a filtering medium.

The principal uses for diatomite may be summarized as follows:

1. **Industrial Filtration.**—Many industrial filtration applications employ diatomite. Products that are filtered with the aid of diatomite include acids and chemicals, polluted waters, petroleum compounds, varnishes, shellacs, waxes, resins, gum solutions, metallurgical and plating solutions, sugar solutions, vegetable oils, animal fats, gelatins, casein, antibiotics, and many others.

2. **Mineral Fillers.**—Mineral fillers have two purposes: To supply bulk and to impart desirable physical properties, such as strength, toughness, elasticity, flexibility, absorptiveness, reduce brittleness, and heat conductivity. Diatomite is used as a filler in asphalt products, rubber and paper products, plastics, in explosives, in insecticides, and other products.

3. **Insulating Material.**—The high percentage of voids and relatively high melting point of diatomite make it a good insulating agent, and it finds many such uses in industrial furnaces and kilns for heat and in building construction for both heat and sound insulation.

4. **Miscellaneous Uses.**—Diatomite also is used as an absorbent (it will absorb 150 to 200 percent of its weight of water without changing its form), as a mild abrasive (in hand soaps and cleansing compounds, in silver and other metal polishes, in dental powder and pastes, and as a friction-creating agent on safety-match heads and boxes), as a support and carrier for catalysts in chemical processes, in ceramics, to furnish silica needed in bodies and glazes, in the manufacture of sodium silicate or "water-glass," and as a base for pigments. When pulverized and mixed with about one-third its weight of tallow or other hard grease and molded into bricks or sticks, it forms the so-called "grease bricks" used on buffing wheels.

The United States is self-sufficient in regard to its diatomite supply. However, currently its production is concentrated in the Pacific Coast States, with a substantial transportation charge to eastern areas of large consumption.

The Oil, Paint and Drug Reporter quoted the following 1953 prices for diatomite: Domestic, bags, c. l., Atlantic Coast, ton, \$52–\$55; California, ton, \$42–\$45; l. c. l., warehouse, \$85–\$90; purified, bags, c. l., Atlantic Coast, ton, \$65; California, \$53; l. c. l., warehouse \$95–\$100; Atlantic coast, \$95–\$100; imported Mexican, white, bags, c. l., Atlantic coast, lb., 3 cents; l. c. l. Atlantic coast, lb., 6 cents.

The average value of diatomite produced in 1951–53 was \$29.97 a ton at this mine.

The larger companies in the diatomite industry have carried on extensive sales-development campaigns that include research into the physical and chemical structure of diatomite and of its many applications in the industrial field and have developed testing techniques for the various types of diatomite. This has resulted in a rapid increase in market demand.

OUTLOOK

The sales efforts of the larger producers of diatomite have built up a demand that is expanding, and further growth is anticipated. If the present trend is maintained, output will soon approximate 350,000 tons annually. The large reserve of diatomite available in the United States and the relative ease with which it can be mined and processed indicate that a much larger market can be served as it grows.

PROBLEMS

In addition to those that concern industry in general the problems confronting the diatomite industry are largely those of developing new uses for the material and distributing a commodity that has a wide application in industry but a limited area of supply.

The diatomite industry has been successful in developing and promoting new uses for the products of its western mines, but the problem of utilizing the smaller or lower grade deposits closer to the eastern markets has not been solved.

There is a need, therefore, to maintain the flow of technical, statistical, and economic information on which efficient management can be based and special emphasis on those aspects that may lead to successful use of eastern deposits is warranted.

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FELDSPAR

By

Brooke L. Gunsallus [1]

FELDSPAR, chiefly used as a ceramic flux, is a common constituent of the earth's crust, but commercial deposits are not abundant.

Summary

Feldspars are anhydrous silicates of aluminum containing potash, soda, and lime in varying proportions. They occur in association with quartz, mica, beryl, and other minerals to form pegmatites and in other combinations to form many of the rocks in the earth's crust. The principal commercial deposits in the United States are in North Carolina, South Dakota, Colorado, New Hampshire, Maine, Virginia, Connecticut, Georgia, and Arizona.

The domestic production of crude feldspar in 1953 was 452,600 long tons valued at $4,594,450. Imports in the same year were 60,501 long tons.

Many feldspar grinders mine their own raw material, either themselves or through affiliated firms. A small part of their supply of crude feldspar, however, is purchased from small mine operators.

Most of the feldspar produced is obtained from coarse-grained pegmatites by hand selecting and hand cobbing. Some deposits are mined without hand sorting, and the material is crushed and the ore hand-picked from a conveyor belt. At other deposits feldspar is recovered entirely by flotation; and in 1953, 35 to 40 percent of the United States production of feldspar was as flotation concentrates.

High-grade potash feldspar reserves have been gradually diminishing for the past 25 years.

Nepheline syenite has replaced feldspar in many ceramic bodies and nepheline syenite, aplite, and blast-furnace slag are used in some glass batches. Deposits of nepheline syenite in the United States usually contain too much iron for ceramic purposes. The entire supply is imported from Canada and in 1953 represented about 19 percent of the apparent consumption of feldspar. Aplite is produced commercially only in Virginia. Blast-furnace slag is a byproduct of the iron and steel industry.

The glass, pottery, and enameling industries consume almost the entire feldspar output of the merchant grinders' mills.

Feldspar is a low-priced mineral and ordinarily cannot be marketed far from its source.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

MINERALOGY

Feldspar is a group name for a series of anhydrous aluminum silicate minerals of potassium, sodium, and calcium present in virtually all igneous rocks (*3*, *4*, *20*).[2] Feldspars usually occur as small grains intimately associated with other minerals, but commercial deposits are obtained from pegmatites in which segregations of the component minerals occur in larger masses.

Feldspars have a vitreous to pearly luster, uneven fracture, and a hardness of 6.0 to 6.5. Colors are generally white, cream, and pink, also milky, clear, buff, brown, red, gray, green, and bluish. The refractive index for the major types ranges from 1.524 to 1.584; the specific gravity ranges from 2.56 to 2.63; the melting points range from 1,110° to 1,532° C.

A mineralogical classification is as follows (*13*):

Potash spar	Orthoclase, microcline.
Soda spar	Albite.
Lime spar	Anorthite.
Lime-soda spar	Oligoclase, andesine, labradorite, bytownite.

The principal commercial feldspar minerals are orthoclase, microline, albite, and anorthite (*20*). None of the minerals in the feldspar group is found pure. The potash feldspars always contain some albite and the soda feldspars some anorthite (*13*). High-potash feldspar is preferred by nearly all branches of the ceramic industry except the glass industry. High-soda feldspar, although not as desirable as potash feldspar, is used in the production of ceramics. The bulk of feldspar used in both container and flat-glass batches is produced by flotation and is a soda-potash product. Potash feldspar is used to a limited extent but only because of geographical and freight-rate considerations. Lime and lime-soda feldspars are used in manufacturing certain kinds of glass.

Most commercial feldspars contain 0.04 to 0.15 percent of iron oxide and traces of magnesia. The free-quartz content varies considerably, figures for commercial shipments ranging from less than 1 to over 25 percent.

Free quartz acts as a diluent in feldspar, decreasing the fluxing power. The free quartz in soda feldspar decreases the vitrification range of ceramic bodies to a greater extent than does that in potash feldspars. The fusion point of a feldspar depends upon the alkalies present and becomes lower as the soda content increases and the potassium oxide content decreases.

PRODUCTION, CONSUMPTION, AND USES

Production, imports, and apparent consumption of crude feldspar, 1925–53, are given in long tons and value in table 1.

Table 2 shows the crude feldspar production by States, 1951–53. The important States in 1953, listed in order of decreasing production, are: North Carolina, South Dakota, and Colorado.

Feldspar concentrates obtained by flotation of feldspathic rocks and sands are listed by the Bureau of Mines under crude-feldspar production for statistical purposes.

World feldspar production, 1949–53, is given in table 3.

Many merchant grinders also mine feldspar, either themselves or through affiliated firms. A low percentage of the supply of crude feldspar, however, is obtained from small operators who sell their product principally to the merchant mills.

Most consumers of feldspar buy material ground, sized, and ready for use in their products. Some pottery, enamel, and soap manufacturers, however, purchase all or part of their requirements crude and crush or grind it to their own specifications in their own mills. Some Canadian crude feldspar is purchased directly by United States consumers. Feldspar for glass production is usually ground to about 20-mesh. The remainder is finely ground, most of it between 100- and 200-mesh. Table 4 shows the ground feldspar sold by merchant mills in the United States, 1951–53, by States, and table 5 the quantity of domestic and Canadian feldspar ground and sold by merchant mills in the United States, 1944–48 (average) and 1949–53.

The glass, pottery, and enamel industries consume 99 percent of all ground feldspar (see table 6). In 1953 glass consumed 55 percent, pottery 39 percent, and enamel 3 percent; the remaining 3 percent was consumed in other industries, including soaps and abrasives.

Manufacturers of artificial teeth annually consume a small tonnage of very carefully selected crude feldspar, which must be free from grit and is marketed at a considerable premium over No. 1 grade commercial fieldspar.

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

TABLE 1.—*Feldspar (crude) production, imports, and apparent consumption in the United States, 1925–53*

Year	Production		Imports		Apparent domestic consumption	
	Long tons	Value	Long tons	Value	Long tons	Value
1925	185, 706	$1, 315, 654	24, 994	$203, 524	210, 700	$1, 519, 178
1926	209, 989	1, 607, 101	29, 941	251, 896	239, 930	1, 858, 997
1927	202, 497	1, 424, 755	27, 424	206, 856	229, 921	1, 631, 611
1928	210, 811	1, 418, 975	27, 857	224, 920	238, 668	1, 643, 895
1929	197, 699	1, 276, 640	29, 927	241, 852	227, 626	1, 518, 492
1930	171, 788	1, 066, 636	21, 006	167, 157	192, 794	1, 233, 793
1931	147, 119	861, 059	10, 719	95, 096	157, 838	956, 155
1932	104, 715	539, 641	1, 872	14, 346	106, 587	553, 987
1933	150, 633	778, 826	3, 239	21, 877	153, 872	800, 703
1934	154, 188	853, 136	9, 744	67, 258	163, 932	920, 394
1935	189, 550	1, 005, 021	8, 937	56, 175	198, 487	1, 061, 196
1936	244, 726	1, 303. 090	10, 786	68, 198	255, 512	1, 371, 288
1937	268, 532	1, 383, 249	12, 956	91, 885	281, 488	1, 475, 134
1938	196, 119	895, 081	7, 651	56, 126	203, 770	951, 207
1939	253, 466	1, 112, 857	7, 460	52, 141	260, 926	1, 164, 998
1940	290, 763	1, 271, 995	15, 522	80, 274	306, 285	1, 352, 269
1941	338, 860	1, 519, 456	11, 253	73, 236	350, 113	1, 592, 692
1942	316, 166	1, 546, 702	9, 525	69, 797	325, 691	1, 616, 499
1943	308, 180	1, 646, 277	10, 758	83, 073	318, 938	1, 729, 350
1944	327, 408	1, 813, 937	11, 686	95, 956	339, 094	1, 909, 893
1945	373, 054	2, 021, 529	14, 924	114, 917	387, 978	2, 136, 446
1946	508, 380	2, 594, 099	16, 365	127, 654	524, 745	2, 721, 753
1947	459, 910	2, 410, 940	16, 685	124, 587	476, 595	2, 535, 527
1948	460, 713	2, 564, 387	31, 047	219, 785	491, 760	2, 784, 172
1949	369, 378	2, 278, 441	15, 826	107, 925	385, 204	2, 386, 366
1950	407, 925	2, 558, 390	12, 367	84, 136	420, 292	2, 642, 526
1951	400, 439	2, 815, 587	17, 128	146, 565	417, 567	2, 962, 152
1952	420, 831	3, 696, 018	5, 576	53, 016	426, 407	3, 749, 034
1953	452, 600	4, 594, 450	5, 901	60, 501	458, 501	4, 654, 951

TABLE 2.—*Crude feldspar sold or used by producers in the United States, 1951–53, by States*

State	1951		1952		1953	
	Long tons	Value	Long tons	Value	Long tons	Value
Colorado	50, 451	$283, 153	38, 268	$224, 385	43, 508	$267, 642
Connecticut	13, 811	107, 083	10, 929	87, 432	9, 829	63, 049
Maine	19, 273	154, 695	18, 644	147, 371	17, 637	117, 090
New Hampshire	(1)	(1)	(1)	(1)	28, 961	286, 069
North Carolina	166, 361	1, 230, 404	240, 364	2, 416, 031	268, 042	3, 290, 495
South Dakota	48, 559	290, 520	40, 163	220, 954	50, 601	321, 026
Texas	(1)	(1)	2, 600	31, 200	---------	---------
Virginia	30, 979	232, 099	(1)	(1)	(1)	(1)
Other States [2]	71, 005	517, 633	69, 863	568, 645	34, 022	249, 079
Total	[3] 400, 439	$2, 815, 587	[3] 420, 831	$3, 696, 018	[3] 452, 600	$4, 594, 450

[1] Included with other States to avoid disclosure of individual company operations.
[2] Includes Arizona, California, Georgia (1951), New Hampshire (1951–52), Texas (1951), Virginia (1952–53), and Wyoming (1953).
[3] Flotation concentrates included in total.

TABLE 3.—*World production of feldspar by countries,*[1] *1944–48 (average) and 1949–53* [2]

[Metric tons]

(Compiled by Helen L. Hunt)

Country [1]	1944–48 (average)	1949	1950	1951	1952	1953
North America:						
Canada (sales)	32, 650	33, 518	32, 248	36, 967	18, 386	18, 655
United States (sold or used)	432, 729	375, 307	414, 472	406, 866	427, 585	459, 864
South America:						
Argentina	4, 820	(3)	(3)	(3)	(3)	(3)
Brazil [4]	63	11, 111	12, 000	(3)	(3)	----------
Chile	[5] 318	125	871	1, 200	[4] 1, 000	[4] 1, 000
Peru	[5] 186	300	----------	131	----------	----------
Uruguay	1, 352	811	710	675	898	792
Europe:						
Austria	837	1, 912	3, 909	3, 751	2, 578	1, 353
Czechoslovakia [4]	7, 723	(3)	(3)	(3)	(3)	(3)
Finland	4, 690	10, 074	8, 000	8, 198	9, 790	9, 327
France	30, 724	47, 514	47, 727	66, 000	65, 000	60, 000
Germany, West	28, 674	48, 262	76, 712	71, 531	119, 291	116, 023
Italy	7, 168	13, 522	18, 071	29, 144	25, 476	23, 602
Norway	14, 564	27, 482	23, 695	31, 118	29, 297	[4] 23, 000
Portugal	974	1, 240	----------	470	700	60
Spain (quarry)[6]	2, 655	396	1, 650	1, 760	----------	----------
Sweden	26, 525	38, 959	36, 031	41, 072	47, 871	(3)
Asia:						
India	948	863	1, 800	3, 439	2, 052	(3)
Israel	35	----------	----------	----------	----------	----------
Japan [7]	11, 555	20, 055	13, 187	26, 528	24, 194	25, 078
Africa:						
Egypt	29	----------	----------	----------	----------	----------
Eritrea	[8] 167	200	----------	----------	----------	----------
Kenya	50	20	----------	----------	----------	----------
Madagascar	9	----------	----------	----------	----------	----------
Southern Rhodesia	----------	----------	3, 520	1, 148	----------	----------
Union of South Africa	1, 423	3, 549	6, 001	3, 343	7, 479	5, 568
Australia [9]	8, 047	10, 902	13, 276	15, 080	13, 807	6, 884
Total (estimate)	622, 000	660, 000	720, 000	770, 000	820, 000	810, 000

[1] In addition to countries listed, feldspar is produced in China, Rumania, and U. S. S. R. but data are not available; no estimates are included in the total.

[2] This table incorporates a number of revisions of data published in previous Feldspar chapters of Minerals Yearbook.

[3] Data not available; estimate by senior author included in total.

[4] Estimate.

[5] Average for 1945–48.

[6] In addition, the following quantity of feldspar is reported as ground, but there are no crude production data to support this ground figure: 1949, data not available; 1950, 8,254 tons; 1951, 11,043 tons; 1952, 10,359 tons; 1953, 10,663 tons.

[7] In addition, the following quantities of aplite and other feldspathic rock were produced: 1949, 50,943 tons; 1950, 45,679 tons; 1951, 59,919 tons; 1952, 17,124 tons; 1953, data not available.

[8] Average for 1946–48.

[9] Includes some china stone.

Names and addresses of merchant grinders of feldspar in the United States in 1953 are listed below:

Abingdon Potteries, Inc., 801 West Main Street, Abingdon, Ill.
Bell Minerals Co., West Paris, Maine.
Clinchfield Sand Feldspar Corp., 413 Washington Avenue, Towson 4, Baltimore, Md.
Consolidated Feldspar Dept. of International Mineral & Chemical Corp., Erwin, Tenn.
Del Monte Properties Co., Box 150, Pacific Grove, Calif.
Eureka Mica Mining & Milling Co., 190 West State Street, Newark, N. J.
Feldspar Flotation, Inc., Spruce Pine, N. C.
Feldspar Milling Co., Burnsville, N. C.
Golding-Keene Co., Box 456, Keene, N. H.
Golding-Keene Co., Trenton Feldspar Plant, Trenton 8, N. J.
J. F. Morton, Inc., P. O. Box 232, Trenton 2, N. J.
North Carolina Feldspar Corp., Erwin, Tenn.
Topsham Feldspar Co., Box 34, Topsham, Maine.
Western Feldspar Milling Co., Box 671, Salida, Colo.
Worth Spar Co., P. O. Box 763, Middletown, Conn.

TECHNOLOGY

Feldspar and feldspathic rocks and sands are mined by three methods, depending on the quality of the deposit (*13*). Most of the feldspar is obtained from coarse-grained pegmatites by hand selecting and hand cobbing. Some deposits are mined without hand sorting, and the material is crushed and the ore handpicked from a conveyor belt. About 35 to 40 percent of the salable feldspar is obtained by flotation. Ore for flotation feed is mined by

TABLE 4.—*Ground feldspar sold by merchant mills*[1] *in the United States, 1951–53, by States*

State	1951			1952			1953		
	Active mills	Short tons	Value	Active mills	Short tons	Value	Active mills	Short tons	Value
Arizona	(2)	(2)	(2)	(2)	(2)	(2)	1	60, 204	$766, 832
Colorado	(2)	(2)	(2)	(2)	(2)	(2)	2		
Connecticut	2	25, 740	$528, 246	2	19, 109	$386, 191	2	11, 647	226, 300
New Jersey	1			1			(2)	(2)	(2)
Georgia	1	47, 755	668, 347	-----	-----	-----	-----	-----	-----
Virginia	2			(2)	(2)	(2)	(2)	(2)	(2)
Maine	3	20, 504	376, 258	3	16, 791	317, 365	3	17, 901	354, 639
New Hampshire	2	34, 149	716, 660	2	28, 592	605, 342	2	32, 397	700, 653
New York	1			1			1		
North Carolina	3	197, 704	2, 886, 655	3	270, 775	3, 714, 084	2	272, 059	3, 891, 684
Tennessee	1			1			1		
Texas	-----	-----	-----	1	2, 000	30, 000	-----	-----	-----
Other States [3]	7	128, 763	1, 756, 712	10	121, 653	1, 659, 499	8	69, 668	1, 208, 581
Total	23	454, 615	6, 932, 878	24	458, 920	6, 712, 481	22	463, 876	7, 148, 689

[1] Excludes potters and others who grind for consumption in their own plants.

[2] Included with other States in order to avoid disclosure of individual company operations.

[3] Includes (number of active mills in parentheses) Arizona (1 in 1951–52), California (1 in 1951, 2 in 1952–53), Colorado (2 in 1951–52), Illinois (1), New Jersey (1 in 1953), South Dakota (2), and Virginia (2 in 1952–53).

TABLE 5.—*Ground feldspar sold by merchant mills*[1] *in the United States, 1944–48 (average) and 1949–53*

Year	Active mills	Domestic feldspar			Canadian feldspar			Total	
		Short tons	Value		Short tons	Value		Short tons	Value
			Total	Average		Total	Average		
1944–48 (average)	28	422, 797	$4, 851, 616	$11. 48	14, 059	$304, 279	$21. 64	436, 856	$5, 155, 895
1949	27	369, 824	5, 212, 246	14. 09	16, 883	396, 855	23. 51	386, 707	5, 609, 101
1950	23	429, 787	5, 952, 019	13. 85	16, 736	391, 600	23. 40	446, 523	6, 343, 619
1951	23	441, 816	6, 633, 378	15. 01	12, 799	299, 500	23. 40	454, 615	6, 932, 878
1952	24	448, 839	6, 473, 203	14. 42	10, 081	239, 278	23. 74	458, 920	6, 712, 481
1953	22	454, 692	6, 909, 177	15. 20	9, 184	239, 512	26. 08	463, 876	7, 148, 689

[1] Excludes potters and others who grind for consumption in their own plants.

mass production methods, crushed, and wet-ground in rod mills to about 20-mesh (*10*).

Most finely ground feldspar is ground dry in silex-lined (or equivalent) pebble mills to keep iron contamination to a minimum. Sizing may be controlled by screening or air classification. The product usually is passed over a high-intensity magnetic separator to remove small quantities of iron and iron-bearing minerals, such as tourmaline and garnet.

Flotation feed is treated in a number of successive operations to separate feldspar and by-product quartz and mica from such impurities as pyrite, tourmaline, free iron, and possibly others. Feldspar produced by this method is remarkably pure and uniform in composition.

Alaskite rock is being used as raw material at several flotation plants in North Carolina (*15*). The quantity of feldspar obtained from the alaskite rock averages about 30 to 40 percent, by weight, of the crude ore. The feldspar product contains about 5 percent K_2O and 4 percent Na_2O.

The reagents in general use in feldspar flotation are hydrofluoric acid, sulfuric acid, pine oil, and various soap mixtures.

RESERVES

For over a quarter century, domestic reserves of high-grade potash spar (9 to 11 percent K_2O) have been gradually diminishing. At present 3 companies—1 in Virginia, 1 in North Carolina, and 1 in Georgia—are producing spar of this

TABLE 6.—*Ground feldspar sold by merchant mills in the United States 1937–53, by uses*

[Short tons]

Year	Glass	Pottery	Enamel	Other	Total
1937 [1]	142,028	102,346	25,111	9,787	279,272
1938	117,800	74,035	19,395	3,284	214,514
1939	138,336	87,209	28,356	5,293	259,194
1940	149,623	104,586	26,420	5,084	285,713
1941	182,878	127,140	34,841	9,558	354,417
1942	195,601	106,081	13,899	12,205	327,786
1943	214,668	97,887	7,147	16,108	335,810
1944	220,734	106,341	8,464	7,662	343,201
1945	249,927	111,695	13,755	6,351	381,728
1946	289,559	154,340	22,500	3,800	470,199
1947	266,720	183,829	24,159	7,992	482,700
1948	270,065	202,905	25,282	8,199	506,451
1949	199,852	158,218	25,351	3,286	386,707
1950	212,481	197,817	33,037	3,188	446,523
1951	197,483	231,725	21,778	3,629	454,615
1952	251,489	179,469	21,809	6,153	458,920
1953	253,596	179,323	14,383	16,574	463,876

[1] New classification for ceramic uses adopted in 1937 was as follows: Glass, pottery, enamel, and other ceramics. Except for glass, figures for 1937 are not directly comparable with those for earlier years.

classification. The one in Georgia uses flotation. The remainder of the domestic feldspar production is mostly lower-grade potash feldspar and some soda feldspar. Imports of feldspar from Canada are high-potash feldspar, but no information is available concerning the extent of the reserves. World potash-feldspar reserves are probably adequate; but, because of the relatively low value of the ore and high transportation charges, feldspar cannot be transported great distances.

SUBSTITUTES

As in many nonmetallic industries, the economics of the feldspar industry cannot be considered without considering several competitive minerals. In its various ceramic applications, there are a number of direct or partial substitutes for feldspar, including nepheline syenite, aplite, talc, pyrophyllite, and blast-furnace slag.

Nepheline syenite is a quartz-free crystalline rock consisting largely of nephelite, albite, and microcline (*17*). Impurities may be iron-bearing minerals, such as mica and magnetite, and other minerals, such as zircon and corundum. Originally used almost entirely in glass manufacture, substantial quantities now are consumed in making pottery. Nepheline syenite occurs in New Jersey, Arkansas, and other localities in the United States, but all the domestic material found thus far in any appreciable tonnage contains too much iron for ceramic purposes.

The American Nepheline, Ltd., Lakefield, Ontario, Canada, is the only producer of crude and ground nepheline syenite in the Western Hemisphere. In 1950 the company dismantled its grinding mill at Rochester, N. Y., and moved it to Ontario, Canada. As shown by table 7, except for minor quantities of crude, only ground nepheline syenite is imported into the United States for consumption. Imports and domestic consumption increased from 1944 through 1953, owing mainly to its increased use in the glass-container industry, which has grown phenomenally in the past few years. Small increases in consumption have occurred in other branches of the ceramic industry. If the present competitive position of nepheline syenite and feldspar remains about the same it is believed that the maximum consumption of nepheline syenite for many years to come will not greatly exceed 100,000 short tons per year. The alumina content of nepheline syenite always is taken into consideration by the purchaser and should not be less than about 22 percent.

Aplite is a rock composed principally of albite, zoisite, sericite, and small quantities of other minerals (*3*). It is used in manufacturing containers and flat glass and is a low-cost source of alumina where the color of the glass is not a factor (*13*). It replaces feldspar in the glass batch.

The tonnage of aplite produced in the United States decreased progressively from 1949 through 1952, except in 1950. Production in 1953 also increased. There are only two aplite producers in the United States, and the Bureau of Mines cannot publish production data.

Talc acts as a flux to reduce the quantity of feldspar needed to produce the desired strength in ceramic bodies.

Pyrophyllite is used as a source of alumina in enamels and in some wall-tile bodies as a

TABLE 7.—*Nepheline syenite imported for consumption in the United States, 1944–48 (average) and 1949–53*

(U. S. Department of Commerce)

Year	Crude		Ground		Year	Crude		Ground	
	Short tons	Value	Short tons	Value		Short tons	Value	Short tons	Value
1944–48 (ave.)	50, 126	$189, 456	1, 934	$30, 692	1951	-------	---------	65, 773	$936, 256
1949	41, 215	167, 567	18, 779	248, 224	1952	4	$125	68, 398	984, 050
1950	8, 966	36, 453	54, 242	703, 008	1953	181	659	89, 195	1, 308, 058

substitute for part or all of the flint and feldspar.

Blast-furnace slag is used as a cheap source of alumina and for this reason competes with aplite and feldspar in glass batches where color is not a prime consideration.

PRICES

Price quotations for crude feldspar do not appear in the trade press. Average values are computed from the returns of producers reporting their output annually to the Bureau of Mines. In 1953 the average selling price per long ton for all feldspar mined in the United States was $10.15 compared with $8.78 in 1952 and $7.03 in 1951.

The average realization per short ton for ground feldspar in 1953 was $15.41, a 5-percent increase over 1952 and a 1-percent increase over 1951. Of the large producing States, the one having the highest average value per short ton was New Jersey, $26.85 ($27.85 for 1952), followed by New York, $26.08 ($23.74 for 1952), and Illinois, $21.27 ($21.49 for 1952). North Carolina, by far the largest producer, realized only $14.22 per short ton in 1953. The State reporting the lowest average value per short ton in 1953 was Colorado, $11.65 ($10.76 for 1952).

The value of ground feldspar reported for nonproducing States includes freight on crude material.

Quotations on ground feldspar appearing in E&MJ Metal and Mineral Market Reports for December 1953 were the same as in each previous year, starting with 1949 as follows: North Carolina, bulk carlots, 200-mesh, $18.50 per short ton; 325-mesh, $22.50; glass feldspar, No. 18, $12.50; and semigranular, $11.75 (add $3.00 per ton to bulk quotation for bags and bagging). Quotations on Virginia feldspar were not listed in E&MJ for 1952. The following prices were given for 1951: No. 1, 230-mesh, $18.50 per ton, and 200-mesh, $17.50; No. 17, glassmakers' feldspar, $11.75, and No. 18, $12.50. Enamelers' feldspar was listed at $15 to $18.

FOREIGN TRADE[3]

Crude-feldspar imports for consumption in 1953 totaled 5,901 long tons (all from Canada) valued at $60,501. Compared with 1952, there was a 6-percent increase in tonnage and a 14-percent increase in value. This was the lowest imported tonnage of feldspar since 1933.

According to reports by the merchant grinders, ground feldspar exported from the United States in 1953 totaled 2,989 short tons, a 13-percent decrease from 1952. Countries of destination were Canada, Mexico, Cuba, Belgium, and Puerto Rico.

TABLE 8.—*Feldspar imported for consumption in the United States, 1947–53*

(U. S. Department of Commerce)

Year	Crude		Ground		Year	Crude		Ground	
	Long tons	Value	Long tons	Value		Long tons	Value	Long tons	Value
1947	16, 685	$124, 587	------	-------	1951	17, 128	$146, 565	(1)	$26
1948	31, 047	219, 785	(1)	$328	1952	5, 576	53, 016	------	------
1949	15, 826	107, 925	------	-------	1953	5, 901	60, 501	------	------
1950	12, 367	84, 136	------	-------					

(1) Less than 1 ton.

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[3] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

OUTLOOK

The feldspar-mining industry has depended almost entirely on the ceramic industry as a market for its product. The ceramic industry, in return, has demanded a product of high purity, uniform quality, and low price. Twenty-five years ago almost the entire feldspar output was derived from pegmatites in which the feldspar occurs in crystals or masses large enough to permit the selection of clean lump material by hand-cobbing methods and to a smaller extent from mined rock where a method of coarse crushing and hand sorting from a picking belt could be utilized. However, with progressive exhaustion of most of the large, economically situated feldspar-rich pegmatites, improved beneficiating methods (including flotation and magnetic separation) are being employed to recover feldspar from rock considered waste a few years ago. In 1953 the feldspar concentrates resulting from the processing of feldspathic rocks and sands by flotation composed about 35 to 40 percent of the total crude feldspar produced; this percentage is expected to increase materially in the near future. The commercial application of froth flotation to the feldspar-mineral family has completely changed the outlook for the feldspar industry.

The widespread development of the ceramic industry, the high cost of transporting feldspar long distances, and the prospect of diminishing domestic supplies of low-cost, high-grade potash feldspar have caused all branches of the ceramic industry to make a considerable effort to develop and use low-potash domestic feldspars and substitute materials.

Feldspar producers using flotation and producing feldspar exclusively for the glass industry are directly affected by nepheline syenite imports. However, there are indications that future imports of nepheline syenite will not greatly exceed the present tonnage unless favorable changes in its competitive position are brought about by lower freight rates to consuming industries, by a change in the ratio of the selling price of nepheline syenite and competitive products, or an unexpectedly large increase in the demand for glass and other ceramic products.

Almost all feldspar imports from Canada, all classified as high-potash, have entered the United States by railroad, but it is reported that plans are being made to ship by boat to Atlantic seaboard ports. It may be possible to upgrade some domestic feldspars by blending these high-potash imports.

PROBLEMS

The rapid expansion of the ceramic industry has accentuated the growing shortage of high-potash feldspar. Only one new area—Georgia—containing deposits of high-potash feldspar has been developed in recent years. New deposits have been discovered in Canada that may lead to exports to the United States.

The ceramic industry expansion has been on a nationwide basis, so the demand for feldspar comes from all sections of the United States; transportation costs are a major problem in some areas.

Even though there has been considerable information published on investigations and evaluation of feldspar in South Dakota (*8*), North Carolina, and several New England States (*5*, *21*), much work remains to be done in determining the potential commercial possibilities of pegmatites containing feldspar (*20*). The interrelationship of feldspar and other pegmatite minerals is not well understood.

An area where reserves are especially low in relation to potential demand and where investigations and evaluation are greatly needed embraces California and the Pacific Northwest.

Information on advantageously located marginal deposits is needed to determine whether they can be brought into commercial production by more efficient mining and milling, by beneficiation, or by utilization of byproducts or coproducts.

Some pegmatites contain considerable potash feldspar associated with soda feldspar, quartz, beryl, and mica. The cost of separating the feldspar from such pegmatites is often prohibitive, unless the other components can be recovered and marketed. It may be possible to apply nonselective mining and simplified processing methods to produce standardized blends of feldspar, quartz, and muscovite as an additive to clays and shales in the manufacture of heavy clay products. Such processing techniques would not exclude the recovery of any heavy minerals, like columbite, that may be present. The recovery of marketable feldspar from pegmatites that are being mined only for beryl and mica offers important possibilities. Research and development along these conservational lines would not only help promote marginal feldspar deposits but also aid development of associated minerals.

Although certain electrostatic or similar processes do allow the separation of potash feldspar and soda feldspar from pegmatites, the cost is prohibitive. If a commercial separation process is developed, it will be very beneficial to the industry.

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FERROALLOYS, GENERAL

By

Robert W. Geehan[1]

THE FERROALLOY industry normally exceeds many better known industries in dollar volume of products shipped. In 1951 the value of ferroalloys shipped was larger than the combined value of the lead and zinc produced from domestic ores. In spite of this high-ranking position, probably less is known of its organization and operating procedures than of many less important branches of the mineral industry.

Summary

Many elements other than iron and carbon have found substantial utilization in the manufacture of irons and steels; some impart special properties, others are needed in the manufacturing processes. In many instances, these elements are added to irons and steels in the form of a ferroalloy "intermediate product" that has been produced from ores or oxides.

A ferroalloy has been defined as "iron so rich in some element other than carbon that it is used as a vehicle for introducing that element in the manufacture of iron and steel." Usually the iron content is unimportant, and some products that contain little or no iron are now listed as ferroalloys.

This chapter deals with aspects applicable to the ferroalloys group as a whole; some of the elements used are so important that individual chapters discussing them have been prepared.

Soon after the advent of the alloy age in the steel industry, about the turn of the century, it became evident that ferroalloy production processes were diverging from iron and steel practice. At present many ferroalloys are manufactured in special equipment, such as submerged-arc furnaces and vacuum furnaces; however, some are produced in blast furnaces that are essentially the same as those used in manufacturing pig iron. Control of the carbon content has become very important in certain products, particularly ferrochromium and ferromanganese.

Production and capacity to produce ferroalloys are related to similar elements of the steel industry; this relationship is expected to continue. Changes in specifications and manufacturing methods may follow the use of lower grade ores. Many problems of the ferroalloy industry are associated with raw materials and new products. Because of the highly competitive nature of the ferroalloy industry, producers are reluctant to release information on production practices, and very little information of this sort appears in the literature.

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[1] Assistant Chief, Division of Minerals, Bureau of Mines.

BACKGROUND

TECHNOLOGY

Early ironmasters were able to produce some natural alloys from iron ores that contained the needed alloying element or from meteorites that contained iron and nickel. As the metallurgical art developed, the need for ferroalloys as an intermediate stage between the ore of the alloying element and the final alloy became apparent. A ferroalloy bears much the same relationship to steel as does pig iron, as both are the intermediate stage between raw material and final product. This being true, it is not strange to find blast furnaces in use to produce ferromanganese, ferrosilicon (silvery pig iron), and spiegeleisen as well as pig iron. The process differs from the manufacture of pig iron in factors such as charge composition, bosh temperatures, and air blast. Because of the smaller tonnage required, the older pig-iron furnaces are often used in the production of ferroalloys.

Most other ferroalloys are manufactured in electric furnaces using carbon, silicon, or aluminum as the reducing agent. Details of processes used for particular alloys vary, and manufacturers generally have declined to document their practice. Electric furnaces used include:

1. The open-top, submerged-arc type, which is nontiltable, charged continuously from the top, and tapped near the bottom in much the same manner as a blast furnace.
2. Direct-arc furnaces, tiltable, open side, charged more or less continuously from the side and, in some instances, equipped with a rotating hearth.
3. Direct-arc furnaces, steel-furnace type, with cover, charged from top with cover removed.
4. Knockdown furnaces used for production of ferroalloys with a melting point so high that tapping is not practicable. Ferrotungsten is produced in equipment of this type.

Other ferroalloys, particularly the relatively pure metals, are produced by the thermite process in which a highly active metal, usually aluminum powder or shot, is mixed with ore or prepared oxides of the alloying element. The mixture is ignited, and the oxygen of the alloying element transfers to the reducing metal with a strong evolution of heat which melts the charge.

Very pure alloying elements are produced increasingly by electrolytic deposition. The raw materials are dissolved in an electrolyte from which the metals are plated on special cathodes. Processes for the production of electrolytic manganese and chromium were developed by the Bureau of Mines and have been adapted to commercial production.

GEOGRAPHIC DISTRIBUTION OF INDUSTRY

The domestic industry is centered in Pennsylvania, Ohio, and New York (60 percent of tonnage, 68 percent of value in 1953). Other States that produced more than 100,000 tons in 1953 are West Virginia, Iowa, Alabama, and Tennessee.

Canada, France, Germany, Japan, Norway, Sweden, and United Kingdom are the leading foreign producers of ferroalloys.

FOREIGN TRADE

The value of United States imports for consumption in 1953 included: Ferrochromium $10,397,933, ferromanganese $27,181,009, ferrosilicon $834,712, ferrotitanium $114,507, and ferrotungsten $1,686,690. The value of exports included: Ferrochromium $285,900, ferromanganese $389,064, ferromolybdenum $548,502, ferrophosphorus $1,147,707, ferrosilicon $287,539, ferrotitanium $48,722, ferrotungsten $122,949, ferrovanadium $296,157, and all other $256,029.

TRANSPORTATION

Most ferroalloy plants are near consuming centers, and movement of finished products has seldom created a problem. However, movement of raw materials, often from foreign sources, has presented serious difficulties (chromium and manganese) during emergency periods because of the lack of adequate transportation facilities between foreign mines and the port of shipment and, at times, a shortage of vessels.

STATISTICAL REPORTING

The Bureau collects and publishes production and shipment data on manufactured ferroalloys and consumption data on ferrochromium, ferromanganese, and ferrosilicon. However, some producers report special combinations grouped in such a way as to prevent analysis by alloying elements; in other instances, the Bureau publishes combined tonnages so as to prevent release of individual company records. The value of Bureau statistics would be enhanced if all

TABLE 1.—*Producers of ferroalloys in the United States in 1953*

Producer	Plant	Alloy
American Agricultural Chemical Co	South Amboy, N. J	Ferrophosphorus (byproduct).
Anaconda Copper Mining Co	Anaconda, Mont	Ferromanganese.
	Black Eagle, Mont	
Bethlehem Steel Co	Johnstown, Pa	Do.
Chromium Mining & Smelting Co., Ltd.	Riverdale, Ill	Exothermic ferrochromium and ferrochromium-silicon.
Climax Molybdenum Co	Langeloth, Pa	Ferromolybdenum, calcium molybdate, molybdenum silicide, ammonium molybdate, molybdenum oxide,[1] molybdenum trioxide, sodium molybdate, cobalt molybdenum, molybdenum sulfide.
Electro Metallurgical Co	Alloy, W. Va	Ferromanganese,[1] silicomanganese,[1] manganese metal, ferrosilicon,[1] silicon metal, magnesium-ferrosilicon, ferrochromium,[1] exothermic ferrochrome and ferrosiliconchrome, ferrochrome-silicon, chromium metal, ferroboron, calcium-silicon, calcium metal, ferrotitanium, manganese-nickel-titanium, ferrotungsten, tungsten metal, ferrovanadium, zirconium alloys, ferrocolumbium, ferrotantalum-columbium.
	Ashtabula, Ohio	
	Glen Ferris, W. Va	
	Holcomb Rock, Va	
	Marietta, Ohio	
	Niagara Falls, N. Y	
	Portland, Oreg	
	Sault Ste. Marie, Mich	
	Sheffield, Ala	
General Abrasive Co., Inc	Niagara Falls, N. Y	Ferrosilicon (byproduct).
Globe Iron Co	Jackson, Ohio	Silvery pig iron, ferrosilicon.
Hanna Furnace Corp	Buffalo, N. Y	Silvery pig iron.
Inland Steel Co	East Chicago, Ind	Spiegeleisen.
Jackson Iron & Steel Co	Jackson, Ohio	Silvery pig iron.
Kaiser Aluminum & Chemical Corp	Permanente, Calif	Ferrosilicon.
Keokuk Electro-Metals Co	Keokuk, Iowa	Ferrosilicon, silvery pig iron, silicon metal.
	Wenatchee, Wash	
E. J. Lavino & Co	Reusens, Va	Ferromanganese.
	Sheridan, Pa	
Metal & Thermit Corp	Carteret, N. J	Ferrotitanium, chromium metal.
Molybdenum Corp. of America	Washington, Pa	Ferromolybdenum, molybdic oxide, ferrotungsten, manganese boride, ferroboron.
Monsanto Chemical Co	Anniston, Ala	Ferrosilicon. Ferrophosphorus (byproduct).
	Columbia, Tenn	
	Soda Springs, Idaho	
Montana Ferro-Alloys Co	Woodstock, Tenn	Ferrochromium, ferrochrome silicide.
New Jersey Zinc Co	Palmerton, Pa	Spiegeleisen, ferromanganese.
Ohio Ferro-Alloys Co	Brilliant, Ohio	Ferrochromium, ferrochrome-silicon, ferrosilicon. Ferrosilicon, ferrosilicon-boron, silicon-manganese-aluminum, silicon metal, ferromanganese,[1] silicomanganese, manganese sulfide. Ferrochromium,[1] ferrochrome silicon.
	Philo, Ohio	
	Tacoma, Wash	
Oldbury Electro-Chemical Co	Niagara Falls, N. Y	Ferrophosphorus (byproduct).
Pacific Northwest Alloys, Inc	Mead, Wash	Ferrosilicon, ferrochromium.
Pittsburgh Metallurgical Co	Calvert City, Ky	Ferrosilicon, silicon metal, ferrochromium, ferrochrome silicon, silicomanganese.
	Niagara Falls, N. Y	
	Charleston, S. C	
Reading Chemical Co	Wyomissing, Pa	Ferrotungsten.
Shea Chemical Corp	Columbia, Tenn	Ferrophosphorus.
Tennessee Products & Chemical Corp	Chattanooga, Tenn	Ferrosilicon,[1] ferromanganese,[1] silicomanganese.[1]
Tennessee Valley Authority	Muscle Shoals, Ala	Ferrophosphorus (byproduct).
Tenn-Tex Alloy-Chemical Corp	Houston, Tex	Ferromanganese, ferrosilicon, spiegeleisen.
Titanium Alloy Mfg. Div. of National Lead Co.	Niagara Falls, N. Y	Ferrotitanium, ferrocarbon-titanium.
U. S. Steel Corp. subsidiaries	Clairton, Pa	Ferromanganese, spiegeleisen.
	Etna, Pa	
	Duquesne, Pa	
	Ensley, Ala	
Vanadium Corp. of America	Cambridge, Ohio	Ferrotitanium, ferrocarbon-titanium, ferrovanadium, ferrosilicon,[1] silicon metal, ferrochromium,[1] ferrochrome silicon, aluminum-silicon-iron, boron alloys, alumino vanadium, ammonium meta vanadate, titanium aluminum, vanadium metal.
	Graham, W. Va	
	Niagara Falls, N. Y	

[1] Briquets also produced.

TABLE 1.—*Producers of ferroalloys in the United States in 1953*—Continued

Producer	Plant	Alloy
Victor Chemical Works	Mount Pleasant, Tenn. Silver Bow, Mont. Tarpon Springs, Fla.	Ferrophosphorus (byproduct).
Virginia-Carolina Chemical Corp.	Charleston, S. C. Nichols, Fla.	Do.
Westvaco Chemical Div. of Food Machinery & Chemical Corp.	Pocatello, Idaho	Do.

TABLE 2.—*Ferroalloys produced and shipped from furnaces in the United States, 1952–53*

Alloy	1952			1953		
	Production (short tons)	Shipments		Production (short tons)	Shipments	
		(Short tons)	Value		(Short tons)	Value
Ferromanganese	758, 721	738, 088	$133, 996, 006	907, 533	900, 110	$185, 192, 588
Ferrosilicon	781, 888	760, 981	84, 095, 168	808, 605	772, 697	86, 096, 619
Ferrochromium [1]	248, 421	242, 572	88, 937, 103	284, 793	253, 636	97, 849, 597
Ferrophosphorus	50, 850	53, 960	2, 672, 731	54, 361	36, 795	1, 790, 109
Ferrotitanium	12, 051	11, 577	13, 328, 409	11, 954	11, 824	13, 757, 060
Ferrovanadium						
Ferrotungsten	33, 372	33, 366	52, 019, 126	23, 570	23, 167	40, 496, 355
Ferromolybdenum						
Other molybdenum products						
Spiegeleisen	176, 628	177, 074	26, 586, 156	226, 376	192, 656	32, 855, 142
Silicomanganese						
Manganese briquets						
Other ferroalloys [2]	20, 768	20, 848	7, 224, 269	19, 094	18, 073	8, 159, 379
Total	2, 082, 699	2, 038, 466	408, 858, 968	2, 336, 286	2, 208, 958	466, 196, 849

[1] Includes ferrochrome-silicon.

[2] Ferrocolumbium, ferroboron, zirconium-ferrosilicon ,and miscellaneous ferroalloys.

the information obtained could be published; however, it does not seem probable that producers will authorize this use of data now considered confidential. The Bureau is attempting to obtain information on foreign production and shipments.

Other sources of information include the American Iron and Steel Institute and the British Iron and Steel Federation.

PRICES

Price schedules are published regularly in Steel, American Metal Market, Daily Metal Reporter, E&MJ Mineral and Metal Market Report, and other trade journals.

FUNCTIONS OF FERROALLOYS

In many instances ferroalloys added to iron or steel are intended to remain in the metal and to provide special properties such as increased strength or improved resistance to corrosion; an example is chromium in stainless steel. Sometimes, however, the ferroalloy is added to remove unwanted elements, such as sulfur and oxygen; for example, calcium removes oxygen, and the products of deoxidation are rapidly eliminated from the steel. In still other instances, the ferroalloy fixes some undesirable element in a form that remains in the metal (such as sulfur in manganese sulfide) but is in a physical condition that eliminates the unwanted characteristics. Some elements, particularly manganese, combine these three functions.

Although the functions of the alloying elements have been studied extensively by physical metallurgists, the extent and ramifications of alloy influence are incompletely known. Nevertheless, some of the more widely used materials are listed, with their principal effect in ferrous metallurgy applications:

Aluminum.—Acts as a deoxidizer.

Boron.—Increases hardenability.

Calcium.—Is a strong deoxidizer.

Chromium.—Imparts a wide range of physical properties to steel, including increased resistance to corrosion, greater depth hardening, and increased strength.

Cobalt.—Imparts high-temperature stability, useful magnetic properties, increased resistance to corrosion, and, with nickel, a minimum coefficient of expansion.

Columbium.—Used principally in alloys for high-temperature applications; inhibits intergranular corrosion in stainless steels.

Manganese.—Neutralizes sulfur and acts as a deoxidizer. Large amounts in steel impart the property of progressive work-hardening. It can be substituted for nickel in certain stainless steels.

Molybdenum.—Increases tensile strength and hardenability; is an effective additive for imparting high-temperature strength.

Nickel.—Imparts corrosion resistance at high temperatures, increases hardenability, tensile strength, and toughness, and imparts useful magnetic properties.

Phosphorus.—Is usually detrimental to steel, causing embrittlement and low resistance to impact, but it is employed in special cases to increase machinability and to impart useful electrical properties; it increases fluidity of molten cast iron.

Selenium.—Imparts improved machinability, particularly in stainless steels.

Silicon.—Deoxidizer; imparts desirable electrical properties to steel for certain applications, and, in proper amounts, improves tensile strength.

Sulfur.—Usually detrimental to steel, it is used in special instances to improve machinability.

Titanium.—Inhibits grain growth; is a deoxidizer, desulfurizer, and degasifier (nitrogen); inhibits intergranular corrosion of stainless steel.

Tungsten.—Imparts great hardness, which is retained at high temperatures.

Vanadium.—Grain refiner; imparts high-temperature strength and hardness.

Zirconium.—Is a deoxidizer and desulfurizer; fixes nitrogen.

Some qualities attributed to alloying elements above are obtainable to an appreciable degree only when they are present in combination with one or more of the other elements. Often complex ferroalloys, such as ferrochromium-silicon, are manufactured and used to add two or more elements simultaneously.

Functions of ferroalloys might also be grouped in another manner, as follows:

Deoxidizers.—Aluminum, calcium, manganese, silicon, titanium, zirconium.

Agents to Improve Hardenability.—Boron, chromium, manganese, molybdenum, nickel, silicon, tungsten, and vanadium.

Agents to Improve Resistance to Corrosion.—Chromium, columbium, nickel, phosphorus, silicon, titanium.

Agents to Provide Special Magnetic and Electrical Properties.—Cobalt, nickel, phosphorus, and silicon.

Desulfurizers and Sulfur Neutralizers.—Manganese, titanium, zirconium.

Agents Much Used to Provide Properties Desired in Alloys for Use at High Temperatures.—Chromium, cobalt, columbium, molybdenum, nickel, titanium, tungsten, and vanadium.

OUTLOOK

Both capacity to produce and actual production of ferroalloys are related to similar factors in the steel industry, and this relationship probably will continue. It is estimated that domestic production of ferroalloys will hover near 2 million tons a year until there is a significant change in steelmaking capacity. About 85 percent of this output will continue to be alloys of manganese and silicon.

Gradual migration of the ferroalloy industry to the Southern and Western States can be expected as a long-term development.

Ores will be lower grade; an increasing use of concentrates, rather than direct-shipping ores, will lead to new furnace practices. Ferroalloys, particularly ferromanganese and ferrochromium, may be lower grade, in part because high-grade ore will be less abundant and in part because of the economic advantage of using less costly raw materials.

A portion of the ores held in the National Strategic Stockpile and those now being obtained by the Government from low-grade domestic sources may be converted to ferroalloys. The combined stimulus of new capacity and extensive research in conjunction with defense programs favors technical developments at both ferroalloy and steel plants. Such developments may lead to important changes in uses and in manufacturing methods.

PROBLEMS

Production of ferroalloys requires large quantities of electric power. During periods of high industrial activity, when there is a maximum demand for ferroalloys, this type of energy is at a premium; however, some companies build power units as integrated parts of a ferroalloy plant.

Stockpiling ferroalloys leads to the joint problems: "What kind? How much?" It is difficult to project the anticipated needs of a period to some unknown time in future.

Technical problems include development of new products, methods for using concentrates rather than lump ore, and production of satisfactory alloys from low-grade raw materials.

Producers of ferroalloys tend to regard their processes as trade secrets, and there is remarkably little published information on methods of production. The development of improved techniques may be hampered by this practice.

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FLUORINE

By

John E. Holtzinger[1]

FLUORINE is a poisonous, violently reactive gas that burns water; attacks glass, metals, and even asbestos; yet combines with other elements to form compounds highly resistant to chemical and physical attack. Important in war and peace, fluorine compounds are used in preparing uranium isotopes for atomic energy; in producing aluminum and steel; in preparing high-octane aviation fuels; in refrigerants, propellants, and plastics; in producing glass and enameled products; and in many other uses essential to the national economy.

Summary

Fluorite (CaF_2), more commonly known as fluorspar, is the principal fluorine mineral. Natural cryolite (Na_3AlF_6) has a higher fluorine content, but its occurrence is almost unique—the only known deposit of commercial tonnage is at Ivigtut, Greenland, and the total supply is relatively small. Phosphate rock often contains 2½ to 3½ percent fluorine and is potentially a large source; however, recovery of this fluorine is limited at present by technologic and economic problems.

Although the United States has been the world's largest fluorspar producer, domestic consumption far exceeds output, and a large part of the supply is imported. Most of the fluorspar produced in the United States is mined in an area along the Ohio River in southern Illinois and western Kentucky. This district also contains the larger part of known domestic reserves. However, capacity in the West has increased substantially in recent years, particularly in Colorado, Nevada, and Montana, and in future a larger portion of domestic output is expected to be produced in this area. Mexico, Canada, Germany, Italy, and Spain are the principal foreign suppliers.

Historically, the largest use of fluorspar has been as a flux in the production of basic open-hearth and basic electric-furnace steel. However, the second largest use, for production of hydrofluoric acid which in turn is used as a catalyst in the production of high-octane gasoline and to produce synthetic cryolite, aluminum fluoride, and many other fluorine compounds, is increasing rapidly and may soon equal or exceed metallurgical uses. The third major use is in the manufacture of glass and enameled products.

There are no adequate substitutes for fluorspar in any of its major uses. Potentially a large alternative source of fluorine exists in waste gases evolved in processing phosphate rock to produce fertilizers and other phosphate products. A relatively small quantity of this fluorine now is being recovered in the form of fluosilicates by several phosphate-rock processors, and a small tonnage of synthetic fluorspar also is obtained from this source by the Tennessee Valley Authority, but to date commercial recovery of fluorine from phosphate rock in forms suitable for widespread industrial use has been on a relatively small scale.

The outlook is for increased consumption of fluorspar, particularly as a source of fluorine for the chemical industry. A long-range expansion in domestic output also may occur, but at present producers are confronted with a major problem in competing against greatly increased imports; future output to a large degree depends on the volume and price competition of future imports.

Some producers in the Illinois-Kentucky district must pump large quantities of water from the mines, adding substantially to production costs. Other problems include the need for more efficient recovery at reduced costs, increased recovery of byproducts, processes for utilizing lower grade and complex ores, the development of wider markets for flotation concentrates, and the need for additional information on reserves.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

SIZE OF FLUORSPAR INDUSTRY

The growth of the fluorspar industry in the United States during the present century has been spectacular, particularly since 1941. From an annual average of about 40,000 short tons in the decade 1900–09, production of fluorspar increased to an average of about 323,000 tons during the decade 1940–49. In the same period, average annual imports of fluorspar rose from about 15,000 tons to 57,000. In more recent years requirements for consumption and stockpiling have greatly stimulated imports, and since 1952 imports have exceeded domestic output. Shipments of fluorspar from domestic mines in 1953 totaled 318,036 short tons valued at $15,737,000 and imports 361,219 tons valued (at the foreign port of shipment) at $11,551,267. Table 1 shows the trends in production, imports, and consumption of fluorspar in the United States since 1900.

TABLE 1.—*Trends in shipments, imports, and consumption of fluorspar in the United States since 1900*

[Short tons]

Date	Shipments	Imports	Consumption
1900–9 (ave.)	40,222	15,000	(1)
1910–19 (ave.)	139,733	18,694	(1)
1920–29 (ave.)	125,130	45,463	[2] 189,067
1930–39 (ave.)	107,832	24,079	128,940
1940–49 (ave.)	323,422	57,326	346,886
1950	301,510	164,634	426,121
1951	347,024	181,275	497,012
1952	331,273	352,503	520,197
1953	318,036	361,219	586,798

[1] Figures not available.

[2] Average for 1927–29.

Most of the domestic output of fluorspar is mined in an area along the Ohio River in southern Illinois and western Kentucky. However, capacity in the West is increasing. In 1953 Colorado was the second largest producing State. Important tonnages also are mined in Montana, New Mexico, Nevada, and Utah and small outputs in Arizona and Idaho. Table 2 shows the principal producers of fluorspar in the United States. Shipments by States, for 1944–53, with maximum shipments in any year and cumulative shipments to the end of 1953, are given in table 3.

Fluorspar is consumed in almost all States and the District of Columbia, but three States—Illinois, Ohio, and Pennsylvania—supply about half of the total consumption. Consumption by States for 1952 and 1953 is given in table 4.

TABLE 2.—*Principal producers of finished fluorspar in the United States*

Producer	Home office of company	Location of milling plant		Chief products [1]
		County	Nearest town	
Colorado				
General Chemical Division, Allied Chemical & Dye Corp.	New York, N. Y.	Boulder	Boulder	A
Ozark-Mahoning Co.	Tulsa, Okla.	do	Jamestown	A
Do	do	Jackson	Walden	A
Reynolds Mining Corp.	Richmond, Va.	Chaffee	Salida	A
Illinois				
Aluminum Co. of America	Pittsburgh, Pa.	Hardin	Rosiclare	A
Frazer & Hettiger	Marion, Ky.	do	do	M
Mackey-Humm Fluorspar Mining Co.	Elizabethtown	do	Elizabethtown	M
Minerva Oil Co.	St. Louis, Mo.	do	Cave in Rock	C, M
Do	do	do	Elizabethtown	C, M
Ozark-Mahoning Co.	Tulsa, Okla.	do	Rosiclare	A, M
Rosiclare Lead & Fluorspar Mining Co.	Rosiclare	do	Rosiclare	C, M
Victory Fluorspar Mining Co.	Elizabethtown	do	Elizabethtown	M
Kentucky				
J. Willis Crider Fluorspar Co.	Mexico	Crittenden	Mexico	M
Kentucky Fluor Spar Co.	Marion	do	Marion	C, M
Pigmy Corporation	Rosiclare, Ill.	do	Mexico	M
Pennsylvania Salt Mfg. Co.	Philadelphia, Pa.	do	Marion	A
Roberts & Frazer	Marion, Ky.	do	do	C

[1] Abbreviations: A, acid grade; C, ceramic grade; M, metallurgical grade.

TABLE 2.—*Principal producers of finished fluorspar in the United States*—Continued

Producer	Home office of company	Location of milling plant		Chief products[1]
		County	Nearest town	
Montana				
Cummings-Roberts	Compton, Calif	Ravalli	Darby[2]	M
Nevada				
Crowell, J. Irving, Jr	Beatty	Nye	Beatty	M
Kaiser Aluminum & Chemical Corp	Washington, D. C	Mineral	Fallon	A
New Mexico				
General Chemical Division, Allied Chemical & Dye Corp.	New York, N. Y	Luna	Deming	A
Utah				
Chesley & Black	Delta	Juab	Delta[2]	M
Willden Bros	do	do	do[2]	M
Claridge, T. A	do	do	do[2]	M
Bell Hill Mining Co	do	do	do[2]	M

[1] Abbreviations: A, acid grade; C, ceramic grade; M, metallurgical grade.
[2] Location of mine; no mill.

TABLE 3.—*Fluorspar shipped*[1] *from mines in the United States, by States, 1944–48 (average) and 1949–53, with shipments of maximum year and cumulative shipments from earliest record to end of 1953*[2]

[Short tons]

State	Maximum shipments		Shipments by years							Total shipments[1] from earliest record to end of 1953	
	Year	Short tons	1944–48 (average)	1949	1950	1951	1952	1953			
								Short tons	Percent of total	Short tons	Percent of total
Arizona	1953	1,951	1,072	846	952	1,623	434	1,951	0.6	21,102	.2
California	1934	181	5	------	------	------	------	------	------	341	(3)
Colorado[4]	1944	65,209	42,007	22,324	18,489	20,661	29,185	53,276	16.8	677,739	7.7
Idaho	1951	(5)	------	------	------	9,408	14,798	18,487	5.8	143,219	1.6
Nevada	1953	(5)	7,644	5,847	7,577						
Illinois[4]	1951	204,328	163,550	120,881	154,623	204,328	188,293	163,303	51.3	4,700,102	53.3
Kentucky[4]	1941	142,862	89,244	63,438	80,137	68,635	48,308	47,244	14.9	2,758,664	31.3
New Hampshire	1917	1,274	------	------	------	------	------	------	------	8,302	.1
Montana	1952	16,160	63	422	41	------	16,160	17,822	5.6	396,641	4.5
New Mexico	1944	42,973	25,500	12,844	20,036	24,402	16,443				
Tennessee	1953	[6] 426	------	------	------	140	[6] 348	[6] 426	.1	2,111	(3)
Texas	1944	4,769	2,245	1,770	719	------	------	------	------	14,779	.1
Utah	1950	18,936	4,012	8,332	18,936	17,827	17,304	15,527	4.9	102,917	1.2
Washington	1945	132	34	------	------	------	------	------	------	382	(3)
Wyoming	1944	19	4	------	------	------	------	------	------	19	(3)
Total	1944	413,781	335,380	236,704	301,510	347,024	331,273	318,036	100.0	8,826,318	100.0

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[1] Figures for 1880–1905 represent production.
[2] Quantity and value figures, by States, for 1880–1925 in Mineral Resources, 1925, pt. 2, pp. 13–14, and for 1910–40 in Minerals Yearbook, Review of 1940, p. 1297.
[3] Less than 0.005 percent.
[4] Figures on production not recorded for Colorado before 1905, for Illinois before 1880, and for Kentucky before 1886 and for 1888–95. Total unrecorded production (estimated) included in "Total shipments" column as follows: Colorado, 4,400 tons; Illinois, 20,000 tons; and Kentucky, 600 tons.
[5] Figures withheld to avoid disclosure of individual company operations, included in total.
[6] Synthetic calcium fluoride recovered by TVA.

TABLE 4.—*Fluorspar (domestic and foreign) consumed in the United States, by States, in 1952 and 1953*

[Short tons]

State	1952	1953
Alabama, Florida, Georgia, Mississippi, North Carolina, and South Carolina	13, 963	18, 556
Arkansas, Kansas, Louisiana, and Oklahoma	47, 434	51, 331
California	14, 626	26, 914
Colorado, Utah, and Wyoming	17, 551	23, 512
Connecticut	1, 496	1, 370
Delaware, District of Columbia, and New Jersey	31, 590	47, 737
Illinois	94, 259	100, 079
Indiana	29, 155	29, 898
Iowa, Minnesota, Nebraska, South Dakota, and Wisconsin	5, 609	5, 009
Kentucky	19, 520	24, 920
Maryland	6, 310	6, 649
Massachusetts and Rhode Island	1, 757	1, 829
Michigan	20, 625	24, 149
Missouri	5, 069	5, 216
New York	16, 644	21, 308
Ohio	71, 240	74, 065
Oregon and Washington	4, 256	2, 775
Pennsylvania	91, 261	92, 003
Tennessee	719	594
Texas	20, 612	22, 394
Virginia	702	307
West Virginia	5, 799	6, 183
Total	520, 197	586, 798

Much of the fluorspar consumed in the United States is supplied by foreign countries, principally Canada, Germany, Italy, Mexico, and Spain. Most of the European fluorspar imported into the United States enters at Philadelphia; the Canadian at Buffalo, New York, Cleveland, and Philadelphia; and the Mexican at Douglas, Ariz., and Laredo, El Paso, Eagle Pass, Brownsville, and Marathon, Tex.

Production of fluorspar in Canada is centered chiefly in the St. Lawrence district of the Province of Newfoundland, although there also is a small production from the Madoc area in the Province of Ontario.

The fluorspar deposits in West Germany are in Baden and Bavaria. The principal Bavarian mines extend about 12 miles from Schwarzenfeld to Mabburg, but the most productive are in the Stulln-Nabburg-Woelsendorf area and supply by far the greater part of the output. The mines in Baden are near Pforzheim and Schwarzwald. In general, the mines in West Germany are well developed and are served by modern milling plants.

Fluorspar occurs in Italy in the Provinces of Lombardy, Trento, and Venetia and on the Island of Sardinia.

The Mexican fluorspar mines are widely scattered throughout the States of Chihuahua, Coahuila, Durango, Guerrero, Mexico, and Sonora. A large part of the Mexican fluorspar has been produced by hand labor. Little machinery is used, and much fluorspar has come from surface workings and been hand-sorted to yield a premium product. However, 3 flotation plants and 1 heavy-medium unit have been installed in Mexico in the past few years.

The principal mines in Spain are in the Provinces of Barcelona, Cordoba, Gerona, and Oviedo. The Collada and Caravia mines in the Province of Oviedo and the Osor mine in the Province of Gerona are the largest fluorspar producers in Spain.

The entire world production of natural cryolite is centered at Ivigtut, Greenland. The deposit is owned by the Danish State and the mining concession by the Kryolitselskabet Oresund Ald, Copenhagen. In the United States synthetic cryolite is produced at East St. Louis, Ill., and at Hurricane Creek, Ark.

RESERVES

The largest fluorspar deposits in the United States are in the districts or States that are now producing the largest quantities of fluorspar; accordingly, by far the largest reserves are in the Illinois-Kentucky field and the next largest in Colorado and New Mexico. Other States having important fluorspar resources are Idaho, Montana, Nevada, Texas, and Utah. Smaller occurrences are known in Arizona, California, New Hampshire, Tennessee, Washington, and Wyoming. In Alaska fluorspar deposits are known at the Lost River tin district on Seward Peninsula; occurrences have been reported elsewhere, but there has been no commercial production.

A recent complete study of domestic fluorspar reserves is not available. However, according to the last comprehensive survey by the Federal Bureau of Mines and the Federal Geological Survey in 1944, measured, indicated, and inferred reserves of crude fluorspar containing more than 35 percent CaF_2 totaled approximately 15 million tons, with an estimated CaF_2 content of about 7 million tons; and reserves of ore containing from 15 to 34 percent CaF_2 totaled 10 million tons, with a CaF_2 content of about 2 million tons (see table 5). Although recovery from many of the lower grade deposits is not economically feasible at the present time, the presence of other valuable minerals and factors such as advantageous geographic location and favorable mining conditions, have made it possible to mine some deposits containing less than 30 percent CaF_2.

Large tonnages of fluorspar have been mined since 1944; but the reserves known at that time did not include recent discoveries, such as the Crystal Mountain deposits near Darby, Mont., and the deposits near Delta, Utah, or new ore bodies that have been discovered in established fluorspar-producing districts.

There are no commercial reserves of cryolite in the United States.

Figures on the reserves of fluorine contained in phosphate rock are not available. The United States has almost unlimited supplies of phosphate rock— at least 13 billion long tons— and some authorities have estimated the fluorine content at about 420 million tons, or an equivalent of about 900 million tons of fluorspar. However, recovery of this fluorine or any part of it depends upon the quantity of phosphate rock mined and sold, upon the treatment processes to which the phosphate rock is subjected, and upon further development of methods of recovering the fluorine in the desired commercial forms.

TABLE 5.—*Estimated fluorspar reserves of the United States as of January 1944, by regions*

[Short tons]

Region	Material (ore) containing 35 percent or more CaF_2					
	Measured	Indicated	Inferred	Total crude material	CaF_2 content	Average grade (estimated percent CaF_2)
Kentucky and Illinois	1, 722, 000	2, 328, 000	7, 000, 000	11, 050, 000	5, 525, 000	50
Rocky Mountain States (Colorado, Utah, Idaho, Wyoming, Montana)	104, 000	782, 000	1, 250, 000	2, 136, 000	961, 200	45
Southwestern States (Arizona, New Mexico, Texas)	102, 000	209, 000	800, 000	1, 111, 000	610, 500	55
Pacific Coast States and Nevada	7, 000	30, 000	80, 000	117, 000	81, 900	70
All other States			4, 000	4, 000	2, 000	50
Alaska		56, 000	250, 000	306, 000	128, 500	42
Total	1, 935, 000	3, 405, 000	9, 384, 000	14, 724, 000	7, 309, 100	49

Region	Material containing 15 to 34 percent of CaF_2 [1] (measured, indicated, and inferred)			Total	
	Crude material	CaF_2 content (estimated)	Average grade (estimated percent CaF_2)	Crude material	CaF_2 content
Kentucky and Illinois	4, 500, 000	675, 000	15	15, 550, 000	6, 200, 000
Rocky Mountain States (Colorado, Utah, Idaho, Wyoming, Montana)	3, 500, 000	700, 000	20	5, 636, 000	1, 661, 200
Southwestern States (Arizona, New Mexico, Texas)	1, 000, 000	200, 000	20	2, 111, 000	810, 500
Pacific Coast States and Nevada	450, 000	90, 000	20	567, 000	171, 900
All other States	140, 000	28, 000	20	144, 000	30, 000
Alaska	2, 000	400	20	308, 000	128, 900
Total	9, 592, 000	1, 693, 400	18	24, 316, 000	9, 002, 500

[1] Based on incomplete data and believed to be conservative.

There are large reserves of fluorspar in many foreign countries, notably Mexico, Canada, Germany, Italy, Spain, and South-West Africa, but no figures as to the quantity are available. Total North American reserves are extensive. Mexico, the leading foreign supplier, is believed to have large reserves of high-quality ore that have been increased by recent discoveries in Coahuila and Chihuahua. Canadian reserves in Newfoundland are quite large and have been estimated by some investigators at several million tons.

No estimate of the cryolite reserves in Greenland is available, but recent reports indicate that exploration has failed to develop appreciable new reserves, and the deposit may be approaching depletion.

Commercial fluorspar deposits occur as veins in fissures or shear zones and as bedding-replacement deposits in both igneous and sedimentary rocks. Even where fluorspar is enclosed by sedimentary rocks, such as limestone, sandstone, or shale, evidence of igneous activity usually may be found. In the Illinois-Kentucky district dikes, sills, and plugs of igneous rock have penetrated the sedimentaries typical of that locality.

GEOLOGY

In the Illinois-Kentucky district the country rock is limestone, shale, and sandstone. Fluorspar occurs as veins along faults ranging in thickness from a mere film to a width of more than 30 feet and in extensive flat-lying replacement-type deposits in limestone. Residual deposits resulting from weathering of fluorite-bearing veins also are fairly common in the district and often indicate the presence of vein deposits at greater depth.

In the Western States fluorspar occurs under a wide variety of conditions—as fillings in fractures and shear zones forming more or less well-defined veins and as replacements in the country rock. Much occurs in igneous formations.

At Ivigtut, Greenland, the cryolite is associated with pegmatite within an intrusive mass of porphyritic granite. Associated minerals are silica, fluorspar, galena, pyrite, and siderite.

MINING

The methods employed in mining fluorspar vary according to the nature of the deposits and the technical and financial ability of the operators. Fluorspar mining is, in many respects, similar to the mining of metalliferous deposits in steeply dipping veins or in flat-lying deposits. Mining is done by shafts, drifts, and room-and-pillar systems and by means of opencuts.

Shaft mines differ greatly in size, capacity, methods of working, and efficiency. At prospect shafts not over 50 feet deep the simplest mining method may be employed with little other hoisting equipment than a hand-operated windlass and a bucket. At small mines more than 50 feet deep single-compartment shafts with wood headframes are used; the ore is loaded at the face and trammed by hand on narrow-gage tracks to the shaft, through which it is hoisted by single-drum hoists powered by gasoline engines.

At larger mines the main shafts usually include two hoisting compartments and a manway compartment containing a ladder, air and water pipes, and electric-power conduits. Steel headframes generally have been adopted, and hoisting is done with double-drum hoists with skips or cages. Mining methods follow the practice of metal mines, adopting the method best suited to conditions. Top slicing, cut-and-fill, and shrinkage and open stoping are among the methods commonly used. Bedded deposits are usually worked by a room-and-pillar system. Some of the larger mines are extensive mechanized, using diesel-powered hauling and loading equipment.

PROCESSING

Although some domestic fluorspar is sold with little or no processing after mining, most crude ore requires beneficiation to yield a finished product. Milling practices range from rather simple methods, such as hand sorting, washing, screening, and gravity separation by jigs and tables, to sink-float and froth-flotation processes. Most of the domestic output is produced by sink-float, flotation, or a combination. These processes permit utilization of lower grade ore; recovery is much more efficient and the finished product of higher quality. Also, the flotation process permits recovery of the lead and zinc values often associated with the fluorspar ores.

To a large extent the selection of a milling process depends on the character of ore to be treated and the type of product desired. Some highly disseminated ores can be treated successfully only by flotation, since fine grinding is necessary to liberate the component particles. Flotation also is commonly used where a pure product of fine particle size is desired, such as Ceramic- and Acid-grade fluorspar. The heavy-medium or sink-float process is usually employed where a coarse product, such as Metallurgical-grade gravel is desired and enough concentration can be achieved without fine grinding. Some producers use both processes in combination, producing gravel products by sink-float and treating undersize from the heavy-medium mill by flotation or using sink-float to produce a preconcentrate, which is then used as feed for the flotation units.

Flotation and other mineral-dressing techniques are employed to treat cryolite.

Chemical processes are used to recover fluorine from phosphate rock. At present the greater part recovered is in the acidulation step in the production of ordinary superphosphate. The fluosilicic acid is sold or used to prepare salts such as sodium silicofluoride and ammonium, magnesium, and zinc fluosilicates. A small tonnage of fluorine in the form of synthetic fluorspar also is recovered by the Tennessee Valley Authority from stack gases evolved in processing phosphate rock. The recovery process consists in absorbing the fluorine, which is present as hydrogen fluoride, in a bed of lump limestone at temperatures above the dewpoint of the stack gases. The calcium fluoride reaction product separates from the limestone lumps in the form of fines; portions of the bed are withdrawn from the tower at intervals and screened to remove the fines, and the oversize (partly reacted limestone) is recycled to the tower with fresh limestone.

GRADES OF FLUORSPAR

Fluorspar is marketed in three major grades—Metallurgical, Ceramic, and Acid. Metallurgical grade, the lowest priced of the three, may be sold in the form of gravel, lump, artificial pellets, or fine flotation concentrates and usually is required to contain a minimum of 60 effective units of CaF_2. Effective units are determined by subtracting 2½ units of CaF_2 for each unit of SiO_2 present. Ceramic-grade fluorspar is not standardized. Specifications depend largely on requirements of individual consumers, but most ceramic spar is fairly pure, often containing about 95 percent CaF_2, 2½ percent SiO_2, and 1½ percent $CaCO_3$. Acid-grade fluorspar should generally contain a minimum of 97 percent CaF_2 and not more than 1.1 percent SiO_2 for the manufacture of aqueous hydrofluoric acid, but for anhydrous hydrofluoric acid a maximum of 1.5 percent SiO_2 may be tolerated.

USES

Quantitywise the most important use of fluorspar is as a flux in the manufacture of basic open-hearth and basic electric-furnace steel. A very small quantity is used in making Bessemer steel, and small tonnages also are used in many other metallurgical operations, including the production of iron castings, ferroalloys, and nickel and its alloys; the melting and casting of aluminum and magnesium; the smelting of secondary metals; and the manufacture of fluxing compounds.

The second largest use—in the production of hydrofluoric acid—is growing rapidly and may soon equal metallurgical uses. Hydrofluoric acid is a raw material vital to the aluminum industry, where it is used to manufacture synthetic cryolite and aluminum fluoride; it is the major source of fluorine for the chemical industry.

Acid-grade spar requirements of the aluminum industry have expanded rapidly in recent years, paralleling the expansion in aluminum production. Cryolite, which serves as electrolyte in the electrolytic reduction of alumina, is required in large quantities to fill reduction cells when new aluminum-production units are started; substantial quantities are absorbed in cell linings in the first few months of operation, and in regular reduction operations the cryolite disassociates and must be replaced by additional cryolite or aluminum fluoride. Based on 1951 data, it was estimated that 47 pounds of cryolite and 58 pounds of aluminum fluoride were consumed per ton of virgin aluminum produced.

Hydrofluoric acid is used by industry as a source of fluorine for the production of refrigerants and propellants, the fluorocarbon plastics and other fluorine compounds; as a catalyst in the production of high-octane aviation fuels; in etching and polishing glass, pickling steel, cleaning sand from metal castings, and enamel stripping; as a laboratory reagent; and for many other industrial purposes.

The third major use of fluorspar is for ceramic purposes in the manufacture of opal and flint container glass and as an ingredient in enamels for coating steel and cast iron. Ceramic-grade material also is consumed for welding-rod coatings, fiber-glass production, and as an additive to brick clays.

Important quantities of Ceramic- or Acid-grade fluorspar also are consumed by the magnesium industry.

Small quantities of clean, colorless (or nearly so), and flawless crystalline fluorspar have been used in lenses or prisms of microscopes, telescopes, and spectroscopes, but synthetic materials have largely displaced fluorite in these uses.

The largest use of cryolite is in the production of metallic aluminum. Smaller quantities are used in abrasives, insecticides, glass, and enamel.

Markets for the fluosilicates (the principal fluorine products recovered from phosphate rock) are somewhat limited. Fluosilicic acid is used as a disinfectant in the brewing industry, as a preservative, in electroplating, as a concrete hardener, and in the manufacture of silicofluoride salts. Sodium silicofluoride is used for fluorination of community water supplies. Sodium silicofluoride and ammonium fluosilicate are used in insecticides, laundry applications, fluxing, ceramics, and casting light metals. The magnesium and zinc fluosilicates are used as concrete hardeners and the magnesium salt also finds application in magnesium foundries.

SUBSTITUTES

During World War II topaz, ilmenite, aluminum dross, and salt were studied as substitutes for Metallurgical-grade fluorspar, but none was as effective. Bauxite has been used as a flux in the open-hearth steel plant at Port Kembla, Australia; and, although it is cheaper, it is only about one-third as effective as fluorspar. In the past several years the production of a special flux, employing fluorspar as a base, has increased somewhat, and many iron foundries that formerly used fluorspar alone are now using this flux.

At present there is no satisfactory substitute for Acid-grade fluorspar in the manufacture of hydrofluoric acid.

PRODUCTION

Although consumption of fluorspar has increased steadily in the past few years, domestic production has declined since 1951 and currently is at the lowest rate since 1949. Most of the decline in production has been in the output of Metallurgical-grade fluorspar. Since 1953 several producers of Metallurgical-grade spar have closed mines or curtailed operations because of high costs or inability to market their output. Production of Ceramic-grade fluorspar also has declined, although to a smaller extent. However, production of Acid-grade fluorspar in the past few years has been at an alltime high. A large part of the Acid-grade production is captive. Shipments of fluorspar from domestic mines, by uses, for 1944–53 are shown in table 6.

TABLE 6.—*Fluorspar shipped from mines in the United States, 1944–53, by uses*

[Short tons]

Year	Steel	Hydrofluoric acid	Glass	Enamel	Government stockpile	Miscellaneous	Exported	Total
1944	219, 361	121, 084	27, 174	2, 685	24, 396	17, 101	1, 980	413, 781
1945	186, 073	80, 155	32, 300	3, 660	9, 449	10, 904	1, 420	323, 961
1946	134, 295	79, 047	39, 837	7, 540	3, 907	11, 585	1, 729	277, 940
1947	165, 427	89, 667	40, 843	8, 716	9, 109	14, 542	1, 180	329, 484
1948	170, 633	96, 848	35, 960	9, 415	----------	18, 249	644	331, 749
1949	119, 264	70, 759	27, 727	4, 625	----------	13, 546	783	236, 704
1950	149, 410	97, 659	30, 450	7, 832	----------	15, 431	728	301, 510
1951	167, 042	[1] 123, 125	33, 036	6, 356	([1])	16, 317	1, 148	347, 024
1952	142, 058	[1] 136, 514	29, 781	3, 706	([1])	18, 549	665	331, 273
1953	109, 250	[1] 159, 196	27, 535	5, 033	([1])	16, 327	695	318, 036

[1] Shipments to National Stockpile included with hydrofluoric acid.

BYPRODUCTS AND COPRODUCTS

Lead and zinc and minor quantities of other metals, such as silver and cadmium, are often produced as byproducts or coproducts of fluorspar. In the Illinois-Kentucky district the value of lead and zinc produced in the fluorspar district is substantial, averaging about one-fifth of the value of fluorspar shipments from that area in the past 5 years. Germanium is also present in the zinc concentrates.

TABLE 7.—*Fluorspar imported into the United States, 1944–53, by countries*

[Short tons]

Year	Africa	Canada	France	Germany	Mexico	Spain	Italy	Other countries	Total
1944	3, 557	16, 141	--------	--------	58, 324	9, 177	--------	1	87, 200
1945	--------	15, 028	--------	--------	62, 575	27, 322	--------	--------	104, 925
1946	--------	2, 998	--------	--------	24, 063	2, 791	--------	--------	29, 852
1947	--------	10, 591	--------	--------	51, 965	12, 042	4, 126	1	78, 725
1948	--------	16, 380	--------	1, 375	79, 408	9, 718	4, 745	--------	111, 626
1949	--------	15, 344	1, 532	--------	58, 238	12, 648	7, 857	--------	95, 619
1950	--------	14, 163	2, 772	29, 624	73, 051	35, 302	9, 722	--------	164, 634
1951	632	21, 460	1, 415	49, 074	64, 025	32, 865	11, 804	--------	181, 275
1952	7, 065	18, 675	1, 120	60, 451	175, 102	59, 687	30, 403	--------	352, 503
1953	4, 980	22, 020	1, 164	36, 366	190, 413	52, 286	53, 868	122	361, 219

FOREIGN TRADE [2]

Since 1952 imports have exceeded domestic production of fluorspar. Mexico supplied about half the total imports. Important tonnages also are imported from Canada, Spain, Italy, and Germany. Imports, by country of origin, for 1944–53 are shown in table 7.

Several domestic producers have voiced concern over the increased volume of imports in recent years and attribute the current low rate of domestic output, in large part, to heavy competition from imported material. Fluorspar is subject to a duty of $1.87½ per short ton for material containing over 97 percent CaF_2 and $7.50 per ton for lower grade material. Some producers have protested that these rates afford inadequate protection to the domestic industry; and the United States Tariff Commission, in accordance with a resolution of the United States Senate Finance Committee, has initiated a study of the facts relative to production, trade, imports, and consumption of fluorspar in the United States and the effect of the present tariff status upon the competitive position of the domestic industry.

Natural cryolite is imported from the sole producer, Greenland, and synthetic cryolite from several countries. Imports of cryolite for the past 10 years are shown in table 8. There is no tariff on cryolite.

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[2] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

TABLE 8.—*Cryolite imported into the United States, 1944–53*

Year:	Short tons
1944	17, 562
1945	20, 106
1946	11, 424
1947	22, 008
1948	2, 353
1949	20, 506
1950	17, 134
1951	38, 851
1952	38, 373
1953	29, 457

Export trade of the United States in fluorspar (see table 6) and cryolite is small.

CONSUMPTION

Consumption of fluorspar has increased rapidly during the past few years, principally as the result of greatly expanded aluminum and steel production and increased use of fluorine chemicals. Although the steel industry is the largest consumer, an increasing proportion of total consumption is being used for production of hydrofluoric acid. Usage at glass and enamel plants has declined slightly in recent years. Consumption, by uses, for 1927–53 is shown in table 9.

TABLE 9.—*Fluorspar (domestic and foreign) consumed in the United States, 1927–53*

[Short tons]

Year	Steel	Hydrofluoric acid	Glass	Enamel	Iron foundry and ferro-alloys	All other	Total
1927	142, 700	15, 500	6, 800	5, 800	3, 900	1, 500	176, 200
1928	158, 100	20, 500	6, 200	5, 700	4, 100	1, 600	196, 200
1929	162, 100	15, 600	6, 600	5, 200	3, 800	1, 500	194, 800
1930	112, 600	12, 600	4, 300	4, 000	2, 700	2, 000	138, 200
1931	69, 300	12, 000	7, 100	3, 000	1, 300	1, 300	94, 000
1932	38, 400	7, 000	6, 700	2, 400	800	700	56, 000
1933	64, 700	7, 800	7, 000	3, 200	1, 200	700	84, 600
1934	85, 300	11, 000	7, 700	3, 500	2, 100	1, 000	110, 600
1935	105, 000	12, 900	11, 000	4, 900	2, 600	1, 000	137, 400
1936	140, 800	20, 100	11, 600	5, 400	2, 700	1, 800	182, 400
1937	146, 400	24, 100	11, 600	5, 900	3, 700	2, 600	194, 300
1938	77, 600	18, 900	10, 500	4, 000	2, 800	1, 300	115, 100
1939	123, 800	26, 300	15, 300	6, 100	3, 500	1, 800	176, 800
1940	155, 500	37, 000	13, 400	5, 500	4, 600	2, 500	218, 500
1941	210, 400	56, 000	20, 300	7, 300	5, 100	4, 500	303, 600
1942	242, 600	81, 600	18, 500	3, 100	7, 800	7, 200	360, 800
1943	234, 148	113, 614	20, 592	1, 726	7, 260	11, 545	388, 885
1944	230, 201	129, 553	27, 315	2, 547	7, 815	12, 739	410, 170
1945	197, 916	109, 315	31, 874	3, 695	6, 786	6, 504	356, 090
1946	160, 735	83, 901	39, 852	6, 739	6, 969	4, 994	303, 190
1947	209, 395	100, 363	42, 130	8, 938	6, 567	8, 745	376, 138
1948	232, 687	107, 280	37, 247	8, 871	8, 817	11, 367	406, 269
1949	201, 501	89, 152	30, 797	5, 510	7, 816	10, 445	345, 221
1950	240, 802	124, 440	33, 440	7, 723	9, 709	10, 007	426, 121
1951	276, 654	151, 698	35, 505	6, 736	9, 348	17, 071	497, 012
1952	272, 476	178, 267	33, 837	5, 205	9, 957	20, 455	520, 197
1953	287, 607	223, 359	32, 955	5, 863	17, 656	19, 358	586, 798

PRICES

The average selling price of all grades of domestic fluorspar shipped reached an alltime high in 1953, but in 1954 prices dropped sharply. Table 10 shows selling prices to specified industries since 1944.

TABLE 10.—*Selling prices of domestic fluorspar shipped to specified industries, 1944–53, per short ton f. o. b. mine shipping point*

Year	Steel	Hydrofluoric acid	Glass	Enamel	All industries (including exports)
1944	$27.75	$35.11	$32.85	$33.68	$30.22
1945	27.85	36.13	32.00	35.14	30.55
1946	28.62	39.36	32.78	34.82	32.52
1947	29.01	40.84	35.13	36.20	33.25
1948	29.65	39.78	35.99	38.46	33.84
1949	29.81	42.27	37.64	40.28	34.92
1950	29.34	42.65	38.65	41.76	35.22
1951	34.38	50.05	42.61	46.15	41.41
1952	35.29	58.07	44.40	48.84	46.35
1953	35.45	59.54	46.38	54.36	49.48

GOVERNMENT PROGRAMS

Several Government programs have been initiated to assure supplies of fluorspar adequate for domestic requirements. Substantial quantities of Acid and Metallurgical grades of fluorspar have been purchased for the National Stockpile, and recently a program for purchases of additional quantities of Metallurgical-grade fluorspar has been announced. Domestic capacity to produce Acid-grade fluorspar was increased considerably as the result of contracts entered into by the Defense Materials Procurement Agency during the Korean emergency. Currently Government loans of 50 percent of the allowable cost of exploration projects for fluorspar are available under the Defense Minerals Exploration Program. Such loans are repayable only if a certificate of discovery is awarded as the result of the exploration. The Internal Revenue Code of 1954 provides that fluorspar shall be subject to a 23-percent depletion allowance, compared with the 15 percent specified in the previous law. Several certificates authorizing rapid amortization of newly constructed fluorspar-production units have been awarded.

Considerable research has been conducted by Government agencies, both State and Federal, relating to fluorine supplies. Methods of concentrating fluorspar ores by froth flotation have been developed and other metallurgical studies on fluorspar conducted. Ores from a number of properties have been investigated to determine their amenability to flotation and other beneficiation processes. Investigations have been made of various binders for pelletizing fine fluorspar for metallurgical use. Mining methods have been studied and improvements suggested. Diamond-drilling programs have been carried out in fluorspar-producing districts. Research is now being conducted to develop improved methods of recovering fluorine from waste industrial gases evolved in processing phosphate rock.

OUTLOOK

A substantial upward trend during the next decade in fluorspar consumption in the United States is indicated. No major new uses for fluorspar itself are in sight, but new and expanded uses of fluorine chemicals and increases in steel and aluminum production are expected to result in increased fluorspar requirements.

Consumption of Metallurgical-grade spar is largely controlled by basic steel production. Although the steel industry is currently operating at a level considerably below the rate in recent past years, increasing population and possible increases in per capita consumption of steel point to a long-range expansion in steel output. Additional supplies of fluorspar may be required for production of synthetic cryolite and aluminum fluoride as aluminum production increases and supplies of natural cryolite are depleted; however, it also is likely that more cryolite will be reclaimed at aluminum plants and reused. Possibly the largest area for growth is the fluorine-chemicals industry. Fluorine compounds, such as the Freon and Genetron refrigerants and propellants and the fluorocarbon plastics, are being applied in increasing quantities to a wide variety of uses. New fluorocarbon compounds—synthetic compounds in which fluorine replaces hydrogen in hydrocarbons—with outstanding chemical and physical stability and other useful properties are being developed. Production of high-octane aviation fuels, which requires hydrofluoric acid for catalytic purposes, is expected to increase. The field of fluorine chemistry is relatively new, and some authorities have predicted that domestic requirements for fluorine will increase to 2 or 3 times the present usage.

A long-range increase in domestic production of fluorspar also may occur, but to a large degree future output from domestic mines is conditional on the quantity imported and the price structure.

Another important factor in the outlook for fluorspar is the possibility of increasing the commercial recovery of fluorine from waste gases evolved in processing phosphate rock as a high-quality synthetic fluorspar or cryolite, hydrofluoric acid, or other compound suitable as a source of fluorine for industrial uses.

PROBLEMS

One of the most urgent problems facing domestic producers is increasingly heavy competition from imported fluorspar. Several producers have closed mines or curtailed operations because of inability to market their product as supplies from foreign sources increased. Prices have been reduced sharply in the past year. The problem is a complex one, involving low production costs in foreign countries, import duties, freight costs, specifications, and other economic and technical factors.

Scant information is available on both domestic and foreign reserves. Fluorspar has been produced at a relatively high rate since the last comprehensive study of domestic reserves in 1944. Some large mines have been exhausted. On the other hand, important new discoveries have been made in the past decade; and in many instances, exploration and development work in established producing areas has been successful. Imports, especially from Mexico, have increased substantially in recent years and constitute an important part of the supply to domestic consumers. However, very little information is available as to the extent and nature of reserves in many leading foreign countries.

Although some of the larger producers have installed modern, efficient mining and milling equipment, still greater mechanization and improved milling methods may be needed to increase productivity and to make possible recovery from lower grade ores. Technical problems include the need for new and improved processes to increase byproduct recovery and to produce fluorspar from complex ores, such as those containing fluorspar and barite.

Development of methods for wider utilization of flourspar in the form of flotation concentrates or as artificial pellets for metallurgical purposes, which would improve the marketing situation of the domestic industry, constitutes another problem. Many of the lower grade ores in the United States that contain large quantities of undesirable minerals finely intermixed with the fluorite can be concentrated by flotation to a product pure enough for metallurgical uses. However, flotation fines are objectionable for basic open-hearth steel production because of dust losses, which waste the fluorspar and cause damage to the checker brick.

Specifications vary considerably even within a single field of use, and this variation often is a serious commodity problem, particularly to producers.

A baffling problem in some mines in the Illinois-Kentucky district is the handling of large quantities of water that enter the mines through channels in the limestone. For example, water enters the Rosiclare mine—in the past one of the largest producers in the United States—at the rate of 7,000 gallons per minute.

Phosphate-rock processing and a number of other industrial processes generate fluorine waste products. These wastes present a number of problems, such as air contamination. They could, however, serve as a very large alternative source of fluorine if commercial processes were to be developed to recover the fluorine content in forms permitting widespread utilization by industry.

Fluorspar is essential to industry in a national emergency. This presents problems of assuring adequate supplies. This involves stockpiling, maintenance of domestic production capacity, and import policies.

Because fluorspar and fluorine are indispensable in either a war or peace economy and future requirements may increase greatly, more comprehensive technical and economic data concerning the fluorspar and related industries are needed to provide basic facts required for most effective management by industry and for guidance in mineral-policy decisions.

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GALLIUM

By

John D. Sargent[1]

GALLIUM is a gray, metallic element that melts at less than body temperature, is half as heavy as mercury, and is twice as expensive as gold.

Summary

The only gallium produced in the United States today is recovered from zinc flue dust, electrolytic-zinc-plant residues, and circulating liquors of the Bayer aluminum process. This metal is produced in this country at the rate of a few hundred pounds annually. Present production is adequate to meet current demands. If methods are developed for producing the metal at lower cost, additional markets may be developed.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

HISTORY

Gallium was discovered by the French chemist Lecoq de Boisbaudran in 1875 by spectroscopic examination of Pyrenean sphalerite (*5*).[2] He named the element in honor of France, Gallia being its Latin name (*4*).

Interest in gallium in the United States began in 1915, when F. G. McCutcheon accidentally discovered beads of gallium-indium alloy exuding from slabs of lead that had been exposed to sunlight (*3*). Over a period of years he developed recovery processes and produced the first pound lot of the metal. He processed other materials containing gallium and by 1944 had recovered about 15 pounds of the metal. The Eagle-Picher Co., at Joplin, Mo., became interested in McCutcheon's work and produced gallium for several years.

PROPERTIES

Gallium is unique among metals in that it remains liquid through a temperature range of 3,515.5° F. from a melting point of 85.5° F. to a boiling point of 3,601° F. Unfortunately, it is extremely corrosive and consequently very difficult to handle, particularly at higher temperatures (*1*). The specific gravity of gallium is 5.9 compared with 13.6 for mercury.

EXTRACTIVE METALLURGY

Gallium can be recovered as a byproduct in the metallurgical treatment of ores of other metals by several methods. One of these begins with the Bayer process, a wet chemical procedure used to extract alumina from bauxite ores. In this process sodium aluminate is formed by treating bauxite with aqueous sodium hydroxide and precipitating most of the alumina from the solution. The gallia and remaining alumina are coprecipitated with carbon dioxide, and gallium is separated by electrolysis.

Whether gallium is recovered from zinc ores or not is determined by the process of zinc extraction utilized. All gallium content of zinc ore is lost when zinc is recovered by the Waelz process (*5*). The sulfuric acid leach process produces an "iron mud" from which a gallium concentrate is extracted by a series of caustic soda and acid leachings. Finally, gallium chloride is extracted with ether, and metallic gallium is prepared electrolytically from this chloride.

WORLD DISTRIBUTION

The metal gallium exists in the lithosphere in the same quantity as lead, or about 15 grams per ton of earth.

The coals of Northumberland, Durham, and Yorkshire Counties, England, are rich in gallium. Flue dust from some of these coals contains as much as 0.17 percent of the element. Gallium and germanium have been extracted commercially from flue dust in England.

Bauxite from India contains as much as 0.008 percent gallium. Bauxite from Arkansas, Georgia, Tennessee, Italy, and Netherlands East Indies assays 0.002 to 0.005 percent gallium. Some tin ores from Bolivia contain 0.01 to 0.05 percent gallium.

PRODUCTION AND CONSUMPTION

Before World War II Germany was the principal world producer of gallium. The metal was recovered from aluminum phosphate residues at Mansfield at the rate of 300 pounds a year.

In 1943 the Anaconda Mining Co. began producing both gallium and indium from the residues resulting from the leaching of zinc concentrates in its electrolytic zinc plant. These concentrates are from the western part of the United States and from Canada, Mexico, and South America. As no analyses of the concentrates are made for gallium, the origin of the metal is not known.

Another producer of gallium in the United States has been the Aluminum Co. of America, which obtained the metal from the liquors in the Bayer aluminum process. The bauxite ores are reported to contain 1 ounce of gallium per ton of ore. In 1948 the Saratoga Laboratories, Inc., also produced gallium metal (*1*).

USES

The principal use for the gallium in Germany before World War II appears to have been in dental alloys to replace mercury. Another interesting use, both in that country and other

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

parts of the world, was in high-temperature thermometers, in which the metal was placed in fused quartz tubes under vacuum. Temperatures from 500° to 1,200° C. were measured with these thermometers.

Gallium is frequently used in plating optical mirrors because it reduces light-absorption losses.

Atomic and astrophysical research in spectroanalysis, which was hindered by insufficiencies of the mercury-quartz lamps, can now be aided by the use of gallium lamps.

In a French discovery the metal is used as a cathode material in electron tubes. A filament of gallium or its alloy emits electrons at lower temperatures than ordinary filaments. Electron tubes of that type require less current for heating and are especially adapted for portable receivers. Gallium is being used as a trace alloying agent for producing "P"-type germanium for transistors.

Attempts are being made to substitute gallium for mercury in rectifiers because gallium's higher boiling point will permit operation at much higher capacity. Gallium alloys with low melting points are used in fire alarms. The intermetallic compounds—gallium antimonide and gallium arsenide—are being studied as potential semiconducting materials.

Gallium has been used as an excitant in phosphors for fluorescent lighting and luminous paints. The consumption of the metal for this purpose is small (*1*).

The radioisotope of gallium, Ga–72, has been used in the detection and treatment of bone cancer (*5*). Organic salts of gallium have been used to treat syphilis and tropical blood fevers, such as sleeping sickness.

Gallium alloys readily with most metals. It is completely miscible with liquid tin, and an alloy with 12 percent tin melts at 15° C. Gallium is virtually immiscible with mercury, sodium, and potassium and dissolves only to a slight extent in liquid lead. In general, gallium gives little improvement to structural alloys that cannot be achieved more economically by use of other constituents.

A little gallium increases the strength and hardness of copper, but the electrical conductivity is decreased considerably. A higher gallium content renders the alloys brittle, so that they cannot be turned; this applies also to silver and zinc alloys. Gallium-magnesium alloys, having a hardness up to 50 Brinell, are easily turned or rolled. This combination of properties suggests a use in the aviation industry. Aluminum, indium, and tin form low-melting alloys. An alloy of 60 percent Sn, 30 percent Ga, and 10 percent In has a melting point below 15° C.

An alloy of nickel, silicon, and gallium has proved to be exceptionally strong. Tests demonstrated that a standard alloy sample with a cross-sectional area of 0.003 inch will support a 2,000-pound load.

Gallium forms many compounds, including halides, hydrides, oxides, sulfides, sulfates, nitrates, and alums. Few uses for these compounds have been described. Organic gallium salts exert some curative action in syphilis and trypanosomiasis. Gallium halides can be used as catalysts.

PRICES

From 1950 through 1954 quotations on gallium metal, 99.9 percent pure, were: $4.50 per gram in lots of less than 100 grams, $3.50 a gram for lots of 100 to 999 grams, and $3.00 a gram for lots of 1,000 to 2,499 grams. In the early 1930's gallium had sold for $50.00 per gram (*1*).

RESEARCH

Research by industry has been confined largely to developing the present procedures of metallurgical extraction of gallium as a byproduct from the aluminum and zinc industries. The Eagle-Picher Co. at Joplin, Mo., the Aluminum Co. of America research laboratories at East St. Louis, Ill., and the Anaconda Copper Mining Co. at Great Falls, Mont., are the principal organizations that have guided the basic research on the separation and recovery of gallium. The Thomson Research Laboratory of the General Electric Co., West Lynn, Mass., carried out experiments on gallium in 1925 which indicated that the element was adapted for use in thermometers for recording temperatures above 700° C.

In England, the Chemical Research Laboratory, Teddington, Middlesex, has pioneered the research and development of extracting gallium commercially from flue dust. As much as 1,000 tons of gallium a year is probably lost to the atmosphere in the British Isles in gallium-rich flue dusts and ashes derived from burning British coals.

The Department of Chemistry, University of Illinois, conducted a project during 1948 and 1949 on gallium chemistry. The project report (N6–ori–71, Chemistry Task XVII) chiefly concerns analytical methods for gallium. The Department of Chemistry, University of Chicago, carried on a research project between 1947 and 1949 on chemistry of gallium, N6–ori–20, Task Order XVI. This included studies entitled "The Stability of Gallium Halide Complex Ions in Aqueous Solutions" and "A Study of the Chlorogallates." The Department of Metallurgy, University of Utah, conducted a project for the Office of Naval Research entitled

"Physical Chemistry of Gallium-Indium Alloys," October 1947—September 1948, Project NR 05209, Utah Engineering Experiment Station. The main emphasis has been on low-melting compositions, with the chief interest in those that are liquid at room temperatures. This study concluded that further investigations were necessary for gallium alloys.

The Metallurgical Branch of the Bureau of Mines conducted investigations on gallium at Rolla, Mo., for almost 3 years, 1947–50, through a cooperative agreement with the Office of Naval Research. This work consisted of examination of byproducts and industrial residues obtained in processing zinc, in the manufacture of alumina by the Bayer process, in the production of tin at the Texas City tin smelter, and in the manufacture of coke and producer gas.

The National Bureau of Standards conducted experiments in 1920 which indicated that gallium-cadmium alloy containing 2 to 3 percent gallium was satisfactory for preparing a monochromatic lamp. The Bureau also constructed in 1923 an enclosed quartz vacuum lamp using an alloy of gallium and zinc, similar in many respects to the cadmium-gallium lamp.

The Naval Medical Research Institute, National Naval Medical Center, Bethesda, Md., studied the preparation and properties of gallium lactate in 1948 and published a report on it (Project NM 011013, Report 2, May 24, 1948). This study was prompted because of the nontoxic properties of gallium and potential pharmacological uses.

OUTLOOK

Currently (1954), the supply of gallium is more than adequate to meet the demands of industry. Some American companies suspended production of gallium when sales lagged and stocks of the refined metal mounted.

The reserves of low-grade ores and materials that contain gallium as a byproduct metal of aluminum, zinc, and tin are large. Production of the metal could be increased to probably 100 tons yearly by the zinc and aluminum companies in the United States. The quantity of gallium lost in the refining of tin ores in this country has been reported to exceed the potential gallium production of both the aluminum and zinc plants. Some zinc plants are stockpiling gallium-rich plant residues. In the aluminum and tin operations, the gallium-rich liquors and residues are being wasted in the tailings products. To save and stockpile the gallium from these two industries, the metal would have to be put into a semiprocessed state.

The outlook is for a continuing annual increase in gallium consumption, but not enough to absorb the annual production potential. Greatly increased consumption probably would be accompanied by a corresponding lowering of gallium prices.

PROBLEMS

The cost of producing gallium by current methods is high. Of major importance is the need to reduce the cost of recovering gallium either by simplifying present extraction methods or by developing new and cheaper metallurgical procedures. Industry is making considerable progress in these directions.

The lack of research on useful applications of gallium is partly attributable to its high cost. Inasmuch as the high cost is chiefly attributable to the low volume of production and consumption, a stalemate in gallium progress has developed.

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GARNET

By

Henry P. Chandler [1]

GARNET is used as a semiprecious gem, and its hardness and sharp fracture make it a satisfactory abrasive for wood, leather and plastic sanding, and for metal and glass polishing.

Summary

About 1880, garnet first became important as an abrasive owing to the introduction of garnet-coated sandpaper. Before that time small quantities of garnet had been used for jewelry and watch bearings. In 1951, 14,285 tons of garnet was produced in the United States, and of this almost all was used by the abrasives industry.

The largest commercial deposits in the United States are in Warren County, N. Y., and Benewah County, Idaho. Production is also reported from Florida.

Eleven abrasives manufacturing companies are producing garnet paper and cloth in the United States, and garnet reground to fine sizes is used for plate glass and lens grinding, metal polishing, and in grinding and lapping compounds.

As this material is well established in its markets, there is an encouraging outlook in the industry, but need continues for research and for all types of information to guide management and government.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

GENERAL INFORMATION

Garnet is found in greatest abundance in crystalline schists and gneisses, but it also occurs in various other rocks, such as pegmatites, crystalline limestones, serpentines, and peridotites. It also frequently occurs as rounded grains in stream and sea sands.

Although nearly all types of garnet are used for gem stones when clear and transparent enough, the type known as almandite is the one now most commonly used as an abrasive.

The use of garnet as an abrasive was an important one percentagewise in the coated-abrasives industry in 1920, when 39 percent of the abrasive used for this purpose in the United States was garnet; this had dropped to 13 percent in 1953. Although the coated-abrasives industry turned to other abrasives for certain applications, a larger tonnage of garnet is used today than in 1920. The expansion of the woodworking, plastics, and leather fields continued to provide a good market for abrasive garnet.

Use is also stimulated by the fact that a large supply of material of excellent abrasive quality is available in New York, conveniently located for most of the more important users. No failure of the domestic garnet supply, even if the demand increases to much larger proportions, can now be foreseen.

The value of the gem and watch-bearings material produced domestically has decreased from a high point of $22,000 in 1901 to 60 pounds valued at only $80 in 1951.

Garnet varies in color, hardness, toughness, and fracture. For abrasive purposes, the mineral must be the hardest obtainable, at least 7.5 on the Mohs scale. It should be tough, but individual grains must break and form cutting edges. Although color does not appear to be a significant indicator of abrasive qualities, the deep red-colored mineral seems to be preferred.

The production and value of the garnet produced in the United States during recent years are as follows:

Year:	*Short tons*	*Value*
1945	6, 306	$375, 198
1946	7, 743	570, 188
1947	8, 722	614, 071
1948	8, 039	587, 797
1949	6, 578	505, 231
1950	9, 304	793, 558
1951	14, 050	1, 246, 947
1952	11, 390	981, 841
1953	10, 520	988, 797

OCCURRENCE

New York, with mines in Warren and Essex Counties, is the largest producer, followed by Idaho and Florida. Although production has been reported in New Hampshire, Connecticut, Pennsylvania, and North Carolina, none of the garnet deposits in them are now being operated.

The largest commercial garnet mine in the United States is that of the Barton Mines Corp. on Gore Mountain, near North Creek, Warren County, N. Y. This mine has been in continuous operation for over 75 years and still has large ore reserves. The garnet-bearing rock is a dark-gray, massive, granular rock consisting largely of hornblende and feldspar with garnet nodules—single individual crystals of imperfect form, from less than 1 inch to 3 feet in diameter, surrounded by shells of hornblende. At the surface the enclosing rock is decomposed. The mine is now operated as an opencut and is fully mechanized. The mill was one of the first to use the heavy-medium separation process, the original installation starting in 1941. Since then improvements in the operating method have been added. Tube mills regrind the product for the glass- and metal-polishing trade.

Erosion of soft mica schists has created extensive alluvial deposits of almandite garnet on Emerald Creek, Benewah County, Idaho. These deposits (owned by Idaho Garnet Abrasive, Fernwood, Idaho), have a garnet content up to 20 percent; the garnets having a maximum grain size of 3/16 inch. Overburden of 3 to 8 feet must be stripped to expose the garnetiferous gravels. The gravel is mined with a 1½-yard dragline, dumping into a trommel-screen dredge to produce a 40- to 50-percent concentrate, which is then fed to jigs and concentrating tables. The resulting 99 percent pure garnet is dried, crushed, and screened into the various sizes required by industry. The crushed material has greater efficiency than the uncrushed, because sharper cutting edges are exposed. A small output was reported as a byproduct from gold dredging near Warren, Idaho County.

Idaho garnet is used mainly for sand blasting, in the aircraft industry, and for cleaning sparkplugs.

USES

Adirondack garnet finds its largest market in the coated-abrasives industry. It is the abrasive on garnet paper and cloth used for woodworking, leather and plastics finishing, and a variety of other grinding operations. In grain form it is used principally in plate-glass and optical-lens grinding, in metal polishing, and in sand blasting. Relatively small quantities are used in bonded grinding wheels.

Abrasive-garnet concentrates have increased in price from an average of about $60 a short ton in 1945 to $87 a short ton in 1951. Adirondack garnet commands a higher price than the other varieties and recently averaged about $105 a ton. Idaho garnet had an f. o. b. sales value of about $60 per ton in 1951.

The Barton Mines Corp. has carried on research into the use of finely ground garnet for optical- and instrument-lens grinding, and its use in these fields has expanded rapidly in recent years.

Manufacturers of coated abrasives have expanded the use of garnet paper and cloth by changing certain grinding and surfacing operations to the use of garnet-coated belts and disks rather than grinding wheels.

Patents have been issued covering the use of garnet for the manufacture of an artificial high-alumina abrasive, but this process is not used commercially.

The abrasive qualities of garnet are improved considerably by roasting for several hours at 800°–900° C. This has the effect of increasing its toughness and sharp fracture. Temperatures of over 1,000° C. have a detrimental effect on the final product.

OUTLOOK

The fact that the demand for garnet has increased in recent years is attributed to a combination of high industrial activity and effective market development. In the past, garnet concentrates have been imported, mostly from Spain, but in recent years the domestic material has become so well established in the market that imports no longer are a factor. As no shortage of garnet is anticipated, prices are expected to continue to be reasonable. With the general expansion of the abrasive market, there is a good prospect that garnet consumption will continue to increase. In view of these favorable aspects, there is an optimistic outlook in this industry.

PROBLEMS

The principal problems of the established producers of garnet are maintenance and increase of sales in the face of active competition of other abrasives. Numerous other mineral operations are potential sources of byproduct garnet, but their development is limited by the fact that the market is being adequately supplied with good-quality material from established producers. Such potential sources need reliable information to assist them in evaluating the commercial outlook for their products.

Both producers and consumers would benefit from more research on garnet utilization. Not enough is known about the substitution potential of garnet. As the United States has a large domestic supply, effective research leading to the substitution of garnet for abrasives that are not as easily available would improve national self-sufficiency. Research on the fundamental abrasive properties of garnet and other abrasive materials, to indicate the best field of use and the degree to which they can be interchanged, is much needed.

Both producers and Government agencies concerned with garnet need reliable technical, statistical, and economic information to guide their activities.

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GEM STONES

By

Robert D. Thomson[1]

OF ALL the wonders provided by nature for the enjoyment of man, few hold a more prominent place than gem stones. All races and social classes have been fascinated by the beauty of gems for jewelry or in gem collections. Pecuniary reward, as well as pleasure, has been gained from the pursuit and possession of gem stones.

Summary

Minerals and organic materials, to be suitable for use as gems, must have three significant qualities—beauty, durability, and rarity. Other factors—fashion, portability, and world economic conditions—also determine the acceptability of a substance for gem purposes, but these are minor to the three principal requirements.

Gem stones occur in a variety of rocks and formations. Principally, gems are obtained from igneous intrusive rocks and placers; pegmatite dikes and stream placers furnish the greater quantity and variety.

The United States has never been important in world gem-stone production. In 1953, the estimated value of gem-stone output, essentially semiprecious, ranged from $400,000 to $500,000. The greatest factor in this production is the thousands of amateur collectors engaged in gem-stone collecting as a hobby. Enthusiasm for gem collecting has grown into a tremendous hobby and commercial activity. The United States virtually depends on foreign sources for precious stones—diamond, ruby, sapphire, and emerald. Imports of diamonds alone totaled $107,600,000 in 1953.

Gem stones are used principally for personal adornment and gem collections. Thousands of amateur collectors, spending weekends and vacations searching for gem stones, produce and utilize large quantities for personal collections or for handmade jewelry. The supply of gem stones from domestic and foreign sources is sufficient to meet demands.

Chief problems of the domestic gem-stone industry are lack of known deposits of high-quality gem stones and high labor costs, especially for cutting and polishing. It is in the national interest to encourage the activities of collectors and lapidaries; their efforts may disclose new sources of strategic minerals, and their technical skills and facilities may serve as a nucleus for emergency expansion of jewel-bearings production and related manufacturing industries. More gem-stone publications of interest to laymen are needed.

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[1] Commodity-industry analyst, Bureau of Mines.

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BACKGROUND

From the earliest times to the present, mankind has been fascinated by gems and has treasured gems for personal adornment, amulets, and charms. Between 100,000 B. C. and 7,000 B. C., paleolithic man used many different varieties of gem stones. Chalcedony, serpentine, obsidian, pyrite, amber, jade, calcite, amethyst, agate, and jet are known to have been in use during this period and bloodstone, emerald, citrine, lapis lazuli, moss agate, zircon, ruby, opal, and diamond before the birth of Christ.

American Indians used many precious and decorative stones. Many were obtained from fragments of float and river gravels; for example, jade in Alaska, amber on the Yukon Delta, and garnet in New Mexico (*2*).[2] However, Indians did excavate to obtain turquois, obsidian, jasper, onyx, and catlinite (allophane). The trade in these and other gems was widespread: New Mexican turquois reached Yucatan; Colombian emerald was traded extensively to Peruvian Indians; and Minnesota catlinite was carried as far as New York and Georgia.

PROPERTIES AND CLASSIFICATION

With few exceptions, the materials used for gem purposes are minerals. Pearl, amber, jet, and coral, organic in origin, are exceptions. Certain rocks are also gems. To be classified as gems, minerals must combine the following significant qualities of splendor or beauty, durability, and rarity. Less important factors are fashion, portability, and world economic conditions.

Splendor in a gem depends upon its transparency, crystal structure, brilliance, luster, and color (*9*, *17*). The beauty of a gem depends to a large extent upon color. Indeed, some gems—for example, turquois—depend solely on color for their charm and appeal. The majority of gem minerals are transparent. Luster, a function of the transparency, refractivity, and crystal structure of a mineral, is either metallic or nonmetallic.

Durability, determined by hardness, is important to a gem. Often a beautiful mineral is too soft to resist normal abrasion that tends to dull the polish. For this reason, to wear well and remain beautiful for any considerable length of time, a gem must be fairly hard. All of the softer stones are classed as semiprecious, as are some of the more common harder stones. The precious stones—diamond, ruby, sapphire, and emerald—have a greater hardness than quartz.

Rarity is a value attributed to gems. If gems were common they would lose much of their value. It is natural to hold scarce things in high regard. Thus, some gems that are beautiful and durable but not rare have relatively little appeal; garnet is an illustration.

Of the 1,500 mineral species, only about 100 have all of the attributes required in gems (*12*). The silicates furnish the greatest number, including beryl (emerald and aquamarine), tourmaline, and topaz. Oxides, such as corundum (ruby and sapphire), quartz (agate, etc.), and opal, are also important. Sulfides, carbonates, and sulfates are of minor importance. Turquois and variscite are phosphates.

Gem materials can be classified into (*a*) group, (*b*) species, and (*c*) variety (*5*). Group refers to two or more gem minerals that are similar in crystal structure and physical properties but have different chemical properties. The individual members of the group are the "species." Varieties of a species have similar crystal structure and chemical characteristics but differ in color. It has become customary for laymen to classify gem materials as follows:

Precious Stones

Diamond.
Corundum (ruby and sapphire).
Beryl (emerald).

Semiprecious Stones

- Anatase.
- Andalusite.
- Apatite.
- Axinite.
- Azurite.
- Benitoite.
- Beryl:
  - Aquamarine.
  - Golden beryl.
  - Heliodor.
  - Morganite.
  - Goshenite.
- Beryllonite.
- Brazilianite.
- Calcite.
- Cassiterite.
- Chlorastrolite.
- Chrysoberyl:
  - Alexandrite.
  - Cat's-eye.
  - Chrysolite.
- Chrysocolla.
- Danburite.
- Datolite.
- Diopside.
- Dioptase.
- Enstatite.
- Epidote.
- Euclase.
- Feldspar:
  - Amazonstone.
  - Moonstone.
  - Sunstone.
- Fluorite.
- Garnet:
  - Almandine.
  - Andradite.
  - Grossularite.
  - Pyrope.
  - Spessartite.
  - Uvarovite.
- Gold.

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

- Gypsum.
- Hematite.
- Idocrase (vesuvianite).
- Iolite.
- Jade:
  - Nephrite.
  - Jadeite.
- Kornerupine.
- Kyanite.
- Lazulite.
- Lazurite (lapis lazuli).
- Malachite.
- Moldavite.
- Obsidian.
- Olivine:
  - Chrysolite.
  - Peridot.
- Opal.
- Phenacite.
- Prehnite.
- Pyrite.
- Quartz:
  - Agate.
  - Agatized wood.
  - Amethyst.
  - Aventurine.
  - Bloodstone.
  - Carnelian.
  - Cat's-eye.
  - Chalcedony.
  - Chrysoprase.
  - Citrine.
  - Eye agate.
  - Gold quartz.
  - Jasper.
  - Moss agate.
- Quartz—Continued
  - Onyx.
  - Plasma.
  - Prase.
  - Rock crystal.
  - Gold quartz.
  - Rose quartz.
  - Rutilated quartz.
  - Sardonyx.
  - Siderite.
  - Smoky quartz.
  - Tiger's-eye.
- Rhodizite.
- Rhodonite.
- Rutile.
- Scapolite.
- Sepiolite.
- Serpentine.
- Sillimanite.
- Smithsonite.
- Sodalite.
- Sphalerite.
- Sphene (titanite).
- Spinel.
- Spodumene.
- Staurolite.
- Steatite.
- Thomsonite.
- Topaz.
- Tourmaline.
- Turquois.
- Variscite.
- Willemite.
- Zircon.
- Zoisite.

Organic Gem Materials [1]

- Pearl.
- Coral.
- Amber.
- Jet.

[1] Not minerals.

GEOLOGIC OCCURRENCE

The great number of gem minerals implies a variety of occurrences and origins (*1, 4, 9, 15*). Gem minerals are formed principally by metamorphic processes, crystallization from fusion, and precipitation from solution.

Usually, gem materials are obtained from igneous rock formations, including pegmatite dikes. The gems can be found disseminated throughout the rocks, in other minerals, or as attached crystals in the rocks. Few gems are found in extrusive igneous rocks. However, basic igneous intrusive rocks have been a principal source of gems based on valuation, if the diamond production from the South African kimberlite pipes and the placers derived from them are included. Pegmatites furnish a variety of different precious and semiprecious stones, namely, emerald, beryl, tourmaline, topaz, spodumene, and quartz.

Stream placers are by far the most important secondary source of gems. Ruby, sapphire, tourmaline, agate, jade, and rock quartz, are obtained from alluvial deposits. Their original site was in pegmatite dikes or metamorphic rocks, from which they were released by weathering and collected by stream action. Most gems are heavy, physically resistant, and chemically inert and in consequence can resist abrasion and transfer by water. Beach placers, formed along seashores by the concentrating effects of wave and shore action, also contain gem stones. Zircon, diamond, garnet, and quartz are a few of the gem materials found in beach deposits.

Gem stones are also found in metamorphic rocks.

GEOGRAPHIC DISTRIBUTION

In the past 150 years the United States has produced a large quantity of numerous varieties of gem stones but has never been an important factor in world gem production. The United States is poor in precious stone resources, having no deposits of major importance. However, few countries have produced as great a variety of gem stones, essentially semiprecious, as has the United States. More than 60 different gem minerals have been produced commercially from domestic sources.

At least one kind of gem stone occurs in every State in the United States. The geographical distribution of several of the more important gem stones (namely, agate, topaz, rose quartz, jade, turquois, sapphire, ruby, tourmaline, and diamond) in the United States is described below:

Agate is widespread in occurrence and the most widely used gem material in the United States. The Oregon coast is regarded as the most productive area for agate. The Alpine-Big Bend area, Presidio and Brewster Counties, Tex., also has been a productive agate area in the past few years. Other relatively productive areas are Deming, N. Mex.; and Saddle Mountain, Ariz., covering parts of Maricopa, Pinal, and Graham Counties. Large quantities of agate also have been found in the southern part of California, mostly in Imperial and Riverside Counties, and in Washington and the eastern part of Oregon. Agate occurs in the stream beds of many rivers in Colorado, Montana, and other regions of the Rocky Mountains. Large quantities of agate are cut, polished, and mounted by lapidary shops in Portland, Oreg.; Seattle, Wash.; and various cities in California and Montana.

Montana was an outstanding supplier of domestic moss agate cut for jewelry from pebbles and cobbles found along the Upper Yellowstone River and its tributaries, but most areas in this State have been depleted. A major portion of this gem material was cut locally, mainly in Billings, Livingstone, and Glendive, and sold in roadside gem shops. Lapidaries in the Black Hills area depended on Montana for rough moss agate, claiming that it was much better than moss agate found in South Dakota.

The Streeter-Kotempsie area of Mason County, Tex., has been known to produce sizable quantities of gem-quality topaz at various times over the past 50 years. Topaz has been found also in California, Colorado, Connecticut, Idaho, Maine, Montana, Utah, and New Hampshire.

The largest deposit of high-grade rose quartz in the United States, if not the world, is 8 miles from Custer, S. Dak. At the Scott mine, the only sizable commercial producer in the United States, the quartz occurs in a pegmatite dike. Rose quartz also has been found near West Paris, Maine.

Jade (nephrite and other varieties) has been found at Green Mountain, Wyo., in a 200-mile circular area south and east of Lander, Fremont County, and on the Seward Peninsula, Alaska. Smaller quantities of poorer quality jade are found in San Benito and Monterey Counties and near Porterville, Tulare County, all in California.

The western part of the United States contains many turquois localities. Occurrences in Arizona are near Tombstone, Cochise County; Mineral Park, Mohave County; and Pierce, Maricopa County. The most important area lies east and south of Mineral Park. In Nevada the principal deposits of turquois are in Esmeralda, Lander, and Nye Counties. A greater quantity of turquois has been produced in New Mexico than any other State. Production has come principally from four districts: Cerrillos Hills, Santa Fe County; Burro Mountains and Little Hachita Mountains, Grant County; and the Jarilla Hills, Otero County. The Cerrillos and Burro districts are perhaps the best known. Turquois is found in smaller quantities near the King mine at Manassa, Conejos County, Colo.; San Bernardino and Fresno Counties, Calif.; and Culberson County, Tex.

The more important sapphire localities in the United States are in Fergus and Granite Counties, Mont. Yogo Gulch in Fergus County furnished the finer quality sapphires. In recent years no production has been reported. A few small sapphires have been found in Adams and Washington Counties, Idaho, and the Piedmont belt of North Carolina and Georgia. A few localities in this belt, especially in North Carolina, have yielded ruby. Corundum Hill, near Franklin, Macon County, has been a major producer of ruby in North Carolina. Sapphire and ruby have been found in Alabama, New Mexico, South Carolina, and Wyoming.

Deposits at Mount Mica in Maine, discovered in 1820, have been worked in a desultory manner, principally for tourmaline. Gems from Mount Mica are of the highest quality and have not been surpassed by any domestic tourmaline. Other localities in Maine, notably Mount Apatite, have also yielded tourmaline. The tourmaline belt in San Diego and Riverside Counties, Calif., has provided good gems but not equal to those from Mount Mica. Mesa Grande, San Diego County, has yielded the largest quantity and finest crystals that have been produced from California. Colorado, Connecticut, Massachusetts, New York, New Jersey, Pennsylvania, New Mexico, Virginia, and Maryland also have minor deposits of tourmaline.

Small quantities of diamond have been found in the United States from North Carolina to California and from Georgia to Wisconsin. The Great Lakes region is the largest diamond field in the United States. Diamond has been found in the kettle moraines or debris of glacial activity in an area 600 miles long and 200 miles wide in a northwest-southeast direction from Milwaukee to Chicago. Approximately 50 fine diamonds have been uncovered in this region. In the Appalachian region diamond has been reported from Virginia, West Virginia, North Carolina, Alabama, Kentucky, and Tennessee. The only location where diamond has been mined commercially is near Murfreesboro, Pike County, Ark. At various times since 1906 mines have been in operation. About 48,000 diamonds have been obtained from Arkansas mines, averaging ¼ carat, but 1 weighing 40.23 carats was found in 1924.

Localities in which gem materials have been found occasionally are listed in the following periodicals:

Mineralogist Magazine, 329 SE 32d St., Portland 15, Oreg. (published monthly).
Rocks and Minerals, Box 29, Peekskill, N. Y. (published bimonthly).
Gems and Gemology, Gemological Institute of America, 541 S. Alexandria Ave., Los Angeles 5, Calif. (published quarterly).
Lapidary Journal, Box 518, Del Mar, Calif. (published bimonthly).
Gemmologist, N. A. G. Press, Ltd., 226 Latymer Court, Hammersmith, London (published monthly).

A detailed list of gem materials and their localities in the United States is given in Gem Hunter's Guide (*11*).

GEM-STONE CUTTING

Primitive men based selection of a stone for gem purposes solely on a consideration of color. Before the development of faceted cutting, gem cutting consisted of smoothing or rounding and polishing the gem materials by crude methods (*7*). Today, this is known as the cabochon cut and is especially suitable for gems that have a sheen, a play of colors, opalescence, or asterism, as in star sapphires and rubies. Stones depending upon color or peculiar mottlings or markings for charm also are cut in cabochon style; however, cabochon cuts do not show the beauty of transparent stones.

The art of faceting was employed in Europe over 500 years ago, and discovery of this method of gem cutting is credited to Louis de Berquen, of Burges, Belgium (*9*). Thus, the lapidary art became a science, dependent upon mathematical relationships. Facet cutting is confined mainly to transparent gems having a hardness of 7 or more. Diamond and zircon, for example, are always cut in facet styles to bring out the brilliance and play of color of the stones. The index of refraction of the mineral and the degree of dispersion of light influence the choice of facet cuts. The cut known as "brilliant" best brings out the fire and brilliance of diamond and other precious gems. At present, the brilliant cut has 57 facets, 33 above the circle "girdle" and 24 below it, arranged in eightfold symmetry.

The principal styles of cut used today may be subdivided as follows (*12*):

1. *Bounded by plane surfaces.*
    - Brilliant cut.
    - Double brilliant or Lisbon cut.
    - Half brilliant or single cut.
    - Trap or split brilliant cut.
    - Portuguese cut.
    - Star cut.
    - Rose cut or briolette.
    - Step brilliant or mixed cut.
    - Table cut.
2. *Bounded by curved surfaces.*
    - Double cabochon cut.
    - Single cabochon cut.
    - Hollow cabochon cut.
3. *Bounded by curved and plane surfaces.*
    - Mixed cabochon cut.

A distinction is made in the gem-cutting industry between "diamond cutters" and "gem cutters" or "lapidaries" who fashion stones other than diamonds.

Diamond cutters are highly skilled individuals. This profession can be subdivided into five activities. The least spectacular, but in some ways the most important, are the *inspector* and *marker* (*9, 15*). The rough stones are carefully examined for crystal form and shape and marked with India ink, indicating the parts of the crystal to be removed or utilized. After notching, a *cleaver* splits the diamond along the lines of growth, taking great care in preparing the crystal. The rough stone is then handled by the *cutter*. The diamond cutter has two principal tasks, to remove imperfections and to give the stone a desired shape or outline without destroying the gem. Finally, the facets are cut or ground by the *polisher*. Faceting is done on horizontally revolving iron wheels treated with a crushed-diamond compound and oil.

No further treatment is needed after polishing except cleaning in boiling acid. Then the stones are sorted and ready for marketing.

Cutting of cabochons is a relatively simple process and is practiced extensively by amateur craftsmen, many of whom also do faceting. It does not require long experience to produce beautifully polished stones for jewelry or display. Gem-stone cutting has become a popular hobby. Many colleges and high schools have added lapidary courses to their list of handicraft classes.

There are four main operations in producing a cabochon gem: Sawing, grinding, sanding, and polishing (*5*). Sawing is done when using large, rough masses or where a large flat surface on a specimen is to be polished. If small pebbles or fragments are worked, the operator can go directly to grinding. Two general methods of sawing are "mud" sawing and "diamond-charged" sawing.

Grinding is the next step after the specimen has been sawed. Gems of different hardness require different grades of grinding wheels. However, two grits of silicon carbide wheels (namely, No. 120 and No. 220) are usually used in the home shop. Wheels 1-inch thick and ranging in diameter from 6 to 12 inches are used, but 8- and 10-inch wheels are most satisfactory. Dull-looking matrices and blemishes are removed in the grinding operation.

Sanding is done to remove the deep scratches made in grinding. Two types of sanders, the horizontal- or vertical-running sander and the drum sander, are used. Usually felt cloth or canvas is coated with No. 220 grit silicon carbide, but aluminum oxide and emery cloth can be used for sanding stones with a hardness less than 7.

Polishing is the final step and brings out the true beauty of a stone. Polishing is usually done on a revolving, hard-felt lap, either horizontal or vertical, or a muslin buff. The hard felt is recommended for general use. Many of the softer stones polish best on the muslin buff. Some commercial lapidaries employ a leather buff. Many polishing agents are in use, including tin oxide, tripoli, cerium oxide, alumina, and rouge; in general, manufactured polishing compounds have uniform grit and are free from foreign materials.

PRODUCTION

For many years the United States has produced a large variety of gem materials but never has been an important factor in world gem production. Domestic gem mining has been, and probably will continue to be a minor mining industry, dealing mainly with semiprecious stones. There are no large gem-mining companies in the United States. The greatest factor in gem-stone production is the thousands of amateur "rockhounds" and lapidaries that spend their weekends and vacations searching for gem materials. Reliable statistics do not exist for domestic production, and an estimated value of production in the United States in 1953 was $400,000 to $500,000.

IMPORTS[3]

The United States depends largely upon imports for precious and many semiprecious stones. Imports of gem stones, exclusive of industrial diamonds, in 1953 totaled $130,182,050, a 44-percent increase over 1952.

In 1953, as in previous years, diamonds were the leading import, representing 83 percent of the total value. Distribution of imports into the United States, by country of origin, is shown in table 1.

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[3] Figures on imports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

TABLE 1.—*Diamonds (exclusive of industrial diamonds) imported for consumption in the United States, 1952–53, by countries*

[U. S. Department of Commerce]

Country	Rough or uncut			Cut but unset		
	Carats	Value		Carats	Value	
		Total	Average		Total	Average
1952						
Australia				142	$41,882	$294.94
Belgium-Luxembourg	4,852	$430,417	$88.71	186,682	22,956,814	122.97
Bermuda	9,545	300,102	31.44			
Bolivia	71	2,119	29.85			
Brazil	9,719	479,114	49.30	2,056	242,763	118.08
British Guiana	1,061	53,855	50.76	22	3,349	152.23
British Malaya	1,723	115,367	66.96			
Canada	3,847	383,463	99.68	169	36,694	217.12
Denmark				15	2,528	168.53
France	50,490	1,075,560	21.30	784	321,310	409.83
French Equatorial Africa	13,976	396,924	28.40			
French Morocco				6	602	100.33
Germany, West				17,658	1,364,251	77.26
India				2,821	25,539	9.05
Indonesia				14	2,532	180.86
Israel and Palestine	1	47	47.00	128,206	10,017.374	78.13
Italy	359	60,808	169.38	187	29,641	158.51
Japan				5	1,250	250.00
Lebanon				7	1,573	224.71
Mexico				111	13,143	118.41
Netherlands	2,271	219,467	96.64	33,636	4,246,138	126.24
Portuguese Asia, n. e. c.				1,021	7,476	7.32
Surinam	135	8,999	66.66			
Switzerland	[1] 94,052	[1] 6,997,845	[1] 74.40	3,319	582,130	175.39
Thailand	1,338	153,564	114.77	968	134,883	139.34
Union of South Africa	[1] 48,515	[1] 1,276,049	[1] 26.30	54,011	10,737,727	198.81
United Kingdom	[1] 440,162	[1] 39,387,889	[1] 89.48	6,706	902,044	134.51
Venezuela	26,926	851,032	31.61			
Total	[1] 709,043	[1] 52,192,621	[1] 73.61	438,546	51,671,643	117.82
1953						
Argentina				18	4,263	236.83
Australia				3	1,771	590.33
Belgian Congo				300	63,603	212.01
Belgium-Luxembourg	19,116	1,818,663	95.14	215,438	25,554,634	118.62
Bermuda	8,985	502,677	55.95			
Brazil	398	99,448	249.87	34	8,722	256.53
British Guiana	2,307	83,958	36.39	30	2,847	94.90
British Malaya	560	65,162	116.36			
British West Africa	121	726	6.00			
Canada	4,744	448,338	94.51	95	88,241	928.85
Dominican Republic				1	235	235.00
France	11,631	280,922	24.15	1,098	170,306	155.11
French Equatorial Africa	39,963	940,002	23.52			
Germany, West	167	1,667	9.98	22,196	1,633,341	73.59
Gold Coast	450	4,219	9.38			
Hong Kong				1	93	93.00
India				2,974	52,853	17.77
Israel and Palestine				122,218	10,276,874	84.09
Italy				48	30,647	638.48
Japan				55	4,919	89.44
Mexico				165	10,239	62.05
Netherlands	4,171	374,437	89.77	29,365	3,491,370	118.90
Switzerland	7,820	841,026	107.55	493	171,765	348.41
Union of South Africa	56,592	2,336,405	41.29	46,441	8,455,833	182.08
United Kingdom	524,826	47,625,107	90.74	3,271	526,641	161.00
Venezuela	51,779	1,587,872	30.67	3	745	248.33
Total	733,630	57,010,629	77.71	444,247	50,549,942	113.79

[1] Revised figure.

TABLE 2.—*Precious and semiprecious stones (exclusive of industrial diamonds) imported for consumption in the United States, 1952–53*

[U. S. Department of Commerce]

Commodity	1952		1953	
	Carats	Value	Carats	Value
Diamonds:				
Rough or uncut (suitable for cutting into gem stones), duty free	[1] 709,043	[1] $52,192,621	733,630	$57,010,629
Cut but unset, suitable for jewelry, dutiable	438,546	51,671,643	444,247	50,549,942
Emeralds:				
Rough or uncut, duty free	8,790	22,213	15,561	27,987
Cut but not set, dutiable	11,162	449,726	26,952	320,739
Pearls and parts, not strung or set, dutiable:				
Natural	----------	465,165	----------	264,873
Cultured or cultivated	----------	3,373,383	----------	3,769,758
Other precious and semiprecious stones:				
Rough or uncut, duty free	----------	226,632	----------	203,667
Cut but not set, dutiable	----------	2,125,456	----------	2,218,868
Imitation, except opaque, dutiable:				
Not cut or faceted	----------	97,502	----------	40,720
Cut or faceted:				
Synthetic	----------	[1] 536,047	----------	677,029
Other	----------	[1] 13,413,526	----------	14,872,795
Imitation, opaque, including imitation pearls, dutiable	----------	39,142	----------	127,641
Marcasites, dutiable:				
Real	----------	75,285	----------	94,813
Imitation	----------	11,061	----------	2,589
Total	----------	[1] 124,699,402	----------	130,182,050

[1] Revised figure.

Emerald is imported mainly from Colombia. Some good-quality emerald has been mined in India, South Africa, and Brazil.

Pearl imported for consumption came mainly from Japan in 1953, while the other values shown in table 2 are for a variety of gem stones imported from numerous countries, of which Ceylon, Burma, and South Africa are important.

Imported gem stones are subject to tariffs, which vary for the different types of gem stone and the degree of manufacture.

USES

The greatest consumption of gems is for decorative purposes, mainly in jewelry, but to a lesser extent as material for statuettes and vases. Bracelets, brooches, hair ornaments, necklaces, earrings, rings, jeweled flowers, clips, and jeweled watches are popular. The finer gems—diamond, ruby, sapphire, emerald, and pearl—are the most prized for jewelry. Among some 20 other gem stones used, amethyst, aquamarine, turquois, and topaz rank high.

In the past, gem stones have had a variety of industrial uses, which have been largely displaced by synthetic materials. Jewel bearings made from synthetic sapphire and ruby are used for meters and various scientific instruments, such as timepieces, mechanical fuses for bombs, switches, and microgears. Tourmaline is used for laboratory tongs and agate for mortars, pestles, textile rollers, and spatulas. Other gem materials are used by industry in numerous ways.

One outstanding use of gem stones is for gem collections. Enthusiasm for collecting gem stones by thousands of amateur gem collectors has grown into a widespread hobby and commercial activity.

VALUE

The value of a gem stone is determined by three main natural characteristics—beauty, durability, and rarity. A fourth characteristic is the perfection of its cutting. Less important are portability, international markets, tariffs, and world economic conditions. Fashion, superstition, nationalistic sentiment and publicity also affect gem stone values, especially those of the less valuable gems (*3*, *9*, *13*, *14*).

Diamond, ruby, sapphire, and emerald are sold by the carat. About 1592 a famous Dutch traveler in the Orient formulated a rule for valuation, namely, to find the value of a stone weighing more than 1 carat, multiply the square of its weight by the value of a 1-carat stone of the same quality (*4*). This rule is used occasionally for ruby and emerald values but has been discarded in diamond valuation. The per carat price of a diamond weighing 3 carats may be twice that of an equally fine diamond of 1 carat; that of a 5-carat diamond three times, and that of a 20-carat diamond four times.

Values assigned to semiprecious stones are subject to the human element. These gems are given a pecuniary value, plus an aesthetic value. Prices quoted by one dealer may differ greatly from those of another dealer on similar stones. Collectors of gem stones regard the value expressed in dollars and cents as secondary. The real value is expressed by pleasure and personal pride gained from pursuit and acquisition of gem stones for personal collections. Small and large collections provide knowledge and enjoyment to collectors, as well as their visitors. Many leisure hours can be spent in collecting, cutting and polishing, and arranging a worthwhile gem collection. Additional satisfaction is gained from the growth of the collection, stone by stone. Amateur gem-stone collectors, comprising the major activity of the gem-stone industry in the United States, consider a gem collection an investment holding not only a market value but also a personal value of greater importance.

SYNTHETIC GEMS

Synthetic gems have the same chemical composition as natural gems and are similar in all other respects except mode and place of origin (*9, 10, 15*). Although synthetic gems may be made in almost any shade or tint, they are principally of four groups—corundum (sapphire and ruby), spinel, rutile, and beryl.

Industry has conducted extensive research projects on the production of synthetic gem stones and market applications. The first corundum to be produced synthetically was ruby in 1902; sapphire was produced 8 years later. Since that time synthetic ruby and sapphire have been put on the market in an array of colors.

Synthetic gem stones compete with natural stones for markets. The popularity of manufactured gems has been stimulated by the fact that synthetic gem stones cost less per carat. The scarcity of domestic sources of natural gem stones increased research in synthetic gems.

Although the United States became self-sufficient in producing synthetic corundum during World War II, European producers in 1950 recaptured the major portion of the American market for synthetic corundum and spinel. The more important manufacturing centers are Monthey and Locarno, Switzerland; Idar-Oberstein district, in the French Zone of West Germany; and Annecy, France. Synthetic gem-stone material (boule) and finished gem stones have been produced in the United States by Linde Air Products Co., a division of Union Carbide & Carbon Corp., Chicago, Ill.; Stuart Laboratories, Inc., New Jersey; National Lead Co., New York; and the Chatham Research Laboratories, San Francisco, Calif.

EFFECT OF WAR ON GEM-STONE INDUSTRY

The gem-stone industry, both foreign and domestic, was greatly affected by World War II, when supplies of sapphire, ruby, and cut zircon from Thailand, rough zircon from Indochina, cultured pearl from Japan, and sapphire, ruby, and jade from Burma were cut off from the United States. Furthermore, imports of gems from India and Ceylon were interrupted.

Imports of natural, synthetic, and imitation jewels from Germany, France, Czechoslovakia, Italy, and Switzerland became difficult to obtain. Destruction of the cutting industry in the Low Countries doubled or even quadrupled the price of small-cut diamond. The price of fine, large stones, however, only increased approximately 20 percent, inasmuch as such goods could be profitably cut in England, South America, and South Africa. India replaced Paris as an important center of fine-gem trading while Paris was under German control.

Before World War II, Bombay, one of the more important precious stone markets of the world, had virtually no direct trade with the United States. Its stones were exported to London or Paris, where American gem merchants purchased their requirements. Bombay does not control the output of any important gem-stone deposits but was a junction point in world circulation of precious stones. However, conditions have changed since World War II. Burmese ruby cut in Burma and India, sapphire from Burma, Ceylon and Kashmir, and emerald from Ceylon now can be purchased in Bombay.

South America (essentially Brazil) supplied the United States with aquamarine, topaz, citrine, amethyst, and tourmaline.

Domestic production of gem stones decreased considerably during the war years. Miners were employed in mining strategic minerals. Gas rationing reduced not only collecting by amateurs but travel by tourists, who comprise a large market for gem materials cut and polished by small shops lining the highways of the Western States. American lapidaries worked overtime cutting South American rough and recutting stones recovered from old jewelry into modern shapes.

OUTLOOK

Currently the supply of gem stones is sufficient to meet the demands of industry. The United States depends on foreign sources for precious stones. Domestic production principally is furnished through amateur gem-stone enthusiasts and consists mainly of semiprecious stones. No significant changes in these supply factors are anticipated.

Operating under a prosperous national economy, the jewelry industry will continue to use large quantities of gem stones. In the future industry will continue to promote jewelry through fashion shows, advertising, and style designing to emphasize jewelry made from gem stones.

Aside from the commercial aspect, activities of amateur collectors are continuously growing. Increasing numbers of people are becoming interested in gem stones for collection items or to fashion into jewelry. In many cities and towns, individual collectors are joining into organizations. Through the efforts of these societies, public interest in mineral and gem exhibits, gem hunts, and other gem-stone activities is being intensified. Also, gem hunts may disclose new sources of strategic minerals. Many civic organizations are sponsoring instruction in lapidary work and jewelry making. These classes are becoming increasingly popular. Through the efforts of the Gemological Institute of America and the American Gem Society, many persons have become informed on scientific aspects of gems.

Jewelry demands and amateur collectors' activities will continue at a high rate, giving greater recognition to gem-stone production. Supplies and requirements for commercial and hobby aspects of this industry will continue to increase.

PROBLEMS

Industry problems originate from a deficiency in domestic high-quality gem-stone deposits and competitive labor costs. Scarcity of domestic deposits necessitates dependence on imports for the major portion of the gem-stone supply. Highly skilled hand labor is essential to the industry, principally in cutting and fashioning the gem stones. Mechanization cannot be substituted for many operations. Lower labor costs in foreign countries hinder establishment of an extensive commercial gem-stone-cutting industry in the United States.

A shortage of publications on the location, technology, and marketing of domestic gem stones has been a problem for amateur collectors and lapidaries.

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GERMANIUM

By

John D. Sargent[1]

BORN OF RESEARCH begun in World War II, germanium semiconductor devices are implementing a revolution in electronics that is destined to give germanium a new economic stature and remove it from the category of "rare" metals.

Summary

The principal domestic source of germanium has been a zinc sulfide (sphalerite) concentrate which contains about 60 percent zinc and 0.01 to 0.015 percent germanium.

In metallurgical recovery of germanium, zinc concentrate is roasted to drive off sulfur and convert zinc, germanium, and cadmium to oxides. The oxides are changed to chlorides, and germanium tetrachloride is hydrolyzed in ice water to obtain pure germanium oxide, which, in turn, is reduced to the metal by heating in hydrogen.

Domestic production of germanium on an industrial basis began in 1942 when the Eagle-Picher Co. erected a germanium recovery plant at its Henryetta zinc smelter.

The principal use of germanium is in the field of electronics, by virtue of its being a semiconductor. Germanium diodes, transistors, and rectifiers, which can perform most of the functions fulfilled by vacuum tubes, are smaller, lighter, much less fragile and potentially cheaper than their tube counterparts. They require no hot filament or hot cathode, hence require much less power to operate, have apparently an indefinitely long life, and can perform some functions that vacuum tubes cannot. To their development is due the much heralded "revolution in electronics."

Current and estimated future supplies of germanium would appear to be ample both for immediate requirements for the metal and the early requirements resulting from new applications in the field of electronics.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

GENERAL

Winkler, a German chemist, isolated germanium from the mineral argyrodite, Ag_8GeS_6, in 1886; Mendeleev, in propounding his periodic table in 1871, had predicted discovery of the metal (*6*).[2]

Germanium metal is a gray-white crystalline solid, which retains its fine luster in air. It is a brittle substance, crystallizing in regular cubes (*3*). The atomic weight of germanium is 72.59; its specific gravity, 5.32; and its melting point, 958.5° C. (*6*). Estimates of the germanium content of the earth's crust vary from 0.0001 to 0.0007 percent; 0.0004 percent may be taken as a fair average, which is a higher content than that of gold, silver, bismuth, mercury, and boron, and nearly equal to that of lead, zinc, and tin.

Germanium is never found in the free state but always in combination with other elements as a mineral. The principal germanium minerals are argyrodite (Ag_8GeS_6; 6 to 7 percent germanium) and germanite ($7CuS \cdot FeS \cdot GeS_2$; 8.7 percent germanium). Both minerals are, however, relatively scarce. Argyrodite was found in the Himmelsfurst mine at Freiberg, Saxony, and is associated with other silver minerals at Colquechaca, Bolivia (*3*). Germanite occurs in ores of the Tsumeb mine in South-West Africa, and a variety of germanite, known as renierite, has been reported at Tsumeb and at the Prince Leopold copper mine of the Union Minière du Haut-Katanga, Kipushi, Belgian Congo.

The most common occurrence of germanium, however, is in very small quantities within the molecular structure of other minerals. In fact, it occurs most abundantly in the sphalerite (ZnS) of low-temperature deposits, in quantities ranging from traces to tenths of 1 percent. Cinnabar sometimes contains germanium in tenths of 1 percent and enargites contain up to 0.1 percent. Pyrargyrite (Ag_3SbS_3) from the silver mines of Bolivia contains up to 1.0 percent, and some tin-bearing sulfides have 0.5 percent germanium. Many other minerals contain germanium in hundredths of 1 percent. Certain coals also contain germanium.

The principal domestic source of germanium is the zinc sulfide (sphalerite) concentrate produced in the Tri-State lead-zinc mining district of Missouri, Kansas, and Oklahoma. The concentrate contains about 60 percent zinc and from 0.01 to 0.015 percent germanium.

METALLURGY

RECOVERY OF GERMANIUM FROM TRI-STATE CONCENTRATE

In the metallurgical recovery of germanium from Tri-State zinc sulfide concentrate, the sulfur in the concentrate is first removed by roasting, in the course of which the zinc, germanium, and cadmium contained in the concentrate are converted to the oxides of the respective metals. The oxides, with the addition of coal and salt (NaCl), are then sintered at high temperature, volatilizing the germanium, cadmium, and lead, which are condensed and collected in an electrostatic precipitator (*6*). Further processing removes the cadmium and lead. The germanium, in the form of germanium tetrachloride, is then recovered separately by distillation.

PRODUCTION OF HIGH-PURITY GERMANIUM DIOXIDE

Crude germanium tetrachloride is purified by multiple distillation, with additions of chlorine and hydrochloric acid. Purified germanium tetrachloride is then hydrolized in ice water to obtain pure germanium dioxide, which is washed completely free from chlorides. The pure, white-powder germanium dioxide is the product usually shipped to electronics manufacturers, who reduce it to metal.

REDUCTION OF GERMANIUM DIOXIDE TO GERMANIUM METAL

Germanium dioxide is reduced by heating in hydrogen. The dioxide is packed into graphite boats and heated in an electric furnace in a current of hydrogen at 650°–675° C. until no more water is evolved. During the reduction the temperature must be kept below 700° C. to avoid loss of volatile germanium monoxide formed in the first stage of reduction. After the reduction has been completed, the temperature is raised to 1,000° C. to melt the germanium-metal powder. Slow cooling or directional solidification is necessary to prevent formation of blowholes in the ingots. Extreme purity can be achieved by multiple-zone melting and fractional recrystallization of the ingots, causing impurities to be concentrated in one end, which is removed.

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

PRODUCTION AND CONSUMPTION

Domestic production of germanium on an industrial basis was begun in 1942. That year the Eagle-Picher Co. erected a pilot plant at its Henryetta, Okla., zinc smelter to treat cadmium byproducts and produced a small quantity of germanium metal, oxide, and chloride. Total shipments during the year represented less than 10 pounds of metal.

Through the 3-year period 1943–45 the domestic output of germanium continued at less than 100 pounds annually, with the Eagle-Picher Co. the sole producer. In 1946 United States production reached several hundred pounds, and output has since increased each year over the preceding year.

Until 1948 the Eagle-Picher Co. was the only domestic producer of germanium; beginning with that year the American Steel & Wire Co. (subsidiary of the United States Steel Corp.) has annually produced a comparatively small quantity of germanium dioxide at its Donora, Pa., smelter. In 1949 the National Zinc Co. also produced a small quantity of germanium dioxide at its Bartlesville, Okla., smelter. The American Zinc Co. placed a germanium-recovery unit in operation at its Fairmont City, Ill., plant in December 1952. In 1953 Sylvania Electric Products, Inc., initiated germanium production at Towanda, Pa., and the Eagle-Picher Co. began germanium production at a new germanium plant at Miami, Okla. (*2*).

RESERVES

Estimated reserves of more than 100 tons of recoverable germanium are contained in the Tri-State lead-zinc reserves. Large reserves of other germaniferous ores have been reported. Vast resources of germanium are available in low concentrations in coal deposits.

In Africa, both the Tusmeb Corp. and Union Minière du Haut-Katanga expect to be able to produce about 10 tons of germanium a year.

PRICES

As recently as 1940, germanium was a really rare metal and priced at $4,500 a pound or $308.56 a troy ounce—nearly 9 times the price of gold. By 1942, however, the metal was quoted in New York by suppliers of rare metals at $3 to $5 a gram or $1,360.80 to $2,268.00 a pound.

In 1945 the metal was quoted at about $200 a pound and in 1946 at about $180 a pound and the oxide at $50 a pound. In 1948 the quotation for high-purity metal increased to $230 a pound and spectrographically pure oxide to $65 a pound. The average price of germanium metal advanced $100 in 1949 to $330 a pound, and high-purity germanium dioxide sold for $65 to $70 a pound. Germanium metal was quoted in 1952 at $340 a pound and germanium oxide at $142 a pound. In 1953 and 1954 the price dropped slightly to $295 a pound for germanium metal (*2*).

USES

The Eagle-Picher Co. reported as early as 1942 that it was furnishing germanium to several universities and manufacturing concerns for use in connection with work for the Army and Navy. Investigation of the use of germanium in electronics was well underway at that time.

In electronics, germanium is used principally in diodes, transistors, and rectifiers, which can achieve most of the functions fulfilled by vacuum tubes and some functions that cannot be accomplished with them. Germanium diodes and transistors are smaller, lighter, and potentially cheaper than their tube counterparts. They require no hot filament or hot cathode, hence require much less power to operate, have apparently an indefinitely long life, and are shock resistant, an essential for use in projectiles. To their development is due the much heralded "revolution in electronics."

Two types of diodes and transistors are on the market at present—the point-contact or "catwhisker" type and the junction type.

The germanium diode, which derives its name from the two-element vacuum tube that it replaced for rectification purposes, is used in various types of electronic circuits, including radio, television, carrier-current (telegraphy and telephony), multiposition-switching, and voltage-multiplier circuits. Diodes have been made in small sizes by the millions for several years; about 6,000,000 were made in 1951 and over 10,000,000 in 1952 (*2*). Large sizes for high-power applications are in production.

The transistor (so named because it transfers an electrical signal across a resistor) was invented in 1948 by J. Bardeen and W. H. Brattain, of the Bell Telephone Laboratories. Invention of the transistor has been pointed to as an example of how research directed at basic understanding of materials and their behavior sooner or later brings to the view of inventive minds opportunities for producing valuable practical devices.

The contact-type transistor, which is essentially a triode form of the well-known crystal diode detector used in radio, can be used as a voice amplifier, a television-picture amplifier, a pulse amplifier, and an oscillator. A survey made in the first half of 1952 indicated that contact-type transistors were being produced at the rate of 8,400 a month, or about 100,000 a year, by 5 companies, comprising the General Electric Co., Syracuse, N. Y.; Radio Corp. of America, Harrison, N. J.; Radio Receptor Co., New York, N. Y.; Raytheon Manufacturing Co., Newton, Mass.; and Western Electric Co., Allentown, Pa. At present, several dozen manufacturers are producing germanium diodes, transistors, and rectifiers.

Germanium metal, up to several centimeters thick, will give appreciable transmission of infrared radiation over broad regions of the spectrum; lenses of the metal are being made for industrial infrared work. Film resistors of germanium are made by depositing the metal on ceramic bodies. Magnesium germanate is used as a phosphor. Germanium (12 percent) and gold form a eutectic that eventually may be used in jewelry as a gold solder and in dental inlays; the alloy melts at 673° F., compared to 1,945° F. for pure gold, expands slightly on solidification, and is the color of 18-carat gold. A glass of high refractive index in which germanium substitutes for silica (the other constituents being TiO_2 and NaF) may be used in wide-angle camera lenses and in microscope objectives. Germanium is also used as a catalyst in the hydrogenation of coal.

SUBSTITUTES

Germanium is only one of many elements and compounds classed as semiconductors. Each of the semiconductors has slightly different physical and chemical properties, and the best semiconductor for each individual application will be found in the normal course of research. Germanium is the most dominant of the semiconductive elements in use today and probably will never be displaced entirely. Substitutes are being developed, however, in those applications where germanium has been disappointing. The Texas Instrument Co. recently (1954) produced the first commercially available silicon transistors for operation at temperatures from 100° to 150° C. This company also manufactures silicon alloyed junction diodes.

Various combinations of arsenic and antimony alloyed with aluminum, gallium, and indium are being tested as semiconductors. Selenium, tellurium, cadmium sulfide, indium phosphide, copper oxide, titanium oxide, borides, nitrides, and carbides are also being investigated as semiconductor materials.

OUTLOOK

In May 1953 the Bureau of Mines conducted a germanium supply and demand survey. The total anticipated production of GeO_2 indicated was 23,330 pounds in 1953, 40,680 pounds in 1954, and 70,000 in 1955. Nonmilitary consumption of GeO_2 by the electronics industry was estimated to be 12,000 pounds in 1953, 22,000 pounds in 1954, and 39,000 pounds in 1955. Since the survey was compiled, the difference between anticipated production and anticipated consumption has been lessened because a few concerns abandoned plans for producing germanium and several others cut back on production. In addition, there are indications that electronic consumption has increased more than was indicated by the survey. Annual consumption of germanium is increasing rapidly, and the introduction of germanium substitutes in some semiconductor applications is not likely to cut into germanium consumption to any large extent. Domestic sources of germanium are adequate to meet the demand for the next several years, and large quantities of foreign germanium probably will be available for export to the United States.

The chemical industry may consume increasing quantities.

Germanium production from coal is not likely in the next few years but is possible in the more distant future.

PROBLEMS

Germanium is produced in greater quantities than can be consumed at present. Its high price prevents expansion of germanium utilization.

Before coal can be utilized as a commercial source of germanium, a method of obtaining a germanium-enriched byproduct from its combustion in power plants must be developed. The present overabundance of germanium has caused curtailment of research on the extraction of germanium from coal.

The life of the Tri-State lead-zinc field is limited. If other domestic sources of germanium have not been developed before the Tri-State reserves are exhausted, the United States will have to depend upon foreign sources.

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GOLD

By

James E. Bell [1]

BECAUSE of its beauty, scarcity, and imperishability, gold has been esteemed above all other metals throughout the ages. Men have roamed the world in search of gold, exploring faroff parts and pushing back frontiers. The winning of gold has been the forerunner of settlement, agriculture, and industry in many primitive lands.

Summary

Gold has been mined by mankind at an accelerated rate since the earliest civilizations. The total world production from 1493 to 1952 is estimated at about 1,750,000,000 troy ounces, of which approximately 15 percent has been lost or dissipated. Gold mining was begun on a small scale in the United States in the early 1800's; but after discovery of gold in California in 1848 and later in other States it became the principal gold producer of the world and held this position until the Transvaal, in South Africa, took the lead in 1905. In recent years the United States has ranked fourth in gold production, exceeded by Canada and (probably) the Soviet Union as well as by South Africa.

The total recorded gold output of the United States to 1953 was 285,169,756 ounces or 16 percent of the world total (since 1493). However, due to the flow of gold to the United States because of its dominant economic position during the past 3 or 4 decades, the country's monetary stocks of gold now far exceed its total output and on December 31, 1952, were approximately 662,464,000 troy ounces (valued at $23,186,000,000), equal to about 64 percent of the world's total monetary stocks (exclusive of the Soviet Union). In 1952 the recorded gold production of the United States was 1,893,261 ounces, of which 22 percent was recovered by placer mining, 40 percent from precious metal ore, and 38 percent as a byproduct of base-metal ore. Modern techniques of gold mining are very efficient.

The straight-gold-mining industry throughout the world has been beset for more than a decade by rising costs of labor and supplies, without a compensating increase in the price of its product. Partial relief has been afforded in some countries by subsidies, tax concessions, devaluation of currencies, and the sale of some current production at premium prices on the "free market." No assistance of this nature has been available to straight gold producers in the United States (except legality to sell at premium prices the small part of the total gold output that qualifies as "natural" gold), and a number of operations have closed or are barely able to keep going. There are also many prewar producers that did not resume operations suspended during World War II. Most of the mines that yield byproduct gold from base-metal ores are not in these straits, having the benefit of higher prices for their principal products. Straight-gold producers the world over, facing still more difficult times, are urging a higher price for gold, and many economists believe that the present price of $35

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[1] Commodity-industry analyst, Bureau of Mines.

will have to be raised in the not-distant future to insure a steady flow of new gold, deemed essential for maintaining the structure of world credits. However, a higher price in the United States can be brought about only through legislation by the Congress, and many authorities in this country hold that such action would be inflationary and inadvisable.

The principal use for gold today is as a monetary reserve of governments and central banks to give stability to paper currencies and to settle international trade balances. Considerable gold is used in the arts, particularly for jewelry, and a small quantity is used in industry.

It is interesting to note that the net consumption of gold in the arts and industries in the United States has considerably exceeded the output of domestic mines in recent years. Gold does not have many industrial uses that are indispensable in wartime, but gold reserves are important to nations at war to maintain solvency. The use of gold has become increasingly controlled by governments for currency and credit stabilization, and any program for gold must consider that the national interest will be more affected by monetary considerations than by the prosperity of the gold industry itself. Nevertheless, a healthy gold-mining industry is a very real source of strength in any national economy.

BACKGROUND

SIZE AND ORGANIZATION OF INDUSTRY

Gold has been highly valued by mankind since antiquity, and gold mining has been practiced since the first civilizations. Early mining methods were primitive and arduous, and the production rate was low. The rate of output increased rapidly upon development of rich deposits in the Americas after 1700. Gold production in the United States began in the Southern Appalachian States in the early 1800's upon discovery of gold in the payable amounts in North Carolina in 1801. The output was small, however, and furnished only a small part of the world gold supply. With discovery of gold placers in California in 1848 and other great deposits in various areas later, the United States (with Alaska) became the leading gold producer of the world, and held this rank during most years for over half a century (*14*).[2] Since 1905 South Africa has been the leading gold-producing country, followed by the United States until 1930, when Canada advanced to second place, then (probably) by the Soviet Union after 1934. Despite an increasing yield of byproduct gold from base-metal mining, the trend in gold production in the United States was generally downward during 2 decades following 1915, but straight gold mining recovered strongly after the price of gold was raised from $20.67 an ounce to $35 in 1934. Production reached an alltime high in 1940 and then dropped sharply during the war. The United States output had not recovered to even one-half the prewar level to 1953; nonetheless, domestic gold mining is still an industry of considerable national importance and the principal industry in some areas.

In 1952, 22 percent of the gold production of the United States was supplied by placer mining, 36 percent from underground mines operated primarily for gold with recovery by amalgamation and cyanidation, and 42 percent as a byproduct of base-metal mining operations (*3*). The 3 leading mines contributed 52 percent of the total gold produced in the United States in 1952 and the leading 25 mines, 82 percent. Most base-metal ores mined in the Western States contain some gold; but the bulk of byproduct gold is contributed by large-tonnage operations, particularly the low-grade copper mines, which owe their outputs to the quantity of ore handled rather than to a substantial gold tenor.

GEOGRAPHICAL DISTRIBUTION OF INDUSTRY

Gold is very widely distributed, being found in some quantity in nearly every country. During recent years gold has been produced in some 75 countries in all 5 continents and Oceania. In the United States gold has been produced in 24 States and Alaska. Over 99 percent of the total output has been mined in the Western States (including South Dakota) and Alaska and most of the remainder in the South Appalachian States; no gold production has been recorded in most of the Central States (*3*). The principal gold-producing areas of the United States may be further identified as follows: (1) Central Alaska, (2) Southeastern Alaska, (3) Sierra Nevada, (4) the Southwest, (5) Great Basin, (6) Rocky Mountains, (7) Black Hills, and (8) South Appalachian. All the Western States (including South Dakota) and Alaska have produced placer gold in considerable quantity, and the South Appalachian States have had a minor production.

TECHNOLOGY

Deposits containing gold occur in almost any rock and in formations of all geological ages from pre-Cambrian to late Tertiary; most of the known deposits are associated with mineralization of either very early or comparatively recent age. Nearly all metalliferous deposits containing gold are in the vicinity of acidic igneous intrusions, with which they show obvious genetic relationship (*7*). The general deposit types include the magmatic; contact metamorphic; replacement (lode, massive, and disseminated); and cavity filling (fissures, stockwork, saddle reef, breccia, and conglomerates). Gold, associated base-metal minerals, and gangue minerals occur in the primary deposits in a great variety of combinations, and weathering and alteration add still further to the variety. The richest straight-gold deposits probably have been of the small-fissure-vein type, with gangue composed predominantly of quartz. However, a great deal more gold has been won from large deposits of moderate to low grade, particularly during the past half century.

As deposits that carry gold in metallic form (rather than in chemical combination) are exposed at the surface by erosion, the rocks and most of the minerals they contain disintegrate and crumble, owing to both oxidation and weathering, thus releasing the particles of free gold. Because of its chemical and physical properties, the gold remains largely unaltered,

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

and most of it works downward to accumulate in nuggets and grains in stream beds, forming creek placers; some gold in fine particles may be carried to the sea and concentrated along beaches to become marine placers (*1*). Stream action may cut channels deeper and leave portions of creek placers above the later drainage level to form bench or terrace placers. Creek placers may also be covered through later sedimentation or lava flows to form buried placers, some portions of which have been exposed by subsequent erosion and worked with success by underground mining.

The techniques of mining gold ore in place are essentially similar to those applying to metalliferous ore deposits in general. The basic procedures of rock breaking, support of workings, transportation and hoisting, drainage, ventilation, etc., are identical. Selection of a suitable mining method is or should be based on the size, shape, attitude, and regularity of the ore body; the physical and mineralogical character of the ore body; the character of the wall rock; the presence of old workings; the kind of labor available; safety; the scale of operations; availability of capital; and various other factors. The goal is to obtain maximum ore extraction, low production costs, and the greatest possible financial returns. It doubtless is a fact that straight-gold ore is or has been mined underground by most of the recognized mining methods in some form, ranging from simple overhand stoping in open stopes to caving; and on the surface from hand methods to power-shovel excavation on a large scale. Base-metal ores containing byproduct gold are mined by an even greater variety of methods and modifications.

Gold placer mining comprises the excavation of placer gravel and transporting it to a washing plant to recover the gold it contains (*1, 8, 9, 10*). Devices for placer mining by manual methods, like the gold pan and rocker, are economic only for bonanza-grade gravel present at some placers when they are discovered and quickly worked out. Generally, successful placer mining is a matter of handling a large volume of low-grade material, with recovery of enough gold at a cost to permit realizing a profit. Placer gravel is excavated and fed to the washing plant by earth-moving equipment of the usual kinds, including horse-drawn scrapers, tractor-drawn scrapers, bulldozers, power shovels and trucks or side-dump cars, dragline scrapers, etc. Jets of water discharged at high pressure (hydraulic mining) were widely used in the past and were capable of moving very large volumes of gravel at low cost but are now unlawful in many areas because of damage by debris to lands downstream. When conditions are suitable, floating dredges combining excavating, transporting, washing, and tailing-disposal plants in one self-contained unit are very efficient, and some are made large enough to handle 12,000 cubic yards or more of gravel a day. At present most of the total placer gold produced is mined by bucketline dredges. The use of large dragline excavators to feed floating washing plants has grown for mining gravel deposits not adapted to dredging. Buried placers are opened by drift tunnels or shafts and mined by underground workings (drift mining), but operating costs are high. A substantial outlay is sometimes necessary at placer operations to insure an adequate supply of water for the washing plants. In some areas stripping of barren or low-grade overburden is practiced; and in regions with cold climates, such as Alaska, some gravels require thawing before they can be worked. Dry placer washing has been employed considerably in the Southwest where water is scarce, with a total production estimated at several million dollars of gold. Dry washers use pulsations of air through a porous medium instead of water to separate the gold from the gravel; operations are usually on a very small scale. Several companies in California and Colorado, recovering building sand and gravel from stream beds, also recover gold in economic amounts as a byproduct.

Most of the gold produced in straight-lode-gold mining operations is recovered by amalgamation or cyanidation or by a combination thereof; metallurgical recoveries are high, sometimes exceeding 95 percent. Amalgamation is a simple process applicable only to ores that carry the gold in metallic particles. Ground ore is passed over copper plates coated with mercury. The mercury amalgamates with the gold, and the resulting amalgam adheres to the copper plates; this is scraped off, and the gold is then recovered by retorting the amalgam. In simple cyanidation the finely ground ore is leached in tanks with a dilute solution of sodium cyanide to dissolve the gold, which is then precipitated from solution with zinc dust and the precipitate fluxed, melted, and cast into bars for shipment (*6, 11*). There are various modifications of these basic treatment processes. If the gold is associated with sulfides or is present as tellurides, the ore may be roasted before the amalgamation and/or cyanidation treatment. Some mines recover a gravity or flotation concentrate directly from the ore, treating this by roasting followed by amalgamation and/or cyaniding; others ship their concentrate to a custom mill or smelter, thus saving outlay for plant, but perhaps at higher operating cost. Massive sulfide ore usually is smelted directly if the gold values therein are relatively high. Some gold ore with gangue very high in quartz is shipped directly to smelters needing siliceous flux.

Byproduct gold from base-metal mines follows base-metal ores through all their processes of mining, direct shipping or concentration and shipping, and smelting and ultimately is recovered as gold bullion in the precious metals unit of base-metal refineries. Losses of gold values in concentrating usually are about the same as those of base-metal values; losses of gold values in smelting are very low.

The basic device usually employed in placer washing plants for separating gold from gravel is some form of the sluice. The sluice has been in use since antiquity and is essentially a launder or trough, usually of wood, through which gravel is washed by a stream of water. Sluices are fitted with obstructions or "riffles" on the bottom to break the current and provide a protected spot where gold can settle and remain undisturbed. Riffles may be almost anything but usually are made of wood, easily removable to facilitate recovery or cleanup of gold. Mercury is often placed in the riffles to aid in catching and holding the gold. Sluices are usually made in units 12 feet long, known as "sluice boxes." Depending on the capacity desired, they may be 1 foot to 6 feet wide and 1 foot to 3 feet deep. Sluice boxes are placed together end to end to form a total sluice length up to 200 or 300 feet. The sluices carried on dredges are relatively short and wide and known as "tables." At some operations, part of the gold occurs in particles of very fine size. To save these values, undercurrents are sometimes employed; these consist essentially of a wide sluice having a perforated false bottom through which the fine material may pass to the undersection, with matting, carpet, corduroy, or small riffles to catch the gold. Most placer operations employ grizzlies in some form to discard boulders and screens or trommels to disintegrate the gravel and eliminate the oversize portion. The size of gold particles in placers varies through nugget, shot, flaky, and flour. Although thousands of cubic yards of placer gravel may be washed through the sluice, recovery of coarse gold and moderately fine gold, where disintegration of the gravel is complete, may be as high as 90 percent or more. Recovery of flour gold is low, particularly where clay is present in quantity, but actual data in this respect are very meager (*1, 8, 9, 10*).

Jigs are being increasingly employed on some dredges and in washing plants in place of, or to supplement, tables. A heavy-sand concentrate containing gold is produced, from which the gold is recovered in amalgamating barrels.

The bulk of the output of placer gold, retort sponge, or bullion of domestic mines and refineries is sold to United States mints or United States assay offices at the Treasury price of $35 per fine troy ounce, less small charges for handling and refining. To be acceptable shipments are required to have a fineness in gold of at least 200 (200/1000). Some of the gold bullion produced by refineries of the larger private companies has a fineness of over 999.5 and does not require further refining. A minor part of the output of placer gold recoverable without processing involving melting or chemicals is sold on the free market; some refineries hold a dealer's license and sell part of their gold to consumers. Crude gold bullion is refined by chlorination in the molten state (Miller process) and by electrolysis (Wholwill process), The latter is used by the United States mints, giving a final bullion product 999.5 to 999.8 fine (*11*).

BYPRODUCTS AND COPRODUCTS

Silver is always associated with gold to some degree in gold deposits and is a coproduct of gold placer mining and lode-gold mining; the average ratio by weight is around 1 silver to 4 gold. Crude platinum is often present in gold placers and recovered as a byproduct; at some areas in Colombia enough is present that it becomes a coproduct. Byproduct cassiterite is recovered in small quantity in certain gold placers in Alaska. Platinum-group metals in very small quantities are present in some lode-gold ores and are recovered in the final refining of gold. An average of about 6,000 ounces of osmiridium is obtained annually as a byproduct of gold mining in the Union of South Africa; the average recovery is around 0.10 ounces of osmiridium per 1,000 tons of gold ore treated. Along with gold, the osmiridium is retained on tables covered with corduroy during milling operations and recovered after the gold has been removed by amalgamation. Uranium minerals also are present in small quantities in the gold ores of the Union of South Africa, and recovery of these as a valuable byproduct was begun recently. Recovery of rare-earth products (including uranium and thorium-bearing minerals) from heavy-sand concentrates in some localities is a new gold-placer-mining development.

SUBSTITUTES AND SECONDARY SOURCES

Platinum alloyed with iridium and palladium alloyed with ruthenium are substituted for gold in high-quality jewelry, watchcases, novelties, etc. This substitution is not due to considerations of unavailability of gold or lower cost but is dictated by popularity and vogue. Platinum leaf is substituted for gold leaf for signs and embellishments. Palladium alloys are substituted for gold in dentistry and stainless steel, chromium, and nickel alloys in corrosion-resistant equipment. The total quantity of gold displaced by substitution of other materials is small, however, compared with the total consumption of gold in nonmonetary uses.

Because of the indestructability and high value of gold, there is a constant return from industrial use of scrap gold, such as old jewelry, watchcases, optical frames, dental crowns, laboratory ware, etc. Lots containing not less than 1 fine troy ounce of gold may be sold directly to the mint at the standard rates of payments and charges. Smaller quantities may be sold to licensed dealers who base their payments on weight and purity, as determined by the color of the streak on a "touchstone"; accumulations by dealers are resold to the mint. According to the Bureau of the Mint, an average of $37,678,390 of scrap gold was returned from industry annually during the period 1948–52.

RESERVES

Based on an estimate as of January 1944 (*4*) the total inferred, indicated, and measured recoverable gold reserves of the United States and Alaska were 69,000,000 troy ounces on that date; this equals about 24 percent of the total domestic production to 1953. Distribution of the reserves, by States, was as follows: Alaska 20,000,000 ounces, Arizona 2,100,000, California 16,500,000, Colorado 5,200,000, Idaho 2,470,000, Montana 1,600,000, Nevada 2,350,000, New Mexico 280,000, Oregon 600,000, South Dakota 9,000,000, Utah 8,150,000, Washington 400,000, Eastern States 350,000. Of the total reserves, 21,000,000 ounces was estimated to be in placers, 37,000,000 in dry and siliceous ores, and 11,000,000 in base-metal ores, chiefly the disseminated-copper deposits. Additional reserves in extensions of known ore deposits and new discoveries of others have largely offset the production of gold since 1944. The question of gold reserves, however, is closely tied to economics—to the price of gold in relation to the cost of producing it.

Data on the gold reserves of most countries of the world are nonexistent or unavailable; in any case they would have only limited meaning because of uncertainties in the price-cost relationships of the future. Statements in trade journals suggest that total reserves in the Rand, West Rand, and the new Orange Free State districts in the Union of South Africa total over 500,000,000 ounces of gold. The gold reserves of the Soviet Union are believed to be very large, also. A total recoverable reserve of 1,000,000,000 ounces of gold for the world does not seem out of reason, provided there is enough incentive to mine it.

Gold is present in minute unit amounts in sea water, but the quantity contained in 1 cubic mile is vast (around $20,000,000). This fact has led to proposals to extract gold from the sea, but no processes tried to the present time have been successful.

Since the United States reached industrial and financial dominance after World War I, there has been an almost constant net inflow of gold to balance the excess of exports over imports, thus augmenting stocks accumulated from its own production. On December 31, 1952, the gold monetary reserves of the United States Treasury were $23,350,000,000. The monetary gold reserves of the rest of the world, excluding the Soviet Union, were estimated at $12,860,000,000.

USES

Normally, 90 to 95 percent of the gold production of the world is used for monetary purposes by governments and central banks, mostly as a reserve in the form of bullion, to give stability to paper currency, and to serve as the ultimate means of settling international balances. It is interesting, however, that the matter of the proper function of such reserves and even their indispensability is controversial among the experts. The remaining 5 to 10 percent of the gold production is consumed by the arts, sciences, and industry. In the United States, jewelry and allied articles, such as watchcases and expensive tableware, are the chief consumers. In dentistry gold is used for fillings and dentures. Gold alloy is used for hairsprings of marine chronometers, in galvanometers, and in various other delicate instruments because of its resistance to corrosion. Gold finds considerable use for chemical-plant and laboratory ware also because of its resistance to chemical action. During the past several years, an abnormally large quantity of gold has been finding its way into private hoards, and it is probable that such hoarding will continue for some years to come.

STATISTICAL CONSIDERATIONS

In the United States statistics on the domestic mine production of gold are gathered, analyzed, and compiled by the Bureau of Mines in cooperation with the geological surveys of some States. The coverage is rather more complete on gold than on many of the other mineral commodities; it permits regular publication of reports by the Bureau on the mine production of gold by mining districts, counties, and States (*3*). Figures published also include details on the source of gold with respect to ores of various types and the portions recovered by the various methods of treatment; figures on placer mining include data on production by various methods of working, with details on yardage treated and average unit yield. General data on the refinery production and on the consumption of gold are supplied to the Bureau of Mines by the Bureau of the Mint; data on foreign trade in gold ore and bullion are supplied by the United States Department of Commerce.

Detailed data on gold production are gathered by some countries of the British Empire, particularly Canada, and the Union of South Africa, and reliable summaries are published. The corresponding data for many countries, however, appear incomplete and even inconsistent.

PRODUCTION, CONSUMPTION, AND FOREIGN TRADE

Data on gold production are given in tables 1 and 2.

It has been estimated that of all the some 1,750,000,000 ounces of gold that has been won from the earth (since 1493), 60 percent is held by governments and central banks, 25 percent is held by corporations and individuals, and 15 percent has been dissipated or lost.

With the exception of gold in its natural state (that is, gold recovered from natural sources which has not been melted, smelted, or refined, or otherwise treated by heating or by a chemical or electrical process), it is unlawful in the United States to hold gold in unmanufactured form. However, gold may be purchased from the Mint through licensed dealers for legitimate uses in the arts, sciences, and industry without restriction. Data on the consumption of gold in the United States from 1943 to 1952, as supplied by the Bureau of the Mint, are given in table 3. No breakdown of the consumption by uses is available, but it is known that the arts provide the largest market by far.

During 1952 the United States imported gold ore or base bullion, refined bullion, and gold coins from 38 countries; during the same period it exported the same products to 38 countries; in both instances, refined bullion formed the principal value. Table 4 presents a summary of gold movements to and from the United States from 1948 to 1952, as supplied by the United States Department of Commerce.

TABLE 1.—*Mine production of recoverable gold in the United States, 1942–52, with production of maximum year, and cumulative production from earliest record to end of 1952, by States*

[Fine ounces]

	Maximum production [1]		Production by years											Total production from earliest record to end of 1952
	Year	Quantity	1942	1943	1944	1945	1946	1947	1948	1949	1950	1951	1952	
Western States and Alaska:														
Alaska	1906	1,066,030	487,621	99,583	49,296	68,117	226,781	279,988	248,395	229,416	289,272	239,637	240,557	27,610,693
Arizona	1937	332,694	253,651	171,810	112,162	77,223	79,024	95,860	109,487	108,993	118,313	116,093	112,355	11,529,260
California	1852	3,932,631	847,997	148,328	117,373	147,938	356,824	431,415	421,473	417,231	412,118	339,732	258,176	104,161,364
Colorado	1900	1,391,364	268,627	137,558	111,455	100,935	142,613	168,279	154,802	102,618	130,390	116,503	124,594	39,855,129
Idaho	1871	212,850	95,020	30,808	25,008	17,780	42,975	64,982	58,454	77,829	79,652	45,064	32,997	8,199,727
Montana	1865	870,750	146,892	59,586	50,021	44,597	70,507	90,124	73,091	52,724	51,764	30,502	24,161	17,374,487
Nevada	1910	913,265	295,112	144,442	119,056	92,265	90,680	89,063	111,532	130,399	178,447	121,036	117,203	26,264.681
New Mexico	1915	70.681	11,961	5,563	6,918	5,604	4,009	3,146	3,414	3,249	3,414	3,959	2,949	2,202,966
Oregon	1940	113,402	46,233	1,097	1,369	4,467	17,598	18,979	14,611	16,226	11,058	7,927	5,509	5,765,862
South Dakota	1939	618,536	522,098	106,444	11,621	55,948	312,247	407,194	377,850	464,650	567,996	458,101	482,534	23,804,626
Texas	1929	1,279	236	4	--------	--------	9	45	57	40	49	32	39	8,552
Utah	1950	457,551	391,544	390,470	344,223	279,979	178,533	421,662	368,422	314,058	457,551	432,216	435,507	13,072,394
Washington	1950	92,117	75,396	65,244	47,277	57,860	51,168	34,965	70,075	71,994	92,117	67,405	54,776	2,570,002
Wyoming	1869	7,498	23	--------	20	2	105	1,486	115	389	--------	9	1	80,041
Total	--------	--------	3,442,411	1,360,937	995,799	952,715	1,573,073	2,107,188	2,011,778	1,989,816	2,392,141	1,978,216	1,891,358	282,499.784
West Central States:														
Missouri	1900	33	--------	--------	--------	--------	--------	--------	--------	--------	--------	--------	--------	33
States east of the Mississippi:														
Alabama	1936	4,726	1	--------	--------	5	1	--------	--------	--------	--------	--------	--------	49,495
Georgia	1882	12,094	30	12	5	--------	21	76	19	18	--------	3	--------	870,663
Indiana	(2)	(2)	--------	--------	--------	--------	--------	--------	--------	--------	--------	--------	--------	(3)
Maryland	1937	1,040	--------	--------	--------	--------	--------	--------	--------	--------	20	1	--------	6,123
Michigan	1890	4,354	--------	--------	--------	--------	--------	--------	--------	--------	--------	--------	--------	33,297
North Carolina	1887	10,884	4,077	131	21	--------	--------	--------	--------	13	--------	--------	--------	1,164,601
Pennsylvania	1942	2,499	2,499	2,218	2,115	1,588	1,150	1,518	2,200	1,645	1,764	2,179	1,500	[4] 36,090
South Carolina	1941	15,508	7,824	147	--------	--------	--------	--------	--------	--------	--------	--------	--------	318,801
Tennessee	1930	696	159	303	222	148	95	303	156	171	160	108	241	22,104
Vermont	1946	165	--------	17	100	104	165	100	104	120	146	156	162	[5] 1,207
Virginia	1938	2,943	109	50	132	12	--------	--------	--------	--------	--------	--------	--------	167,558
Total	--------	--------	14,699	2,878	2,595	1,857	1,432	1,997	2,479	1,967	2,090	2,447	1,903	2,669,939
Grand total	--------	--------	3,457,110	1,363,815	998,394	954,572	1,574,505	2,109,185	2,014,257	1,991,783	2,394,231	1,980,663	1,893,261	285,169,756

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[1] For Central and Eastern State figures are peaks since 1880, except Pennsylvania and Vermont, for which the figures are peaks since 1905. For Alaska, Nevada, and Oregon figures are likewise peaks since 1880 only.

[2] Figure not available.

[3] Small, figure not available.

[4] 1908–52 only.

[5] 1905–52 only.

TABLE 2.—*World production of gold, 1948–52, by countries* [1] [2]

[Fine ounces]

(Compiled by Pauline Roberts and Berenice B. Mitchell)

Country [1]	1948	1949	1950	1951	1952
North America:					
United States (including Alaska) [3]	2, 025, 480	1, 921, 949	2, 288, 708	1, 894, 726	1, 927, 000
Canada	3, 529, 608	4, 123, 518	4, 441, 227	4, 392, 751	4, 419, 570
Central America and West Indies:					
Costa Rica [4]	1, 096	284	115	1	----------
Cuba	334	[4] 5, 692	[4] 6, 915	[4] 835	[4] 881
Dominican Republic [4]	29	993	475	411	332
Guatemala [4]	16	5	397	7	4
Honduras	13, 633	25, 832	36, 545	31, 216	31, 967
Nicaragua (exports)	218, 019	219, 139	229, 206	251, 160	254, 675
Panama	1, 000	[5] 9, 657	1, 118	2, 897	----------
Salvador (exports)	20, 778	27, 091	29, 053	27, 097	27, 682
Mexico	367, 612	405, 550	408, 122	394, 007	459, 370
Total	6, 177, 600	6, 739, 700	7, 441, 900	6, 995, 100	7, 121, 500
South America:					
Argentina (estimate)	8, 000	8, 000	8, 000	8, 000	8, 000
Bolivia	6, 687	33, 533	7, 716	[6] 3, 200	[5] 9, 034
Brazil (estimate)	156, 900	183, 500	195, 500	200, 000	180, 000
British Guiana	16, 518	19, 368	11, 800	13, 485	[6] 21, 000
Chile	165, 062	179, 144	190, 172	173, 646	176, 025
Colombia	335, 260	359, 474	379, 412	446, 314	422, 240
Ecuador	79, 207	99, 241	91, 946	12, 683	24, 267
French Guiana	13, 632	14, 757	12, 249	12, 056	8, 231
Peru	111, 162	113, 754	147, 967	158, 270	134, 865
Surinam	4, 177	3, 794	4, 546	6, 494	6, 134
Venezuela	49, 730	61, 378	34, 462	2, 861	4, 797
Total	946, 000	1, 076, 000	1, 084, 000	1, 037, 000	995, 000
Europe:					
Finland	11, 317	14, 587	9, 465	18, 500	20, 100
France	47, 519	55, 537	63, 015	67, 838	45, 011
Germany, West	(7)	1, 447	[6] 1, 500	1, 479	2, 025
Italy	18, 422	10, 385	10, 674	12, 089	14, 854
Portugal	11, 799	10, 385	19, 900	18, 350	19, 000
Rumania	90, 000	112, 528	(7)	(7)	(7)
Spain	11, 375	30, 318	13, 217	12, 777	8, 944
Sweden	71, 889	80, 280	78, 866	70, 957	(7)
U. S. S. R. (estimate) [8]	7, 000, 000	7, 000, 000	8, 000, 000	9, 500, 000	9, 500, 000
Yugoslavia	26, 331	34, 594	42, 760	21, 380	36, 266
Total (estimate)	7, 300, 000	7, 400, 000	8, 400, 000	9, 800, 000	9, 800, 000
Asia:					
Burma	230	158	150	131	(7)
China	88, 200	[6] 60, 000	108, 000	[6] 100, 000	[6] 100, 000
India	180, 430	164, 203	196, 848	226, 475	243, 629
Indonesia (estimate)	32, 000	35, 000	42, 000	(7)	(7)
Japan	69, 060	84, 492	135, 033	177, 472	201, 392
Korea:					
Korea, Republic of	3, 466	3, 419	5, 144	1, 000	15, 657
North Korea (estimate)	300, 000	300, 000	200, 000	(7)	(7)
Malaya	10, 212	13, 617	18, 436	17, 018	19, 806
Philippines	209, 225	287, 844	333, 991	393, 602	469, 408
Sarawak	599	1, 523	1, 440	931	843
Saudi Arabia	74, 000	66, 835	66, 202	73, 104	69, 394
Taiwan (Formosa)	17, 668	16, 607	18, 232	30, 500	33, 178
U. S. S. R.	(8)	(8)	(8)	(8)	(8)
Total (estimate)	990, 000	1, 040, 000	1, 130, 000	1, 280, 000	1, 420, 000

For footnotes, see end of table.

TABLE 2.—*World production of gold, 1948–52, by countries* [1] [2]—Continued

[Fine ounces]

Country [1]	1948	1949	1950	1951	1952
Africa:					
Angola	443	319	201	61	40
Bechuanaland	1, 507	256	261	493	1, 245
Belgian Congo [9]	299, 774	333, 853	339, 415	352, 308	368, 769
Egypt	3, 853	7, 045	10, 724	16, 469	17, 059
Eritrea	2, 242	2, 243	1, 042	675	(7)
Ethiopia	41, 595	45, 102	43, 524	32, 937	27, 291
French Cameroon	10, 706	8, 938	7, 170	5, 422	2, 604
French Equatorial Africa	63, 713	57, 273	54, 996	52, 849	51, 655
French Morocco	804	643	119	2, 069	4, 051
French West Africa [10]	100, 000	47, 000	96, 000	5, 700	1, 500
Gold Coast	672, 388	676, 934	689, 441	698, 676	715, 036
Kenya	23, 429	20, 072	22, 945	19, 765	10, 210
Liberia	13, 797	14, 656	11, 025	[5] [11] 9, 806	[5] [11] 949
Madagascar	2, 095	1, 663	1, 935	1, 951	1, 768
Mozambique	4, 734	2, 468	997	861	831
Nigeria	2, 899	2, 515	2, 238	1, 566	[6] 1, 350
Northern Rhodesia	[12] 1, 180	[12] 1, 186	[12] 1, 432	101	20
Sierra Leone	2, 405	2, 330	3, 484	3, 292	[6] 2, 700
Southern Rhodesia	514, 440	528, 180	511, 163	486, 907	496, 731
South-West Africa	455	32	32	(7)	(7)
Sudan	3, 579	4, 114	3, 503	1, 495	1, 545
Swaziland	3, 110	2, 841	1, 794	322	1
Tanganyika (exports)	57, 557	68, 989	65, 127	65, 224	64, 693
Uganda (exports)	1, 158	650	509	222	181
Union of South Africa	11, 584, 849	11, 705, 048	11, 663, 713	11, 516, 450	11, 818, 681
Total	13, 415, 000	13, 535, 000	13, 535, 000	13, 275, 000	13, 590, 000
Oceania:					
Australia:					
Commonwealth	885, 507	889, 057	867, 837	895, 536	980, 435
New Guinea	86, 556	93, 045	80, 099	94, 085	122, 431
Papua	163	450	788	248	149
Fiji	93, 059	104, 036	103, 421	93, 635	[6] 95, 000
New Zealand	93, 903	84, 874	76, 527	75, 115	59, 373
Total	1, 159, 188	1, 171, 462	1, 128, 672	1, 158, 619	1, 257, 000
World total (estimate)	30, 000, 000	31, 000, 000	32, 700, 000	33, 500, 000	34, 200, 000

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[1] Figures used derived in part from American Bureau of Metal Statistics. For some countries accurate figures are not possible to obtain owing to clandestine trade in gold (as for example, French West Africa). Data not available for Austria, Bulgaria, Czechoslovakia, Hungary, and Thailand; estimates included in the total. In addition, production in Cyprus and Indonesia was negligible.

[2] This table incorporates a number of revisions of data published in previous gold chapters.

[3] Refinery production. Excludes production of the Philippines.

[4] Imports into United States.

[5] Exports.

[6] Estimate.

[7] Data not available; estimate included in total.

[8] Output from U. S. S. R. in Asia included with U. S. S. R in Europe.

[9] Includes Ruanda-Urundi.

[10] Estimate based on reported production.

[11] Year ended September 30 of year stated.

[12] Included is yield from Nkana mine refinery slimes accumulated during the war: 999 ounces in 1948; 972 in 1949; and 1,296 in 1950.

TABLE 3.—*Net industrial*[1] *consumption of gold in the United States 1943–47 (average) and 1948–52*

[U. S. Bureau of the Mint]

Year	Gold (dollars)		
	Issued for industrial use	Returned from industrial use	Net industrial consumption
1943–47 (average)	131, 518, 846	32, 484, 252	99, 034, 594
1948	90, 128, 764	45, 142, 764	44, 986, 000
1949	148, 975, 571	40, 133, 100	108, 842, 471
1950	134, 587, 773	36, 742, 020	97, 845, 753
1951	105, 012, 094	35, 535, 115	69, 476, 979
1952	127, 189, 489	30, 838, 949	96, 350, 540

[1] Including the arts.

TABLE 4.—*Value of gold imported into and and exported from the United States 1948–52*

[U. S. Department of Commerce]

Year	Gold (dollars)		
	Imports	Exports	Excess of imports over exports[1]
1948	1, 981, 175, 178	300, 771, 144	1, 680, 404, 034
1949	771, 390, 261	84, 935, 678	686, 454, 583
1950	162, 748, 661	534, 035, 794	−371, 287, 133
1951	81, 258, 502	630, 381, 566	−549, 123, 064
1952	740, 254, 160	55, 921, 206	684, 332, 954

[1] Excess of exports over imports indicated by minus sign.

SELF-SUFFICIENCY AND STRATEGIC CONSIDERATIONS

Although the gold output of the United States has declined greatly since 1940, the gold reserves of the country are still substantial, and domestic production probably will be at a level around the current rate for perhaps 2 decades. This gold, however, is far overshadowed by the billions of dollars of gold in the country's gold stocks that exceeds the monetary reserve required by statutes. Moreover, as long as the value of exports exceeds imports, it is normal for gold to flow to rather than from the United States, and it seems unlikely that the movement of gold will be checked or reversed as long as this condition prevails.

Gold does not have many industrial applications that are indispensable from a military standpoint. However, it may be said that, in a world war, the country's gold reserve might have great strategic importance in maintaining its solvency as well as that of certain of its allies. Transfer of credits could be effected by "earmarking" without actual shipment of gold and exposure to wartime transportation risks.

Unlike activities for most mineral commodities, gold mining usually is discouraged, rather than encouraged, by economic forces and control agencies during wartime, particularly in the United States. It was demonstrated in World War II, however, that in the United States regulations to divert manpower to base-metal mines by closing gold mines were generally ineffective.

PRICES AND COSTS

The value of gold in the United States was fixed at $35 per fine troy ounce by Presidential proclamation on January 31, 1934. Since this was the price available for sales of gold by all comers to the United States Treasury (the only market strong enough to absorb the world's gold production), it also determined the official world price. Because of a demand in the late 1940's for gold for hoarding in various parts of the world owing to political unrest and lack of confidence in paper currencies, "free markets" for gold developed in Europe and the East, in which gold was sold at a price of $40 to $45 or higher per ounce, with the supply furnished largely by subterfuge. Pressure by gold producers to be allowed to sell their gold in the free market, however, brought concessions in policy in late 1951 by the International Monetary Fund (an international organization for control, exchange, etc., of gold), and new regulations were promulgated by some gold-producing countries to, in effect, permit such practice. This raised the supply of gold available to the free markets to an estimated 40 percent of the current world production. Regulations of the United States Treasury were not relaxed. The effect of the larger potential supply of gold for the free market was a lowering in the price for gold to around $37 per ounce at the end of 1952. In the United States "natural" gold appears to command a price equivalent to $38 to $40 per fine troy ounce, with sales amounting to less than 1 percent of the total output of placer gold. Legislation by the Congress to raise the price of gold in the United States is being sought at this writing by certain students of monetary theory as well as the gold producers, but some opinion holds that a higher gold price would be inflationary and against the public interest (*2*, *13*, *15*).

At the former price of $20.67 an ounce, world production of gold had fallen steadily for many years because of depletion and rising costs. Under stimulation of the $35 price, straight-gold mining again flourished, and many countries set new yearly production records through the late 1930's and early 1940's. Gold mining declined in most countries during the war for want of labor and supplies and due to higher costs; and, although the end of the war brought relief in some respects, in general costs have risen steadily during postwar years to the extent that some gold-mining operations throughout the world have become or are becoming unprofitable. Devaluation of the British pound to $2.80 in September 1949 restored many mines in the sterling area to financial soundness, but most benefits of this devaluation have been wiped out by new increases in costs. Marginal gold mines in Canada have been aided for several years by subsidies from the Canadian Government based on a formula, and gold mining in Australia has been assisted by tax concessions. Most foreign gold producers are permitted to benefit by the sale of some of their output at premium prices in the free market. Despite the handicaps, the majority of the large straight-gold producers of the world are still operating at a profit and paying substantial dividends. In the United States a number of small gold mines have had to close in recent years, and many others are barely able to keep going; however, current statements of most of the larger straight-gold-mining companies indicate profits from their operations, but on a declining scale. Some companies have been forced to mine their higher grade reserves and sacrifice those of lower grade. Moreover, it appears certain that, if the trend to still higher costs continues, the difficulties of straight-gold mining will increase. Base-metal mines that yield byproduct gold normally are less subject to problems arising from the fixed price of gold than are straight gold mines.

TAXES, TARIFF, ETC.

Gold-producing mines are subject to the taxes that apply to all business enterprises; they include local property taxes, income taxes, and excess-profit taxes. The question of proper and adequate tax credit because of depletion is frequently raised. Dividend payments to stockholders are subject to income taxes, and special levies are made by some countries against dividend payments to nonresident stockholders. Various countries impose a production tax or export tax on gold that may add considerably to mining costs. In the United States promotions or reorganizations of mining ventures, including those concerned with gold, are subject to the scrutiny and approval of the Security and Exchange Commission to protect investors but only to the same extent as those of other types of ventures.

No duties are imposed by the United States on imports of gold in ores or bullion, but gold manufactured goods are subject to duties ranging from 30 to 65 percent ad valorem. The volume of imports of such gold merchandise is relatively small, however.

RESEARCH

By and large, the processes of recovering gold from ores, particularly by cyanidation, have reached a state of high efficiency due to research by laboratories, manufacturers, and mining companies on chemical reagents, design of machinery, and procedures, supplemented by a world of operating experience. Research is still necessary to adapt the basic processes to the peculiarities of some ores or to meet special conditions, and such research is done by the mining companies themselves and by ore-testing laboratories.

OUTLOOK

CURRENT TRENDS

The immediate outlook is for the world output of gold to continue at about the 1952 level. The short-term outlook is for somewhat greater world production, as growing output from the new gold districts in the Orange Free State and the Transvaal in the Union of South Africa is added to the yield of older fields. The long-term outlook depends on greater returns to the gold-mining industry for its product to offset rising costs, whether by sales of more gold at premium prices or a higher world price for gold. In the United States straight-gold-mining operations, both lode and placer, appear headed for progressively more difficult times unless relief, in some form, from rising costs can be brought about. The output of byproduct gold will vary with the fortunes of the metals with which it is associated and will be only mildly affected by the factors that determine the output of straight-gold mines.

NEW USES AND NEW SOURCES IN PROSPECT

Variations in the established uses for gold probably will be developed from time to time, but no basic new uses are in sight. No sources of gold additional to those already known are definitely in prospect. However, large, known, low-grade gold deposits and gold districts in remote areas in some countries may be potential new sources. It is reasonable to believe, also, that important gold deposits now unknown will be found in parts of the world only partly explored, as well as in areas covered by detritus, lava flows, etc., where ordinary methods of prospecting are ineffective.

There is a growing sentiment among statesmen that an expansion of international trade and a reduction in the trade barriers and in monetary restrictions would be an enormous step toward peace. The possibility of gold playing a major role in such international stabilization is important.

FORECAST OF SUPPLIES

Continuance of international tension and maintenance of high level industrial activity for a period of several years would lower the gold output in most countries, particularly in the United States; the flow of gold from other countries to the United States would tend to accelerate. This situation would cause concern to some experts on monetary matters as a threat to the world-credit structure, but others would not find it particularly alarming. The ultimate results probably are tied to political and fiscal problems and developments that cannot be envisaged now.

PROBLEMS

The chief problem of the gold-mining industry, particularly of the straight-gold mines, is the price of gold in relation to the cost of supplies and labor. The price of $35 per ounce as fixed in 1934 brought general prosperity to gold mining that continued through the 1930's. Since 1940 there has been a world war, an era of industrial prosperity, a large increase in population in most countries, the cold war, the war in Korea, and finally the rearmament program of the Western World, all with inflationary effects. Average wages have more than doubled in the intervening years, and the cost of supplies has gone up in proportion. The result is that, despite economies resulting from mechanization and more efficient new mining machinery, many gold mines have been unable to make ends meet. Properties in some foreign countries have had aid in the form of subsidies, tax concessions, or access to the free market for sale of some of their output; but none of these expedients has been available to gold producers in the United States. About all that can be said for the future is that a time of peace would tend to ameliorate conditions harmful to gold mining, and war with its attendant inflation would tend to aggravate them.

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GRAPHITE

By

Frank D. Lamb [1] and Donald R. Irving [2]

GRAPHITE derives its name from the Greek verb "to write," because the mineral is used to manufacture "lead" pencils. This is probably its most familiar application, although graphite is widely used as a dry lubricant, a refractory, and an electrical conductor. Utilization of graphite remains in the craft stage and offers excellent opportunities to apply scientific methods and new technologies.

Summary

Production of natural graphite in the United States has never been large, despite many attempts to mine domestic ores. Domestic deposits are low grade, and domestic producers must compete with much richer foreign deposits.

Requirements for graphite in the United States average about 45,000 tons annually. Of this quantity about 80 percent is the so-called amorphous variety, largely imported from Mexico, and only 15 to 20 percent is the crystalline flake variety, largely imported from Madagascar.

One critical variety of graphite is that used for manufacturing crucibles. There is only negligible production of this type of graphite in the Western Hemisphere, and domestic crucible manufacturers depend upon imports from Madagascar. In normal times the United States has imported 3,000 to 4,000 short tons of crucible-grade graphite annually.

Another critical variety is Ceylon graphite, which is used in carbon brushes for electrical equipment, oilless bearings, powder metallurgy, packings, and pencils. No other variety of graphite can be substituted completely for Ceylon graphite in carbon brushes and some other strategic uses.

Other major uses for natural graphite include foundry facings, recarburizing steel, lubricants, brake linings, and paints. Manufactured graphite is used mostly for electrodes and as a moderator in atomic piles.

Domestic deposits occur in 25 States, and production has been reported from 17. Major foreign producing countries include Mexico, Austria, Norway, Korea, and Germany, in addition to Madagascar and Ceylon.

Problems of the graphite industry include marketing, lack of adequate knowledge concerning the degree of interchangeability of various grades of graphite, and the need for better milling methods to permit the recovery of coarser flake and higher grade graphite from crystalline ores.

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[1] Commodity-industry analyst, Bureau of Mines.

[2] Assistant chief, Branch of Ceramic and Fertilizer Materials, Bureau of Mines.

BACKGROUND

Graphite is a very soft, black, greasy-feeling mineral that occurs in disseminated flakes or in scaly, granular, compact, or earthy masses. It is elemental carbon, usually mixed with such impurities as iron oxide, clay, mica, and other minerals. All graphite is crystalline, but in industrial nomenclature two major classifications are used to distinguish between the coarsely crystalline flake variety and the fine, granular, cryptocrystalline variety. The coarsely crystalline variety is referred to as "crystalline graphite" or "flake graphite" and must contain graphite crystals large enough to be visible to the unaided eye. The cryptocrystalline variety, in which the crystals cannot be distinguished by the eye alone, is referred to as "amorphous graphite." Natural mixtures of crystalline and amorphous graphite occur in all proportions. The terms "manufactured," "artificial," and "synthetic" graphite all are used to describe a form of industrial carbon made from anthracite culm or petroleum coke. Manufactured (artificial) graphite is actually pure crystalline graphite; the only artificial attribute is the method of production.

The quantity of graphite imported for consumption in the United States plus domestic production has ranged from 31,000 to 62,000 short tons a year during the period from 1942 to 1953, inclusive. The figure includes purchases for the National Stockpile. Most of the graphite consumed is of the amorphous variety, largely imported from Mexico, and only about 15 to 20 percent of the total graphite consumed is of the crystalline flake variety, largely imported from Madagascar. In spite of the relatively small consumption, the importance of graphite to United States industry cannot be overemphasized. Virtually all uses of graphite are very important to industrial production; and some of the more critical uses, which require special grades of graphite, would be difficult to dispense with in an emergency. Fortunately, during the past 20 or 30 years the nature of consumption of graphite has altered so that different kinds of graphite are more nearly interchangeable in use. From the standpoint of national security the proportion of graphite from distant sources that can be considered absolutely essential to military, or even civilian, use is relatively small compared with what it was 25 years ago. However, the major part of the world's high-grade graphite is produced in Madagascar, Germany, and Ceylon, and the availability of these grades of graphite during wars invariably has been restricted or curtailed.

USES

CRUCIBLES

An important use for graphite is in the manufacture of crucibles for melting nonferrous metals, mainly brass and aluminum, where it is employed because of its inertness and refractory properties. Graphite of various kinds and from many countries has been used for making crucibles. Agricola (*1*)[3] mentions the use of crucibles made from Passau (Bavaria) graphite, but by the middle of the 17th century European crucible makers were utilizing mostly Ceylon graphite.

The manufacture of graphite crucibles in the United States was begun in 1827, using graphite from Nelson, N. H., where the Joseph Dixon Crucible Co. was established. In 1829 Dixon imported the first shipment of Ceylon graphite (*7*); this material soon became the standard for the American crucible industry, although some domestic graphite was consumed. Most of the formulas for crucible manufacturing before 1915 were based on the use of Ceylon crystalline graphite and German Klingenberg clay; but World War I brought about a sharp curtailment of shipping from Ceylon and Germany, and the United States was cut off entirely from these supplies. Domestic graphite and clays had to be substituted, with the result that the crucibles were greatly inferior to those previously used. Such an abrupt change in the ingredients of the crucible formulas was felt, particularly in the service life of brass-melting crucibles. Graphite crucibles used for nonferrous melting are subject to great thermal shock because they are used over a period of days and subjected to changes of temperature from that of the foundry room to as high as 2,700° or 2,800° F. The service life of crucibles made with Ceylon graphite had been 28 to 30 heats, and at best only 5 or 6 heats could be obtained from domestic graphite crucibles during that period. Some improvement in the service life of crucibles was made during World War I, and heats per crucible were increased to about 12.

After World War I Madagascar flake graphite which had been used exclusively in France during the later years of the war, became available at very low cost. Formulas were established that gave greater service life than had previously been obtained with Ceylon crystalline graphite, and Madagascar flake then became the standard for the industry. Additional refinements in formulas and tech-

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[3] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

nique have been made, and today crucibles have a service life of 30 to 150 heats, with a high of 200 heats in some aluminum-alloy tilting furnaces (*3*).

When France fell to Germany in 1941 and Japan began its expansion southward, it became obvious that imports of Madagascar flake graphite might be cut off; hence, during the early years of World War II, the lack of crucible-grade graphite threatened to endanger the United States war program.

However, the need for coarse flake graphite is decreasing because crucible manufacturers are making carbon-bonded silicon carbide-graphite crucibles in which the percentage of graphite is substantially less than that used in clay-graphite crucibles. One manufacturer states that silicon carbide crucibles require 18 to 25 percent graphite compared with 45 to 50 percent for clay-bonded crucibles. Furthermore, much finer flake can be utilized. Domestic flake is said to be satisfactory from a performance standpoint and probably would be used if it could be obtained at a price comparable with that of Madagascar fine flake. It is estimated that approximately 75 percent of the crucibles manufactured in the United States are carbon-bonded silicon carbide crucibles.

Other refractory articles made from the same type of graphite used for crucibles and employed in conjunction with crucibles or similar furnacing operations are crucible covers, pouring nozzles, crucible rests and stools, funnel or extension tops, skimmers, phosphorizers, ladle stoppers and stopper sleeves, pyrometer sleeves, furnace bricks and doors, annealing boxes, retorts, and case-hardening containers.

CARBON BRUSHES

Ceylon graphite has been a standard ingredient in the higher qualities of carbon brushes for electric motors and other electrical equipment, such as those used in military aircraft, which operate largely on 32-volt systems. Because of the wide variation in design and operating characteristics of motors, generators, dynamotors, amplidynes, and other equipment, it is impossible to eliminate the risk element of failure by bench testing. Hence, for this one purpose, Ceylon graphite is indispensable. United States practice is to use a special grade of amorphous lump graphite which contains 97 to 98 percent graphitic carbon. Substitutes are said to be unsatisfactory because of abrasive impurities and contaminants, such as copper and iron, that are present in manufactured (artificial) graphite and other natural graphites. The quantity required does not exceed 1,000 tons a year.

FOUNDRY FACINGS

By far the largest single use for graphite is for foundry facings—finely pulverized materials used in foundries to give the surface of molds a smooth finish, so that castings may be removed easily after cooling. Graphite is used for this purpose more extensively than any other material. The usual practice is to mix the graphite with sand, clay, talc, or mica; and, except for the universal requirement of extreme fineness, the preparation of facings follows no generally accepted standard. There is no particular need for graphite high in graphitic carbon, because the graphite is diluted with sand and clay, which are commonly present in many low-grade graphites. Consequently, very little refining is necessary in preparing graphite for this use. All types of finely pulverized amorphous and crystalline graphite are used for foundry facings. High-carbon graphite is required for facing some castings where a superior finish is desired, such as some brass castings. This use requires only a small quantity of material.

STEELMAKING

Large quantities of amorphous graphite are used to recarburize steel, although carbon in other forms, such as coal or coke, is used more widely. For this use there is no need for high-carbon graphite.

LUBRICANTS

Graphite of all types is used extensively in manufacturing various forms of lubricants. It is valuable for this purpose because it adheres readily to metal surfaces and fills the pores, giving a veneer that reduces the bearing coefficient of friction virtually to that of graphite itself. The veneer is also protective, being resistant to most chemical and corrosive reactions. Flake graphite is preferable because it readily forms a scaly layer on the metal, essential for successful solid lubrication. Extremely fine graphite tends to flow excessively with the oil supplied to the bearing. Graphite also is used for dry lubrication where oil and grease might be detrimental; where liquids cannot be used, as for instance, on the bridges of pianos, organs, and textile machines; and where lubricants are subjected to wide temperature changes.

Lubricating-grade graphite must be free from abrasive impurities. It may be used alone or mixed with grease, oil, or water. "Oildag," a suspension of finely divided (sometimes termed "colloidal") graphite in oil, and "aquadag," a similar suspension in water, are made from manufactured (artificial) graphite.

Oilless bearings are made by impregnating wood, copper, bronze, or babbitt bearings with a special lubricating mixture containing graphite.

PENCILS

Perhaps the earliest and best known use for graphite was in pencils. The "lead" in pencils consists of a mixture of graphite, clay, and other materials molded and baked for several hours at 1,500° to 2,000° F. Depending upon the relative proportions of graphite and other materials, the hardness of the "lead" is controlled over a wide range. The better, more expensive pencils require finely ground, relatively high-carbon graphite, free from impurities. Either flake or amorphous graphite may be employed, and in the United States approximately equal quantities of the higher grades of Mexican amorphous and high-carbon Ceylon amorphous graphite are used.

PAINTS

Natural graphite of any degree of purity may be used in manufacturing paints. Such paints have value in protecting metal surfaces, such as bridges, railroad cars, smokestacks, boiler fronts, tanks, pipes, and metal roofs. Graphite containing substantial quantities of silica is mixed with iron oxide or zinc compounds, a suitable filler, and an oil vehicle to manufacture most of these noncorrosive paints.

MISCELLANEOUS

Other important uses for graphite are for graphite electrodes, roofing compounds, fillers for dry batteries, various forms of steam and pump packing, and many other minor commodities.

Graphite electrodes, while important, are not made from natural graphite. Petroleum coke, anthracite, and other forms of carbon are bonded with a pitch binder and graphitized in an electric furnace. Substantial quantities of graphite are used in atomic piles as a moderator, or as a slowing-down agent for neutrons, but again only manufactured (artificial) graphite is used.

GRADES AND SPECIFICATIONS

The value of graphite depends upon whether it is of the amorphous or crystalline flake variety, upon its graphitic carbon content, and upon its physical nature. The nomenclature used by the industry to designate the many grades of graphite available is complex and confusing because of the number of products and grades shipped from foreign mines.

Ceylon graphite is divided into four main grades: Amorphous Lump, containing about 90 to 98 percent graphitic carbon; Crystalline Lump and Chip, about 85 to 90 percent graphitic carbon; Amorphous Dust, about 55 to 85 percent graphitic carbon; and Amorphous Flying Dust, about 55 to 90 percent graphitic carbon. Each grade may be further subdivided into best, medium, and poor classes or simply classified as Nos. 1 and 2 Lump, Nos. 1 and 2 Chip, etc.

Madagascar graphite is divided into two main grades, flake and fines. Both grades are of the crystalline-flake variety, and the distinction is simply in particle size. They are further distinguished by the industry, mainly to indicate the quality or graphitic carbon content. Usually it is the physical character of the product that is of the utmost importance; the value of flake graphite depends on its structure and the size and toughness of particles almost as much as on its carbon content. Crucible-grade flake ranges in size from 8- to 60-mesh and contains at least 85 percent graphitic carbon.

Mexican amorphous graphite is divided into high- and low-grade classifications, depending upon arbitrary graphitic carbon contents. High-grade Mexican graphite contains about 85 percent graphitic carbon, and very little low-grade ore is exported to the United States containing less than 80 percent graphitic carbon.

National Stockpile specifications cover the physical and chemical requirements for the three grades of graphite designated strategic and critical, as follows:

P–21—97/98 percent C amorphous lump.
P–22a—crucible-grade crystalline flake.
P–22b—lubricating and packing grade.

GEOGRAPHIC DISTRIBUTION

DOMESTIC DEPOSITS

Graphite has been reported in 25 States in the United States, and production in commercial quantities has come from 17 (*8*). Flake-graphite deposits have been mined in Alabama, California, Massachusetts, New Jersey, New York, Pennsylvania, and Texas, and amorphous graphite has been mined in Colorado, Georgia, Michigan, Nevada, New Mexico, North Carolina, Rhode Island, Utah, and Wisconsin. Lump graphite similar to that produced in Ceylon has been mined near Dillon, Mont., but never in large quantity.

Only three domestic deposits were being worked in 1953: Graphite Mines, Inc., at Cranston, R. I., where amorphous graphite is produced; Alabama Flake Graphite Co., Ashland, Ala., producing crystalline flake graphite, mostly in fine sizes; and Southwestern Graphite Co., Burnet, Tex., producing fine crystalline flake graphite. The Alabama and Texas mines were both closed during 1954. During 1953 and early 1954 experimental mining and milling at the Government-owned Benjamin Franklin Graphite mine near Chester Springs, Chester County, Pa., resulted in production of a small tonnage of flake graphite. About 25 percent of the product was plus-50-mesh, suitable for crucible flake.

Alabama has been the most prolific graphite producer; at one time during World War I, over 40 mines were operating in Clay, Coosa, and Chilton Counties. Graphite has been produced in Pennsylvania intermittently since about the middle of the 18th century; the deposits in Pickering Creek Valley, Chester County, are the most important, but others in Berks and Lehigh Counties have been mined. Deposits in the Adirondack Mountains in New York produced a limited quantity of flake graphite some years ago.

FOREIGN DEPOSITS

Graphite is found in virtually all countries of the world. Most of the world production in 1953 came from Ceylon, Madagascar, Mexico, Austria, Norway, Korea, and Germany. Aside from Ceylon and Madagascar, the bulk of the graphite produced is low-priced amorphous material, and the value of the graphite produced in Ceylon and Madagascar is probably at least half of the world total. The world production of natural graphite, by countries, in metric tons, is given in table 1.

Ceylon.—The commercially important deposits in Ceylon lie in the Western and Southern Provinces. They are all veinlike in character and were deposited along fracture planes in the rocks. The width varies from less than one-eighth inch to several feet. In small veins the graphite usually forms an aggregate of platy needles at right angles to the vein wall, while in larger veins the graphite is coarser and the needlelike structure may be developed only a short distance from the walls.

Germany.—An important graphite area in Germany is east of Passau near the junction of Germany, Austria, and Czechoslovakia. The only operating mine is at Kropfmuehl about 10 miles east of Passau. Crystalline graphite occurs in seams and lenses and disseminated in gneissic and schistose rocks. Individual graphite seams and lenses are mined in thicknesses up to several scores of feet and for distances of several hundred feet. Graphite makes up 10 to 30 percent of the minable lenses and averages 20 to 25 percent after hand sorting (*9*).

Kenya.—Production of flake graphite was begun in 1954 from a deposit in Thika. One crucible manufacturer in the United States reported that tests indicated the Kenya flake graphite is comparable to Madagascar flake and in some respects better.

Korea.—Amorphous and some crystalline-flake graphite is mined in Korea; most of the production goes to Japan. The amorphous graphite occurs in beds and irregular lenses found mainly in the central and eastern parts of the country south of the 38° parallel. The crystalline graphite occurs in the northwestern part of the country just south of Manchuria in North Korea. Korean graphite is generally low grade and is used for foundry facings, paint, and other noncritical uses. The crystalline variety never has been been widely accepted.

Madagascar.—Graphite deposits are widespread on the island of Madagsacar. The principal producing mines are in the lowlands in a broad area centering about 65 miles southwest of Tamatave and those of secondary importance in the uplands in a rough line extending from Tananarive about 120 miles to the south. Flake graphite occurs disseminated throughout certain gneisses and schists ranging in thickness from almost nothing to 40 or 50 feet. The average grade of deposits being worked is 5 to 10 percent graphite.

Mexico.—Amorphous graphite deposits, described as perhaps the largest in the world, occur south of Hermosillo, Sonora, Mex. The graphite occurs in beds up to 24 feet thick (which were originally coal seams), in metamorphosed sandstone. Selected samples contain as much as 95 percent graphitic carbon. Typical analyses show 80 to 85 percent graphitic carbon, 7.6 percent silica, 0.65 percent iron oxide, and 5.0 percent alumina.

Norway.—The only operating mine is A/S Skaland Grafitverk on Senja Island, north Norway. In 1952 Skaland completed an expansion program that increased the capacity of its flotation plant to 8,000 metric tons a year. Ore reserves are estimated to be 1 million tons of 25-percent-carbon, crystalline, battery-grade graphite. Additional veins are known but not considered in computing the reserve estimate.

Russia.—Little is known about the Russian graphite deposits except that they are extensive. Graphite that was used for pencils—and to a small exttet for crucibles—was mined at the Aliber mnne near Irkutsk in eastern Siberia. Other important deposits are those of Yenisei River in eastern Siberia and those in the northern part of the Caucasus Province. In the extreme eastern cape of Siberia are graphite deposits similar to those found on the Seward Peninsula in Alaska. The only deposit in European Russia known to be important is that of the Dzimarsk district in the Caucasus. Other occurrences are the Pudosch district, near Onegasec; the Jelantschik district, southern Ural Mountains; Zytomierc district near Mecherzynce; Volhynia near Mariupol; and the Jekaterinslav district, Ukraine.

TABLE 1.—*World production of natural graphite, by countries,*[1] *1915–39 (5-year averages) and 1940–53 (annual)*

[Metric tons]

	1915–19	1920–24	1925–29	1930–34	1935–39	1940	1941	1942	1943	1944
North America:										
Canada	2,481	1,172	1,756	790	1,639	1,381	905	1,081	1,726	1,435
Mexico	3,059	4,340	5,699	3,521	9,846	12,327	16,928	20,811	20,677	12,977
United States	8,493	4,732	4,974	(2)	(2)	(2)	2,493	6,459	9,016	4,906
South America:										
Argentina	---	---	---	---	16	100	135	244	237	455
Brazil	13	10	4	4	5	(3)	60	72	19	199
Europe:										
Austria	19,657	11,567	19,083	14,653	18,645	22,991	24,264	26,203	25,336	22,487
Czechoslovakia	26,841	13,751	29,276	4,187	4,229	10,725	8,694	13,126	21,252	21,459
Germany	31,308	21,696	17,548	21,333	24,496	29,891	29,771	33,316	34,960	36,357
Italy	9,151	5,722	8,487	3,997	5,352	4,996	4,136	5,483	6,309	3,008
Norway	---	2	---	1,206	3,205	2,646	3,588	2,933	3,178	3,784
Spain	1,184	1,923	580	---	4	353	23	---	57	91
Sweden	101	1	---	---	66	153	205	174	171	---
U. S. S. R. (Russia)	(5)	(5)	3,992	32,333	(5)	(5)	(5)	(5)	(5)	(5)
Yugoslavia	---	---	---	---	---	---	---	---	---	---
Asia:										
Ceylon (exports)	21,042	9,088	13,618	8,677	16,319	24,414	27,670	28,180	20,397	12,461
China	---	---	---	---	---	---	16,900	16,800	10,000	10,000
India	349	29	8	71	585	827	827	1,072	1,152	942
Indochina	6,160	---	289	---	2	(3)	2	14	25	30
Japan	1,380	778	578	572	---	2,088	3,393	2,866	7,791	10,380
Korea	7,679	15,034	18,484	23,721	50,205	12,591	68,640	96,054	96,471	103,306
Malaya	---	---	---	---	---	---	---	[4] 163	[4] 163	[4] 163
Taiwan (Formosa)	---	---	---	---	---	---	---	---	---	---
Africa:										
Egypt	---	---	---	---	---	---	(3)	(3)	---	260
French Morocco	---	---	21	108	417	529	571	1,067	265	213
Kenya	---	---	---	---	---	---	---	---	---	10
Madagascar	16,776	9,929	14,141	6,111	11,183	15,311	13,018	9,562	12,949	14,478
Mozambique	---	---	---	---	---	---	---	---	428	---
Portuguese East Africa	---	---	---	---	---	---	---	165	428	---
South-West Africa	---	---	---	---	---	71	188	181	1,758	1,633
Southern Rhodesia	---	---	---	---	---	---	---	---	---	5
Spanish Morocco	---	---	---	---	36	352	414	251	79	42
Union of South Africa	64	50	51	53	61	78	74	661	442	324
Australia	100	18	12	33	17	110	439	305	580	447
Total (estimate)	156,000	99,000	139,000	121,000	146,000	142,000	223,000	267,000	273,000	262,000

	1945	1946	1947	1948	1949	1950	1951	1952	1953
North America:									
Canada	1,733	1,792	2,175	2,303	1,948	3,253	1,423	1,851	3,153
Mexico	23,634	21,949	27,984	35,261	23,812	24,626	32,286	24,152	30,330
United States	4,434	5,058	3,980	9,026	5,536	4,628	6,473	5,086	5,698
South America:									
Argentina	333	250	(3)	(3)	(3)	(3)	(3)	(3)	(3)
Brazil	131	92	129	8,124	556	471	610	(3)	---
Europe:									
Austria	3,483	252	3,845	11,300	14,400	14,685	18,227	19,711	14,683
Czechoslovakia	10,973	5,108	7,000	15,000	(3)	(3)	(3)	(3)	(3)
Germany	(3)	3,800	4,930	5,757	5,097	6,563	10,300	8,963	7,459
Italy	2,276	2,593	3,845	7,251	4,639	4,521	4,514	3,960	4,910
Norway	1,115	661	2,481	1,083	2,257	2,457	3,453	4,120	[4] 3,000
Spain	128	320	309	241	256	310	274	783	[4] 250
Sweden	802	---	---	64	109	---	---	---	---
U. S. S. R. (Russia)	(5)	(5)	(5)	(5)	(3)	(3)	(3)	(3)	(3)
Yugoslavia	---	---	---	---	---	---	---	687	(3)
Asia:									
Ceylon (exports)	7,946	8,212	9,150	14,221	12,437	13,030	12,824	7,782	7,334
China	10,000	---	---	---	---	---	(3)	(3)	(3)
India	1,316	1,653	1,255	1,675	988	1,611	1,603	2,182	(3)
Indochina	---	---	---	---	---	---	---	---	---
Japan	12,444	7,416	10,584	9,132	5,664	4,008	4,872	4,650	[4] 3,900
Korea	32,407	6,204	[4] 10,000	[6] 15,958	[6] 45,219	[6] 19,049	[6] 21,578	[6] 15,066	[6] 20,629
Malaya	[4] 163	---	---	---	---	---	---	---	---
Taiwan (Formosa)	---	---	---	(3)	(3)	(3)	(3)	700	(3)
Africa:									
Egypt	152	---	---	50	---	---	---	---	(3)
French Morocco	262	640	400	290	72	74	131	21	98
Kenya	3	---	---	---	---	---	---	35	(3)
Madagascar	9,185	6,315	5,170	7,684	9,141	14,013	18,338	18,478	13,325
Mozambique	---	200	126	90	110	---	240	---	---
Portuguese East Africa	---	---	---	---	---	---	---	---	---
South-West Africa	1,318	1,193	1,639	1,627	2,264	1,380	2,626	1,184	(3)
Southern Rhodesia	6	---	---	---	---	---	---	---	---
Spanish Morocco	100	[4] 120	[4] 150	25	15	3	---	17	(3)
Union of South Africa	196	278	221	172	107	244	328	353	375
Australia	114	353	308	235	126	147	135	81	(3)
Total (estimate)	155,000	75,000	97,000	170,000	170,000	160,000	195,000	185,000	180,000

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[1] In addition to countries listed, graphite has been produced in Bulgaria, Greenland, and Myasaland, but production data are not available. No estimates for these countries are included in totals.

[2] Bureau of Mines not at liberty to publish figures.

[3] Data not available; estimates by author included in total.

[4] Estimated figures.

[5] Data not available.

[6] South Korea only.

TECHNOLOGY

GEOLOGY

Graphite occurs most commonly in metamorphic rocks but also is found in sedimentary and igneous rocks. Highly metamorphosed gneisses, schists, and limestones contain the most productive graphite deposits. Bedded deposits such as those in Mexico, Korea, and Rhode Island have probably been derived from coal beds that have been converted into graphite by heat and pressure during metamorphism of the rocks. The formation of flake graphite in gneisses, schists, and marbles is supposed to have occurred during dynamic metamorphism, when organic matter derived from plants or animals was present in the original sedimentary rocks. Graphite in igneous rocks, including pegmatites, is believed to have existed as original gases of hydrocarbons or as oxides of carbon present in volcanic lavas or derived from carbonaceous rocks cut by igneous intrusions.

Minerals commonly found associated with graphite ores are those usually occurring in gneisses, schists, and marbles. Quartz, feldspar, mica, calcite, pyrite, iron oxides, and various silicate minerals of lime, magnesia, and alumina are almost invariably associated with graphite and constitute the principal gangue minerals. Where pronounced weathering of graphite ores has taken place, many of the above minerals have been altered to clays, and in some instances much of this clay has been removed by nature, resulting in considerable enrichment.

MINING

Graphite mining usually is confined to the deeply weathered portions of metamorphic deposits and is therefore relatively easy and cheap. In the United States opencut mining is widely practiced. The ores break easily, and only occasional light blasting is required. Steeply dipping vein deposits, such as those in Ceylon, require underground mining methods involving considerable mining expense. Bedded deposits in Mexico are mined by underground methods because of the thickness of the beds, the thickness of the overburden, and the fact that the beds are folded with pinches and swells on the limbs of the folds. In Madagascar the graphite ore, formerly dug by hand, is being mined using drills, bulldozers, scarifiers, and other earth-moving equipment obtained through assistance from the United States Government. The ore is mined from open pits, usually not exceeding 1,200 feet in length. Water is brought, often many miles, by means of ditches and flumes, to the top of the deposit selected for mining and caused to flow over and down the side of the deposit along the line of the expected pit. Workmen stand on the steep slopes of the pit and shovel the soft, weathered ore into the flowing water; it is carried in a natural channel to a mill, usually less than a mile away in the valley below. As the channel deepens, the sides of the pit are dug back. Overburden is removed in the same manner but bypassed around the mill. The swiftly flowing water not only transports the ore, but the resulting slaking and tumbling action breaks the claylike material and serves as the first step in separating the graphite from the gangue.

MILLING

Amorphous graphite that requires concentration usually is not worth mining. Preparation for foundry facings and paints requires grinding only, and coarse impurities may be removed by screens or air separators. Ceylon graphite is refined by hand methods involving hand cobbing and sorting, wiping lumps with wet burlap, and winnowing to remove fines. Flake graphite from disseminated deposits must be concentrated to meet market requirements. Virtually every known concentrating device and combination of separating principles has been tried at one time or another. Probably in no other industry has such a large proportion of the mills failed to make commercial recoveries. Many so-called secret processes have been employed to concentrate graphite, and the mineral has gained a reputation of being difficult to concentrate. Actually it is one of the easiest minerals to segregate into a rough concentrate and one of the hardest to refine.

Since a premium is placed upon the grain size of flake graphite the problem in milling is simply one of grinding to free the graphite without destroying the flake size. Obviously this is difficult because of the brittle nature of graphite and the fact that any form of crushing forces fine grains of gangue into the soft graphite.

Graphite floats very readily, hence flotation has become the accepted method for beneficiating disseminated ores. High recoveries are common, but acceptable graphitic carbon contents are difficult to attain. In grinding, fine graphite is smeared on the surfaces of gangue minerals, making them floatable and necessitating repeated cleaning to permit recovery of high-grade products. Sometimes it is not possible, even then, to obtain high-grade material.

Graphite with high graphitic carbon may be produced by chemical treatment of low-grade graphite with hydrofluoric acid or sodium hydroxide to remove silica. The cost, however, is excessive, and the treatment is practiced commercially only in Europe (Germany and France) on an extremely small scale. Most refining of flotation concentrates for high-purity products is done by further grinding in rolls or burrstones, followed by screening. This procedure depends on selective grinding of the brittle gangue, leaving the tough flake untorn. Fragile flake cannot withstand this treatment.

STRATEGIC POSITION

The United States never has been self-sufficient with respect to those grades of graphite designated as strategic and critical. These grades are 97 to 98 percent graphitic carbon amorphous lump; crystalline flake graphite, 85 to 87 percent graphitic carbon, suitable for crucibles; and certain grades of graphite used for lubricants and packings. A portion of the requirements of lubricating- and packing-grade graphite is produced in the United States, but high-grade amorphous lump and crucible-grade crystalline flake for certain applications are imported from Ceylon and Madagasacr, respectively. Stockpile objectives for all three grades either were achieved or contracted for in 1953. All other grades of graphite can be supplied by domestic mines or imported over land routes from Mexico. The only producing mine and mill in Canada ceased operating March 31, 1954.

Exploration projects for crucible-grade graphite were eligible for Government assistance in 1954 under the Defense Minerals Exploration Administration program. The Government advanced 50 percent of the allowable cost of an exploration project.

Experiments conducted during 1953 and 1954 showed that appreciable quantities of crucible flake graphite can be produced at the Government-owned Benjamin Franklin graphite mine near Chester Springs, Pa., at a cost 2 to 3 times that of imported material.

PRODUCTION

Production of natural graphite in the United States has never been large, despite many attempts to mine domestic ores. Only in war time and with Government assistance have any domestic mines been able to exist. One reason why the production of graphite in the United States never has been large is the small tonnage of any one grade of graphite consumed, except low-cost amorphous graphite. Another and more important reason is the high cost of production, owing to the low grade of ore.

Table 2 shows the combined amorphous and crystalline graphite production and shipments in the United States since 1915, as compiled by the Bureau of Mines.

CONSUMPTION

The Bureau of Mines coverage in its graphite-consumption canvass is incomplete, and a strict accounting of the uses for which available graphite is consumed would be difficult. Not the least of the difficulties is the overlapping of the materials "used" by one establishment after being "processed for resale" by another. Also, the confusion among some graphite consumers caused by insufficient data on the types and varieties of graphite used tends to make canvass data misleading; however, the totals in table 3 show at least the minimum quantities of graphite used in making various products from 1949 to 1953, the only recent years for which statistics are available. It is believed that the figures are substantially correct for critical uses. The deficiencies are probably largely in the quantities reported for foundry facings, steelmaking, and other noncritical uses.

TABLE 2.—*United States production and shipments of natural graphite, 1915–39 (5–year averages) and 1940–53 (annual).*

Year	Production (short tons)	Shipments (short tons)	Value
1915–19	9, 362	(1)	(1)
1920–24	5, 216	(1)	(1)
1925–29	5, 482	(1)	(1)
1930–34	(1)	(1)	(1)
1935–39	(1)	(1)	(1)
1940	(1)	(1)	(1)
1941	(1)	(1)	(1)
1942	7, 120	(1)	(1)
1943	9, 938	(1)	(1)
1944	5, 408	(1)	(1)
1945	4, 888	5, 334	$289, 207
1946	5, 575	4, 844	252, 596
1947	4, 387	5, 207	221, 260
1948	9, 949	9, 871	450, 759
1949	6, 102	5, 213	475, 264
1950	5, 101	5, 605	427, 908
1951	7, 135	6, 808	771, 434
1952	5, 606	5, 081	594, 618
1953	6, 281	4, 850	488, 008

[1] Not available.

FOREIGN TRADE[4]

New York has always been the principal port of entry for foreign graphite, although the tonnage passing through Arizona from Mexico during the last 15 years has exceeded the total for New York; however, since virtually all the higher priced Ceylon and Madagascar varieties enter at New York, that port still leads in value of the graphite imported.

The United States has had a tariff on graphite since 1913, and the present rates were placed in effect on January 1, 1948. They are: Amorphous, natural and artificial, 5 percent ad valorem; crystalline flake, 15 percent ad valorem, with a specific minimum of 0.4125 cent per pound and a specific maximum of 0.825 cent per pound; crystalline lump, chip, and dust and other crystalline fines, 7½ percent ad valorem.

Imports of natural and artificial graphite, compiled by the United States Department of Commerce, are shown in table 4 for 1940 to 1953.

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[4] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

TABLE 3.—*Consumption of natural graphite in the United States, 1949—53, by uses*

[Short tons]

Use	1949	1950	1951	1952	1953
Crucibles	2, 035	2, 200	3, 273	3, 275	3, 383
Retorts	96	335	696		
Stoppers, sleeves, and nozzles	912	1, 080	1, 466		
Foundry facings	5, 525	6, 581	16, 112	11, 098	11, 944
Batteries	2, 626	3, 314	3, 189	2, 372	2, 003
Lubricants	2, 290	3, 624	5, 579	2, 494	4, 960
Pencils	845	1, 179	2, 276	1, 816	1, 601
Paints and polish	186	175	330	552	673
Packings	178	127	844	754	441
Bearings	22	46	84	80	111
Carbon brushes	(1)	(1)	(1)	805	646
Steelmaking	(1)	(1)	(1)	(1)	7, 751
Brake linings	(1)	(1)	(1)	(1)	588
Other	[2] 1, 587	[2] 2, 217	[2] 4, 469	[3] 3, 665	[4] 720
Total	16, 302	20, 878	38, 318	26, 911	34, 821

[1] Quantity not available.
[2] Includes brake linings, carbon brushes, electrodes, etc.
[3] Includes brake linings, roofing granules, recarburizing steel, etc.
[4] Includes roofing granules, rubber goods, adhesives, and insulation.

TABLE 4.—*Graphite (natural and artificial) imported for consumption in the United States, 1940–53*

(U. S. Department of Commerce)

	Crystalline				Amorphous				Total	
	Flake		Lump, chip, or dust		Natural		Artificial			
	Short tons	Value	Short tons	Value	Short tons	Value	Short tons	Value	Short tons	Value
1940	6, 551	$340, 396	752	$54, 027	23, 766	$487, 675	260	$9, 187	31, 329	$891, 285
1941	4, 877	289, 322	2, 900	370, 413	33, 382	1, 363, 434	38	1, 384	41, 197	2, 024, 553
1942	6, 031	412, 919	4, 152	601, 150	33, 569	1, 480, 079	169	4, 739	43, 921	2, 498, 887
1943	5, 311	493, 382	1, 012	117, 795	22, 390	331, 800	205	15, 515	28, 918	958, 492
1944	6, 191	663, 231	1, 568	251, 832	18, 131	319, 657	131	10, 299	26, 021	1, 245, 019
1945	2, 883	286, 532	5, 207	558, 242	28, 042	569, 600	154	6, 223	36, 286	1, 420, 597
1946	3, 337	253, 163	56	7, 990	29, 743	1, 065, 835	4	558	33, 140	1, 327, 546
1947	2, 730	255, 556	198	16, 325	40, 703	1, 236, 734	28	2, 660	43, 659	1, 511, 275
1948	3, 496	429, 557	554	83, 226	48, 150	1, 529, 312	117	4, 153	52, 317	2, 046, 248
1949	2, 228	277, 368	235	27, 313	29, 298	954, 388	44	1, 398	31, 805	1, 260, 467
1950	6, 130	725, 172	100	7, 514	37, 255	1, 335, 142	184	12, 518	43, 669	2, 080, 346
1951	10, 227	1, 412, 787	336	29, 096	43, 830	1, 561, 494	90	7, 420	54, 483	3, 010, 797
1952										
Canada	181	53, 695	--------	--------	1, 326	128, 320	334	17, 441	1, 841	199, 456
Ceylon	--------	--------	56	9, 664	3, 228	648, 863	--------	--------	3, 284	658, 527
France	30	13, 228	--------	--------	--------	--------	--------	--------	30	13, 228
Germany, West	457	79, 569	--------	--------	221	21, 758	--------	--------	678	101, 327
India	--------	--------	--------	--------	28	3, 875	--------	--------	28	3, 875
Madagascar	8, 210	1, 327, 024	--------	--------	--------	--------	--------	--------	8, 210	1, 327, 024
Mexico	--------	--------	--------	--------	27, 321	447, 248	--------	--------	27, 321	447, 248
Mozambique	--------	--------	--------	--------	100	4, 914	--------	--------	100	4, 914
Norway	--------	--------	11	1, 014	1, 277	100, 963	--------	--------	1, 288	101, 977
Switzerland	--------	--------	--------	--------	3	1, 061	3	1, 061	6	2, 122
United Kingdom	--------	--------	(1)	55	(1)	33	--------	--------	(1)	88
Total	8, 878	1, 473, 516	67	10, 733	33, 504	1, 357, 035	337	18, 502	42, 786	2, 859, 786
1953										
British East Africa	11	1, 356	--------	--------	27	1, 168	--------	--------	38	2, 524
Canada	292	67, 463	--------	--------	2, 762	264, 977	281	15, 200	3, 335	347, 640
Ceylon	--------	--------	77	6, 650	1, 570	171, 204	--------	--------	1, 647	177, 854
Colombia	--------	--------	--------	--------	28	3, 866	--------	--------	28	3, 866
France	2	662	--------	--------	--------	--------	--------	--------	2	662
Germany, West	347	50, 115	1	863	1, 137	126, 190	--------	--------	1, 485	177, 168
Madagascar	9, 927	1, 489, 364	--------	--------	--------	--------	--------	--------	9, 927	1, 489, 364
Mexico	--------	--------	--------	--------	34, 136	553, 443	--------	--------	34, 136	553, 443
Norway	--------	--------	--------	--------	678	50, 945	--------	--------	678	50, 945
Switzerland	--------	--------	--------	--------	5	1, 922	2	447	7	2, 369
Union of South Africa	--------	--------	--------	--------	27	753	--------	--------	27	753
United Kingdom	--------	--------	1	445	12	2, 145	--------	--------	13	2, 590
Total	10, 579	1, 608, 960	79	7, 958	40, 382	1, 176, 613	283	15, 647	51, 323	2, 809, 178

[1] Less than 0.5 ton.

Exports of graphite from the United States consist almost entirely of processed material and in recent years have averaged 1,000 to 2,000 short tons a year. Virtually all of the graphite exported is amorphous material originating in Mexico.

PRICES AND COSTS

Importers buy graphite by the metric and long ton; crucible manufacturers, by the pound. Present prices paid by crucible manufacturers range from 11½ to 16 cents a pound depending upon quality and screen analysis. Prices quoted in E&MJ Metal and Mineral Markets for December 16, 1954, are as follows: Per pound, carlots, f. o. b. shipping point (United States), crystalline flake, natural, 85–88 percent C, crucible grade, 13 cents; 96 percent C, special and dry usage, 22 cents; 94 percent C, normal and wire drawing, 19 cents; 98 percent C, special for brushes, etc., 26½ cents. Amorphous, natural, for foundry facings, etc., up to 85 percent C, 9 cents. Madagascar, c. i. f. New York, "standard grades 85–87 percent C," $235 per short ton. Special mesh, $260; special grade, 99 percent C, nominal.

Amorphous graphite, Mexican, f. o. b. point of shipment (Mexico), per metric ton, $9 to $16, depending on grade.

CONSERVATION AND RESEARCH

Little effort is made to conserve graphite by industry because of the relatively small quantities consumed by individual companies. The cost of graphite as a raw material in manufacturing processes is negligible compared with the cost of finished products, hence attempts to conserve it are made only during times of extreme shortage. Losses of graphite in fabricating crucibles may amount to 25 percent; rejects after firing range from 5 to 8 percent. Thus, as much as one-third of the graphite imported for use in making crucibles may be lost in fabricating.

Research by industry is not extensive, but some work is being done toward developing formulas that will permit use of finer flake in crucible manufacturing. Several manufacturers have indicated that domestic graphite up to about 10 percent can be mixed with Madagascar flake without greatly impairing the life of crucibles. The National Bureau of Standards conducted research that indicated that domestic graphite of equivalent flake size and carbon content can be substituted entirely for Madagascar graphite without impairing the life of the crucible (*4*). These tests have been successful but have not resulted in increased use of domestic graphite because of a lack of such material in the proper sizes at competitive prices. Bureau of Mines work in the past also has indicated that domestic material of equivalent flake size is equal to Madagascar flake graphite.

Current research, as far as known, is confined to the work of the Bureau of Standards in developing formulas for crucibles that will employ domestic graphite of fine flake size (*5*, *6*).

OUTLOOK

World reserves of graphite are adequate for all foreseeable requirements. Large reserves of low-grade amorphous graphite are available in Mexico, Norway, Canada, Ceylon, Korea, Hong Kong, and the United States; except for 97 to 98-percent graphitic carbon grade, amorphous lump graphite is plentiful in Ceylon; and crucible-grade and other types of crystalline flake graphite are abundant in Madagascar, Kenya, and Germany. Reserves of domestic crystalline flake graphite in fine sizes are large but low in grade.

For noncritical uses the production of Mexican amorphous graphite is adequate to supply all requirements of the United States for material of this type. The production of domestic flake graphite can be increased by reactivating idle mines. This would require Government assistance or a substantial increase in the price of flake graphite.

The supply outlook for crucible-grade flake graphite is favorable as long as communications with Madagascar remain uninterrupted; in addition, Kenya offers a possible expanding source of supply. If foreign supplies were cut off, high-cost material is obtainable from domestic deposits.

There is no present indication of a trend either to increase or decrease the use of graphite crucibles in the United States. In recent years the quantity of graphite crucibles used for steel melting has been negligible; but nonferrous melting practice continues to use graphite crucibles extensively, and the present demand will probably continue.

Domestic requirements for current consumption are approximately 4,000 tons a year of crucible flake graphite; however, the need for this grade of graphite is decreasing because of the development of carbon-bonded silicon carbide-graphite crucibles, in which the percentage of graphite is substantially less than that used in clay-graphite crucibles.

PROBLEMS

To compete with foreign sources of flake graphite, domestic mills must operate at capacities high enough to permit low unit costs for their products, making it imperative that they gain a relatively large proportion of the domestic market. Consumers are reluctant to change from satisfactory long-used products to new products that require adjustments in formulas and manufacturing techniques, especially without assurance of long-term continuing supplies. Thus, even assuming that domestic producers can supply equal or superior grades of flake graphite at the same or lower prices, the major problem of domestic producers is one of marketing. The world demand for all types of graphite is relatively small, and all producing countries are competitive for the limited market.

Mexican producers have experienced difficulties in procuring adequate labor and freight cars in an emergency, thus creating temporary shortages of amorphous graphite in the United States.

Another problem is the lack of knowledge concerning the degree of interchangeability possible among various types of graphite and among graphites of the same type from different sources. Detailed consumption data and minimum specifications for the grade of graphite required for specific applications are not available. A wide variety of grades of graphite is imported into the United States, blended, and sold under trade names, thus obscuring the origin of the material. The lack of this information hampers industry and Government planning, especially for mobilization.

Some progress has been made on the problem of achieving independence from foreign crucible flake graphite by both industry and the Government. The use of carbon-bonded crucibles is a partial answer to the problem. Carbon-bonded crucibles do not require as coarse material as clay-bonded crucibles, but they are not entirely interchangeable in use. Industry still requires clay-bonded crucibles for larger sizes and special melting procedures. One method of achieving independence would be development of crucible formulas that would permit the use of finer flake sizes. Methods might be developed to consolidate domestic small-flake graphite into strong, flakelike aggregates. Improved milling methods might permit recovery of more large flake from domestic ores, much of which now may be unnecessarily reduced in size through improper mill treatment.

There are some technical problems in milling graphite ores to produce high-grade lubricant graphite. Graphite, as previously pointed out, is easy to concentrate by flotation but extremely difficult to refine. Cheaper and more efficient processes for refining graphite to increase its market value are required.

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GYPSUM

By

Oliver S. North[1]

THE GYPSUM and gypsum-products industry has grown into a 250-million-dollar business, ranking near the top among the nonmetallics. Consumption of gypsum has more than doubled in the past 10 years; the marked increase was largely due to the growing popularity of prefabricated building materials.

Summary

Consumption of gypsum has increased steadily through the years and reached a new record in 1953.

Commercial gypsum deposits occur in many States; in addition, a large quantity of crude gypsum is imported from neighboring countries for use in product plants in coastal cities. Normally, ample supplies are available from domestic deposits and conveniently located deposits in Canada and Mexico.

Gypsum is mined—usually from open pits—and crushed to meet the specifications for subsequent use or processing. Most gypsum products are made from calcined gypsum, a hemihydrate calcium sulfate formed by driving off part of the combined water. Calcining is accomplished in rotary kilns, where the process is continuous, or in kettles, where calcination is a batch process. Calcined plaster, called "stucco," is removed from the kiln or kettle at the proper stage and cooled. If the stucco is to become wall plaster, a retarding agent and sometimes other materials are interground with it. An agent for accelerating the set is added to calcined material intended for use in prefabricated products. The ground wall plaster is bagged and shipped for use in construction. Prefabricated products, such as wallboard, lath, tile, etc., are manufactured in large, highly mechanized plants.

In its numerous markets, gypsum and gypsum products are noted for low price and versatility in use. Current consumption of this commodity in the average housing unit is nearly three times the 1936 average.

Although the total United States reserves of gypsum are very large, there are areas in which, due to depletion or other factors, a reserve problem exists, particularly in some of the high consuming areas in the eastern United States. There is also need for research into the technical and economic problems of deriving sulfur or sulfuric acid from gypsum, study of the performance characteristics of the various kinds of aggregates used in gypsum products, improvement of mining and processing methods, investigation of the competitive use pattern for gypsum, and dissemination of comprehensive technical, statistical, and economic information on this commodity.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

Measured by the production and sale of gypsum products, the gypsum industry is one of the most important nonmetallic industries in the United States. In each of the years 1950–53 production of crude gypsum in the United States exceeded 8 million tons, and imports exceeded 3 million tons. The apparent crude supply in each year therefore totaled over 11 million tons. The value of gypsum products sold exceeded 200 million dollars in each of those years, with a record 245 million dollars worth of gypsum products marketed in 1953. Table 1 summarizes selected use statistics of the gypsum industry from 1935 to 1953.

SIZE AND ORGANIZATION OF GYPSUM INDUSTRY

The gypsum industry consists of 3 major companies, each with 5 or more mines or quarries and calcining plants; 7 other companies operating mines and/or board-products plants; and a number of smaller firms producing crude gypsum or gypsite for cement retarder or agricultural purposes. The principal producer of both crude and calcined gypsum is United States Gypsum Co., of Chicago, which for many years has held the top position in the industry. In 1953 this organization reported the operation of 13 mines or quarries in 12 States and 19 calcining plants in 16 States. National Gypsum Co., of Buffalo, was rated second, with 5 mines or quarries and 10 calcining plants, while Certain-teed Products Corp., of Ardmore, Pa., was third with 6 mines or quarries and 6 calcining plants. Both United States Gypsum Co. and National Gypsum Co. conduct large mining operations in Nova Scotia.

Active gypsum mines in 1953 were distributed among 19 States; California led in number of mines, and Michigan ranked first in tonnage produced. Other major producing States were Iowa, Kansas, Nevada, New York, and Texas. These same States (New York ranking first) were also the leading producers of calcined gypsum.

TABLE 1.—*Gypsum products (made from domestic, imported, and byproduct crude gypsum) sold or used in the United States, 1935–53, by major use classifications*

[Thousand short tons]

Year	Raw			Calcined		
	Portland-cement retarder	Agricultural gypsum	Other uncalcined	Industrial uses	Building uses	
					Cementitious	Prefabricated
1935	475	79	41	104	1, 043	435
1936	667	74	89	119	1, 431	709
1937	770	75	16	126	1, 669	848
1938	674	68	14	94	1, 485	977
1939	775	75	18	110	1, 780	1, 334
1940	821	92	16	124	1, 863	1, 718
1941	1, 153	149	18	152	1, 939	2, 348
1942	1, 245	252	18	143	1, 157	2, 164
1943	835	382	16	164	769	2, 019
1944	565	477	14	200	754	1, 828
1945	664	462	21	158	880	1, 802
1946	1, 136	472	34	207	1, 938	2, 594
1947	1, 342	576	33	207	[1] 2, 202	3, 257
1948	1, 675	517	34	219	[1] 2, 540	4, 271
1949	1, 528	426	36	212	[1] 2, 321	3, 823
1950	1, 721	465	32	266	[1] 2, 937	5, 062
1951	1, 809	688	34	289	[1] 2, 831	5, 298
1952	1, 815	866	24	252	[1] 2, 551	5, 003
1953	1, 907	722	27	254	2, 581	5, 369

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[1] Revised figure.

MINERALOGY AND GEOLOGY

Gypsum, a common mineral of wide occurrence, is a hydrous calcium sulfate having the chemical formula $CaSO_4.2H_2O$. Pure gypsum in economically minable quantities is rarely found in nature, as the mineral usually occurs mixed with such impurities as the oxides of iron and aluminum, the carbonates of calcium and magnesium, and others. When crystallized the mineral is soft, with a hardness of 2 on Mohs' scale, and its specific gravity is approximately 2.3. Pure gypsum is white, but the material as found often shows other colors due to impurities. Gypsum occurs as rock gypsum, alabaster, gypsite, selenite, and satin spar; rock gypsum is the form in which it is most commonly found and that which is of greatest economic value. The outstanding characteristic of gypsum is its unique property of losing three-fourths of its water of crystallization on heating to comparatively low temperatures. When mixed with water it resets in any desired shape to its original rocklike form; this feature makes it valuable commercially.

In addition to natural gypsum, a quantity of byproduct calcium sulfate is obtained in the production of superphosphate fertilizers and certain other chemicals, but this material is, from a commercial viewpoint, a relatively minor source.

Anhydrous calcium sulfate (anhydrite) often occurs as an impurity in gypsum beds or is found at depth in underground gypsum mines. Many gypsum deposits were originally anhydrite beds that have been altered by the action of meteoric or ground waters. Anhydrite is heavier and harder than gypsum and mainly because of its physical properties does not lend itself to the great variety of uses for which gypsum is adapted. It is often considered an impurity or contaminant in gypsum and whenever possible is bypassed in mining or discarded in subsequent operations.

Most important deposits of gypsum or anhydrite-gypsum are believed to have been formed by the evaporation of sea water.

DOMESTIC OCCURRENCES

Numerous gypsum deposits of good quality occur in many areas. In the Western States deposits in California, Nevada, Utah, Texas, Iowa, Oklahoma, and Kansas have been highly commercialized, and other extensive deposits are known in Montana, Colorado, Arizona, South Dakota, Wyoming, and New Mexico. Smaller quantities were mined in 1953 in Arkansas, Idaho, Louisiana, and Washington. In the East, besides the currently operating large mines in Michigan, New York, Ohio, and Virginia, other potential sources in those same States and in Indiana are known. The Indiana deposits have only recently been discovered, and several companies are building or planning to build plants that will utilize them.

RESERVES

Although the United States has large reserves of gypsum within its borders, many deposits are not near enough to consuming centers to be commercial, while others, which have been worked for many years, are encountering depletion problems.

IMPORTS

For economic reasons a considerable tonnage of the crude material is now imported. Most of the imports come from Canada and are shipped into the New England and Atlantic Coast States, while smaller but important tonnages are imported from Mexico for use in California and Washington. During World War II imports dropped to as low as 6 percent of the domestic supply, as ships were taken over by the various governments for war use, but from 1944–48 the imported crude constituted an increasingly larger percentage of the total crude used. However, in the last few years an apparent balance between the domestic and the imported crude has been reached at about 72 and 28 percent, respectively. No duty is assessed on imported crude gypsum.

TRANSPORTATION

Transportation factors loom large in the economics of the gypsum industry. Because both raw gypsum and most gypsum products are relatively heavy for their value, the plant should be as near as possible to the mine and also to the main consuming area. This ideal condition has been fairly well met by some plants, whereas others, especially in the Western States and on the east coast, have found it necessary to ship either the crude rock, the finished product, or both, for considerable distances. At present the bulk of the raw-gypsum requirements of plants on the Eastern Seaboard is supplied from quarries in Nova Scotia, although some crude from Jamaica and the Dominican Republic is used experimentally.

One large operator in the West found it most practical to build a plant and company town near the mine and ship the finished material a few hundred miles to its principal market area. Another mines a deposit adjacent to an ocean port on San Marcos Island, Lower California (Mexico) and ships the crude in a company-owned freighter over 1,000 miles to 3 processing plants in important marketing areas. Noteworthy from a point of comparison is the fact that, in Southern California, both of these companies sell in the same market.

TECHNOLOGY

The mining of gypsum is accomplished by standard open-pit and underground methods, and the crude is crushed to meet the specifications of subsequent use. Impact mills reduce the size to minus 1½ inches or smaller—a suitable size of feed for rotary kilns and for use as cement retarder. If intended for use as a filler, for agricultural purposes, or to be calcined in kettles, the material is ground to much finer size specifications. For example, agricultural gypsum is usually screened through 40-mesh, and, in general, the finer the particle size the more satisfactory the filler.

Calcining gypsum, wherein the hemihydrate is formed by driving off part of the combined water, may be done in gypsum "kettles" or rotary kilns or under steam pressure in autoclaves. When kettles are used the gypsum must be ground before calcination, but when calcining is done in a rotary kiln, grinding takes place after calcination. Kettles originally were used exclusively, apparently due to the control of conditions that can be maintained during calcining, but in a number of instances they have been replaced by rotary kilns and, more recently, by other newly devised types of calciners.

Kettle calcining is a batch process, with batches ranging from 10 to 15 tons, depending on the size of the kettle. When the mass of ground gypsum has reached the proper temperature, it is important that the calcination process be stopped immediately. To accomplish this a pit of steel or concrete, called a "hot pit," is provided. The "hot pit" is large enough to hold one kettle of plaster, and the object is to discharge the batch into it as quickly as possible. Usually the kettles are emptied completely in two minutes. Hot calcined gypsum flows like water. Some cooling takes place in the "hot pit." Between 30 minutes and 1 hour after the kettle is discharged the material is removed from the "hot pit" by means of conveyors and elevators and transported to bins, where the product is cooled. At this point the material is known as calcined plaster, stucco, or plaster of paris and is the base material for manufacturing gypsum board, gypsum partition tile, roof tile, and various industrial plasters. By varying the time of calcination and/or temperature, several types of plaster of paris are made to meet other needs. Calcined plasters usually have a setting time of 25 to 30 minutes.

The setting time of calcined gypsum used in prefabricated products generally is accelerated. On the other hand, a retarder is used almost always in manufacturing wall plasters to control the time of "setting" to between 2 and 6 hours. Wall plaster is made by tube-milling calcined plaster of paris to very fine particle size. This material is thoroughly mixed with the retarder, fiber, and perhaps small quantities of other materials. Retarder usually is made by processing cattle hair, horns, or hoof meal with caustic soda and lime.

USES

Gypsum and its products are economical materials in their respective fields. Lime and hydraulic cement plasters compete to a limited extent with gypsum in the plaster field. In construction, lime is used today chiefly as an ingredient in the finish coat over gypsum-plaster base coats.

Gypsum plaster is mixed with water and aggregates (such as sand, expanded perlite, or exfoliated vermiculite) in certain proportions and applied, usually in 2 or 3 coats, as an inside finish over gypsum lath, wood lath, metal lath, monolithic concrete or concrete blocks, or other backing. Portland-cement plaster is used chiefly as an exterior stucco, in locations unavoidably affected by dampness, or where subjection to abrasion precludes the use of gypsum plaster.

Gypsum board, lath, and sheathing compete directly with lumber and other building materials used for inside finishing. The low price, ease of handling, fire resistance, and speed of erection of those products assure them a steady market.

In recent years gypsum products have found a growing market in construction work, especially in housing units. Nearly three times as much gypsum lath and wallboard are now used per housing unit as in 1936. This growth has been due in part to relatively high prices and shortages of competitive products and in part to quality advantages.

As a retarder in cement, no other material compares with gypsum for combined economy and effectiveness. The percentage of gypsum added to the various types of cement is nearly constant. Some current experiments on the use of gypsum retarder indicate that a considerably higher gypsum content might help make concrete more lasting. If more cement is produced more gypsum retarder will be required.

Agricultural gypsum is used to offset black alkali (sodium carbonate) and to restore impervious, dispersed soils that have been deflocculated by overdoses of ammonia-rich fertilizers or by soda in irrigation water. Ground gypsum and anhydrite are used in Southern States as soil conditioners for the peanut crop and are considered useful for other leguminous crops. Agricultural gypsum competes with a variety of natural and manufactured materials, and its cheapness in areas surrounding gypsum deposits (or into which it can be economically shipped) makes it a preferred soil additive. Consumption of this product usually is restricted by freight rates to a limited radius about a deposit.

Gypsum and anhydrite, particularly the latter, may be converted to ammonium sulfate fertilizer by processing with coke and ammonia.

Besides the large tonnage outlets of building plasters and prefabricated building products, calcined gypsum is used also for the following "industrial" plasters: Terra cotta, pottery, molding, orthopedic and dental, and statuary, and, in the plate-glass industry, to hold glass while it is being ground and polished; in the glass batch it may be used in lieu of salt cake. Miscellaneous uses for pure white uncalcined gypsum are as "terra alba," which is used as a filler in paper and paints, and as a nutrient in growing yeast. Off-white uncalcined gypsum is used as a filler in various products or as an insecticide carrier. In 1953 the quantities utilized for various purposes were as follows:

	Short tons
Uncalcined:	
Portland-cement retarder	1,907,031
Agricultural gypsum	721,993
Uncalcined fillers, rock dust, in brewer's fixe, in color manufacture, and other	27,422
Calcined:	
Industrial:	
Plate glass and terra cotta plasters	60,290
Pottery plasters	43,957
Orthopedic and dental plasters	10,613
Statuary, industrial casting and molding plasters, dead-burned filler, granite polishing, and others	139,288
Building: Cementitious: Plasters:	
Base coat	1,727,088
Sanded and perlite-premixed	300,603
To mixing plants	11,570
Gaging and molding	168,539
Prepared finishes	12,096
Roof-deck	289,177
Insulating, joint filler, patching and painter's plaster, and others	20,391
Keene's cement	51,475
Calcined—Continued	
Prefabricated:	
Lath	1,864,983
Wallboard and laminated board	3,223,708
Sheathing board	126,876
Tile	153,617

The possibility of using gypsum or anhydrite as a raw material from which to manufacture sulfuric acid has attracted much attention because of its technical and commercial success in Europe and because of the recent shortage of elemental sulfur. Different chemical processes for accomplishing this have been tried; currently the most favored is that of calcining gypsum or anhydrite at high temperatures along with coke, silica, and clay, from which SO_2 and SO_3 gases (and byproduct cement) result. These two gases then are converted to sulfuric acid. To date gypsum and anhydrite are considered to be uneconomical sources of sulfur in the United States; but should better methods be discovered or if sulfur prices increase sufficiently, a new field for large-tonnage usage of raw gypsum would be opened.

RESEARCH

Leading members of the gypsum industry maintain research laboratories and engineering staffs that are concerned with the technical problems of the industry. Progress has been made in methods of mining, crushing, calcining, and end-product manufacture.

Some long-range studies now in progress in the industry relate to the chemistry of the material and seek to determine methods of application productive of maximum usefulness. These include investigations of the general problem of adhesion of calcium sulfate plasters to other materials, study of aggregates problems, study of the effect of gypsum (retarder) content on compressive strengths of cements, and investigation of the compressive strengths of plasters mixed and applied under varying conditions.

OUTLOOK

An outstanding trend through the past several years has been the shift of the building industry to the use of gypsum wallboard in preference to the application of 2 or 3 coats of plaster mortar. Because of its speed of erection and favorable initial cost compared to plastered walls, it is widely expected that the ratio of this construction will continue to increase in the next few years.

Gypsum consumption bears a close relationship to local and regional developments, especially as regards housing activity. When population in a locality or area is markedly increasing or when commercial activity of any type greatly improves the general welfare of an area, construction in which gypsum plays a part is increased, and the demand for gypsum products rises. The well-being of the industry, therefore, is closely tied to the current encouraging outlook for residential construction.

Consumption of agricultural gypsum has increased sharply in recent years; however, in 1953 and 1954 a drop in consumption was noted. Factors contributing to this decline apparently have been reduction of expenditures for such materials by farmers and some reduction of the subsidies under the Agricultural Conservation Program. Therefore the present outlook for this material is somewhat uncertain. The recently developed chemical soil conditioners are not expected to effect adversely the market for agricultural gypsum because they usually do not perform the same functions.

There is a distant prospect of using gypsum or anhydrite as raw material for producing sulfuric acid, as is now being done in several European countries. The prospective costs of present processes are still high, compared with other domestic sources of sulfur; consequently, early commercial use of gypsum as a major source of sulfur in the United States is considered unlikely.

Although no major producing area has been completely abandoned, the geographical pattern of distribution of gypsum-products-manufacturing facilities has shifted somewhat in the last decade or so toward Texas, California, the Southern States, and the Northwestern States. Currently two plants are expected to be in operation by 1955 at the recently discovered deposits in southern Indiana. Because of the encouraging outlook for the industry, some of the larger companies are reported to be contemplating additional gypsum-products plants in New York, California, Louisiana, Texas, and other Southern and Western States.

It is expected that the United States will continue to rely on foreign sources for a portion of its crude gypsum needs—possibly for a larger percentage than in the past if several new processing plants are built along the seacoast, where crude gypsum can be obtained by water from abroad more economically than by rail from deposits in the United States.

Although the gypsum industry is facing economic, technical, and competitive problems, it is expected that consumption of this commodity will continue to grow during the next few years in accordance with the requirements of the construction industry.

PROBLEMS

Problems confronting the gypsum industry include development of new uses and markets for its products, improvement of processing and use practices, and the usual economic problems of cost and distribution.

A major problem facing the gypsum industry is product improvement and development. At least three large producers recently have built laboratories in which to undertake extensive research on products. These new laboratories have been built mainly to enable the industry to develop and improve gypsum materials and thereby maintain or improve its position in the highly competitive construction and industrial markets.

There is need for further investigation of the competitive use of gypsum.

Although at the current or even an accelerated rate of consumption the overall reserves of gypsum in the United States are large, gypsum-products manufacturers in some areas encounter supply problems and are forced to ship crude material from some distant domestic or foreign source. This problem is particularly important in coastal areas far from gypsum deposits of commercial significance and in other areas where demand is high but reserves are being depleted or have become difficult of recovery.

Mining, crushing, grinding, calcining, and products-manufacturing practices all have been improved, but there is a continuing need for further improvement of production and processing methods which will provide greater efficiency and higher quality of products. Fundamental research is receiving increasing attention from the gypsum industry as better understanding of phases, crystal structure, and formation conditions are needed.

The growing use of lightweight vermiculite and perlite as aggregates replacing sand in gypsum plaster necessitate thorough investigation of plaster strengths, chemical reactions between gypsum and aggregates, and methods of mixing, placing, and curing lightweight plasters.

Compilation and distribution of detailed and comprehensive technical, economic, and statistical information is needed to provide the industry and Government agencies with the basic facts required for the most effective management and guidance of policy decisions.

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HELIUM

By

Henry P. Wheeler, Jr.[1]

Helium occurs as a minor constituent in a limited number of natural gases in the United States. Produced in quantity only by the Federal Government, it was not available for public sale until 1937. Helium was required in large quantities in World War II for military purposes; and postwar developments have led to an increased industrial demand primarily as an inert gas shield in shielded-arc welding of aluminum, magnesium, stainless steel, and other metals. About 191 million cu. ft. was produced in 1954, with an estimated demand of 210 million cu. ft. in 1955. The largest demand is for direct shipment to Government agencies, including the Navy, Army, Air Force, Weather Bureau, and Atomic Energy Commission.

Summary

In the 12 months ended December 31, 1954, Bureau of Mines helium plants produced 190.9 million cu. ft. of helium—more than was ever produced before in any year. The plants shipped 189.8 million cu. ft. in the same period—128.5 million to Federal agencies and 61.3 million to private industry, hospitals, laboratories, and other non-Federal customers. Approximately 65 percent of the helium shipped to non-Federal customers was used in connection with national defense contracts; another 23 percent was used in hospitals and for other medical purposes. Only 12 percent of the non-Federal sales—about 4 percent of the total sales—was used for purposes other than Federal, medical, or national defense contracts.

The Federal Government's interest in helium dates back to World War I, when a safe, nonflammable lifting gas was needed for lighter-than-air ships, and the Government conceived the idea of producing helium in quantity. Experimental work was begun by the Bureau of Mines in 1917, with cooperation from commercial concerns, and the first full-scale helium plant started to operate in 1921. Subsequent development of the industry to its present status has been achieved almost entirely through the finances and efforts of the Government—principally the Bureau of Mines, which has had full supervision of the Government's helium program since 1925.

The Helium Act of September 1, 1937, authorized the Bureau to sell helium not needed by Federal agencies to non-Federal customers. These sales have helped to stabilize the overall helium demand and enabled the Government to share with private industry the cost of finding and producing helium.

The usefulness of helium arises out of its unique properties. Before World War II, it was used almost exclusively as a lifting gas, although small but important quantities were used in hospitals and diving operations. During the war, however, a new use—helium-shielded arc welding—was developed, and helium's importance as an industrial gas has grown steadily since. The gas that filled blimps which escorted 89,000 surface craft in World War II without the loss of a single craft to enemy submarines is being used today in welding

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[1] Helium engineer, Bureau of Mines.

aluminum, magnesium, titanium, and stainless steel in the manufacture of aircraft, submarines, and other vessels. It is vital to the operations of the Atomic Energy Commission and is the subject of much research in laboratories seeking new information about the structures and energies of matter.

To meet the demand for helium, the Bureau of Mines is operating 4 helium plants with a combined annual capacity of 206 million cu. ft. of helium. These 4 plants will not be adequate to meet the anticipated demand for helium past 1955, when Federal agencies and non-Federal customers are expected to use about 210 million cu. ft. The Congress has provided funds under a deficiency appropriation (1955) for a fifth plant. The combined capacity of the 5 plants should care for the growth of the industry for the next 5 years.

In addition to new production and shipping facilities, the Bureau of Mines will need new sources of helium supply to meet the ever-growing demand for helium in the future. Fortunately, helium-bearing gas reserves other than those connected to the Bureau's plants are known to exist.

The Bureau proposes to expand and accelerate its helium research to keep pace with the growing demand for helium. This research will be directed primarily toward improvements in processes and equipment for producing, purifying, and shipping helium, including methods applicable to the extraction of helium from natural gases likely to be processed in the future.

Helium demands will continue to increase, and the Bureau of Mines will plan new helium facilities as required. The sources of helium-bearing gas are large enough to insure a supply for several years.

BACKGROUND

HISTORY (*1*)[1]

In 1868 Janssen, Lockyer, and others showed that the solar prominences give a bright-line spectrum proving that they are composed of hydrogen and other gaseous elements. Lockyer noted a line in the spectrum that he could not identify with any element known on earth at that time and correctly assumed that he had discovered a new element, which he called helium—after the Greek word "helios," meaning sun.

William Ramsay discovered helium on the earth in 1895 in a gas that he extracted from cleveite, a uranium-bearing mineral. It was found to be a constituent of natural gas in 1905 when Cady and McFarland, of the University of Kansas, analyzed a sample from a field that had been discovered near Dexter, Kans., in 1903. This gas contained about 2.87 percent of helium.

At the beginning of World War I, helium was a laboratory curiosity extracted in small quantities at a cost equivalent to about $2,000 per cubic foot. The need for a safe, nonflammable lifting gas for airships stimulated considerable interest in helium, and R. B. Moore of the Bureau of Mines suggested in 1917 that it might be produced in enough volume to serve that purpose.

Consequently, the production of helium was undertaken by the Government. Three private companies—Linde Air Products Co., Air Reduction Co., and Jeffries-Norton Corp.—cooperated in the original helium-production venture, but there is little likelihood that any of them considered undertaking helium production on its own initiative and with its own funds. There was virtually no demand for helium other than that proposed for the Government's airships, and the development of a helium-production process involving extremely low temperatures and high pressures presented many new problems.

Each of the three private companies contracted with the Government to design, construct, and operate an experimental helium plant to process natural gas from the Petrolia field of Clay County, Tex. Funds for the work were provided by the Army and Navy, and the plants were built under the direction of the Bureau of Mines. About 200,000 cu. ft. of helium was produced, and 147,000 cu. ft. was ready for shipment to France when the Armistice was signed. In December 1920, the Navy's C–7 was filled with helium at Bolling Field, District of Columbia, and made the first flight of a helium-filled airship.

A full-scale helium plant, built by the Navy near Fort Worth, was placed in operation in April 1921. Helium-bearing natural gas was supplied through a pipeline from the Petrolia field. Linde Air Products Co. was the contractor for the design, installation, and operation of the helium-separation equipment. In 1925 this plant and other governmental helium activities were placed under the Bureau of Mines. The plant produced a little more than 46 million cubic feet of helium before it was shut down on January 10, 1929, because the Petrolia field was seriously depleted and the supply of gas was failing.

A board of three outstanding engineers—called the Board of Helium Engineers—had been appointed in August 1921 to study helium-production processes. That board, in collaboration with members of the staff of the Bureau of Mines, designed a production cycle that was tested in a semicommercial-scale plant built and operated by the Bureau of Mines at Fort Worth, Tex., in the fiscal years 1923 and 1924. A production-scale unit based on the results of the semicommercial tests was completed at Fort Worth in June 1926 and tested there in trial operation. It was moved to Amarillo, Tex., in 1928 and after considerable modification by members of the Bureau's staff became the first unit of the Amarillo plant. That unit, as modified, incorporated the basic principles of the helium-production cycle used today, but many changes and improvements have been made in the intervening years.

In 1926 the Bureau of Mines initiated negotiations to acquire the helium-bearing gas reserve in the Cliffside field near Amarillo, Tex., because it was evident that the Petrolia field was nearly exhausted. In an act approved February 24, 1927 (*5*), the Congress appropriated $1,063,000 to the Bureau of Mines (then in the United States Department of Commerce) for additional helium-production facilities.

In 1927 the Kentucky Oxygen-Hydrogen Co.—a subsidiary of the Girdler Corp. of Louisville, Kentucky, and later known as The Helium Co.—became interested in the production of helium. The venture of this company represents the only attempt by industry to engage in helium production with private financing. With little prospective commercial demand and un-

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[1] Italicized figures in parentheses refer to items in the bibliography at end of this chapter.

satisfactory sources of helium-bearing natural gas, the company ceased operations after a little more than 2 years of limited production.

A contract for a supply of helium-bearing gas from the Cliffside field, to supply a new helium plant to be built near Amarillo, Tex., was completed in May 1927. That contract contained an optional provision under which the Bureau of Mines later acquired the leaseholds from which the gas was supplied. In subsequent years the Bureau purchased all other gas rights in 50,000 acres covering the entire Cliffside field, so the Government holds the gas in fee.

In 1928 construction of the new plant at Amarillo was begun. The production-scale unit that had been tested at Fort Worth was modified and installed at Amarillo as the first unit of the new plant. A second unit was added a little later.

With acquisition of the Cliffside field (which had a reserve of more than 2 billion cu. ft. of helium), and construction of the Amarillo Helium Plant, which began to operate in April 1929, the Government built a firm foundation for its helium program.

Legislation was initiated by the United States Department of the Interior, which eventually emerged as the act of September 1, 1937, authorizing the Secretary of the Interior, through the Bureau of Mines, to sell helium for medical, scientific, and commercial uses, as well as to continue to supply helium to Government agencies. At the instance of The Helium Co., the Congress included a provision authorizing and directing the Secretary to acquire the company's helium properties, if possible under the terms of the act.

The properties purchased from The Helium Co. were never used for helium production by the Government. Careful studies by the Bureau in planning to meet the increased demands of World War II showed that the plants at Dexter and Thatcher could not produce enough helium to warrant their operation. Therefore, those plants were dismantled, and the equipment and materials were salvaged.

The Bureau welcomed the opportunity to sell helium to private users. These sales, although relatively small, helped considerably in enabling the Bureau to hold its helium organization together until it was needed again to meet a large Government demand. It also enabled the Government to share the cost of finding and producing helium with private industry, resulting in a lower unit cost to the Government than if it had to bear the costs alone.

The threat of World War II brought renewed demands for helium. In 1940 the total demand was 9.6 million cu. ft., and the Armed Forces indicated that the demand might soon exceed the capacity of the Amarillo plant.

After the Japanese attacked Pearl Harbor on December 7, 1941, and the United States declared war on Germany and Japan, the estimates of future demand by the Armed Forces were revised sharply upward, reaching a peak in June 1942 when the Navy was authorized to increase its fleet of blimps to 200. At that time the forecasts indicated a monthly demand of 5,450,000 cu. ft. in July 1942, rose rapidly to a peak of 19,300,000 cu. ft. in August 1943, declined to about 16,300,000 cu. ft. in October 1943, and leveled off at that rate.

In spite of the rapid progression of estimates, the Bureau of Mines met every demand on schedule. At no time were the Army and Navy forced to curtail activities because of insufficient helium production. The Bureau was able to accomplish this important job because of its previous experience in designing, constructing, and operating helium plants and its knowledge of the Nation's helium reserves.

The Bureau increased the capacity of the Amarillo plant and constructed 4 new plants to meet the anticipated demand of almost 20 million cu. ft. a month. The designed capacities of the plants, in cubic feet of helium measured on a base of 14.7 p. s. i. a. and 70° F., and the dates that the plants began operating were as follows:

Plant	Initial operation	Designed capacity (million cu. ft. per month)
Amarillo, Tex	April 1929	[1] 3
Exell, Tex	March 1943	5
Otis, Kans	October 1943	4
Cunningham, Kans	January 1944	4
Navajo (Shiprock) N. Mex	March 1944	4
Total		20

[1] Capacity after expansion in 1942.

The cost of the wartime expansion was a little over 15 million dollars, which included new construction at Amarillo, Exell, Otis, Cunningham, and Navajo (Shiprock) and the cost of leasehold gas rights and the payment of advance rentals and royalties in the Rattlesnake field, which supplies the Navajo plant with helium-bearing gas, plus the cost of drilling wells and constructing pipelines for the Amarillo and Navajo projects, plus general expenses in the field and in Washington.

The maximum annual demand for helium

during World War II was 130.8 million cu. ft. in the fiscal year 1944. The peak monthly demand was 14.6 million cu. ft. in December 1943. With the surrender of Germany on May 7, 1945, the Navy's use of blimps in antisubmarine warfare decreased, and deliveries of helium fell sharply from 110 million cu. ft. in 1945 to only 35.5 million cu. ft. in 1946. The actual consumption of helium in 1946 was somewhat more, because consumers were reducing stocks they had built up during the war.

The Cunningham plant was shut down in July 1945 and dismantled. The helium-bearing gas reserves available to that plant were known to be relatively short-lived and incapable of sustaining a helium plant for a long period. The Amarillo and Otis plants were shut down and placed in standby status in August 1945. The Navajo plant, which was completed about the time the demand started to fall and had been operated for only 19 days in March 1944, also was placed in standby status.

Only the Exell plant continued to operate in the years immediately following World War II, except for about 4 months of operation of the Amarillo plant from June to September 1946. The Exell plant, like the Otis plant, processes helium-bearing natural gas that is produced by private companies for distribution in fuel markets, whether or not the helium is extracted. Therefore, it was operated in preference to the Amarillo and Navajo plants, where the helium-bearing natural gas is controlled by the Government. Operation of the Exell plant offered an additional advantage in that it could be operated continuously at or near its capacity, and helium produced in excess of the current demand could be transported about 20 miles in a 2-inch pipeline and injected into the Government-owned Cliffside field to increase the helium reserve there. Helium also could be transported in the 2-inch pipeline an additional 12 miles to the Amarillo plant, which is located advantageously for handling shipments of helium in standard-type cylinders.

Before World War II, helium was used largely, as a lifting gas; however, small but important amounts were used in hospitals and diving operations. During the war, a new use—helium-shielded arc-welding—was developed; and the importance of helium as an industrial gas has grown steadily since. The non-Federal demand for helium increased from 4.2 million to 21.1 million cu. ft. in the 5 years (1946–50) between the surrender of Germany and the intervention of United Nations forces in Korea. During the same period, the Federal demand ranged from 31.3 million to 40.0 million cu. ft. Nearly all of the demand was met by production at the Exell plant.

In the four years (1951–54) following intervention in Korea, the demand for helium by both Federal and non-Federal users has increased rapidly, and the total demand in 1954 was 189.7 million cu. ft.—59 million more than the World War II peak. To meet this demand, it has been necessary to resume operation of the Amarillo, Navajo, and Otis Helium Plants. Thus, the Bureau is utilizing all of its present helium plants.

ORGANIZATION

The Bureau of Mines is the only producer of helium in large quantities in the United States and probably in the world. Consequently, it is the key unit in the organization of the industry. Except for small amounts of helium that may be produced in scientific laboratories, all of the helium used by the Government and private industry is produced by the Bureau.

The helium functions of the Bureau of Mines are conducted under the overall supervision of an Assistant Director—Helium who heads the Bureau of Mines Helium Activity with offices in the Barfield Building, Amarillo, Tex.

The work of the Helium Activity is carried out under the immediate supervision of the chief, Helium Operations, with four functional divisions—Administration, Production, Engineering, and Gas-Field Operations—and the chief, Division of Research. All these offices are at the Amarillo Helium Plant on the western outskirts of Amarillo. In addition, there is a Helium Liaison Office in Washington, D. C.

Each of the plants at Amarillo and Exell, Tex., Otis, Kans., and Shiprock (Navajo), N. Mex., is supervised by a superintendent at the plant who reports to the chief, Division of Production.

The investment to June 30, 1947, without adjustment for depreciation and depletion, in the 4 plants, and related properties, was about $15,026,000. Of this amount, about $9,170,000 was invested in plants and their equipment, including high-pressure storage; $1,504,000 in camps for employees; $1,514,000 in pipelines; $1,435,000 in gas rights, lands, and rights-of-way; $872,000 in gas wells and gas-field equipment; and $531,000 in facilities for general supervision and services, administrative expenses incurred in the development, and miscellaneous items. Additional investments since June 30, 1947, are minor. The authorized new plant will increase this investment by about $8 million.

Approximately 365 employees are required in the field in the supervision, operation, and maintenance of the 4 plants and the related gas properties and in rendering various services to consumers of helium. As the demand for helium increases and operation of the new plant is begun, the cost of operating the plants and the number of employees required will increase accordingly.

The following organizations have the responsibility for procurement of helium for the largest Federal users:

1. *Navy.*—The Helium Branch, Supply Division, Bureau of Aeronautics, Washington, D. C. After July 1, 1955, this responsibility will be shifted to the General Stores Supply Office, Philadelphia, Pa.
2. *Army.*—The Office of the Chief, Corps of Engineers, St. Louis, Mo.
3. *Air Force.*—The Commanding General, Middletown Air Materiel Area, Olmsted Air Force Base, Middletown, Pa.
4. *Atomic Energy Commission.*—The Procurement and Contracts Branch, Division of Construction and Supply, Washington, D. C.
5. *Weather Bureau.*—The Procurement and Supply Division, Washington, D. C.

In addition to private companies that purchase helium for their own use, over 30 private concerns have purchased helium from the Bureau for resale. Two of these concerns—the Air Reduction Sales Co., 60 East 42d Street, New York 17, N. Y., and the National Cylinder Gas Co., 840 North Michigan Avenue, Chicago 6, Ill.—deserve special mention because they have established central distributing points with facilities for receiving helium in tank cars and reshipping it in standard cylinders. By utilizing tank cars they are able to effect a considerable saving in freight, part of which can be passed on to their customers.

GEOGRAPHICAL DISTRIBUTION

Although the helium administrative offices and the production of helium are concentrated in Texas, New Mexico, and Kansas, the utilization of helium is worldwide, and the cost of transporting it to the point of consumption is usually considerably more than the cost of production.

Helium is available in more than 100 cities in 35 States, the District of Columbia, and the Territory of Hawaii. The principal points of distribution by private concerns are Lima, Ohio, Bound Brook, N. J., and Vernon, Calif., where the Air Reduction Sales Co. maintains facilities for receiving helium in tank cars and reshipping it in standard cylinders; and North Bergen, N. J., where the National Cylinder Gas Co. does likewise. Shipments to other non-Federal purchasers are made from the helium plants in standard cylinders. Helium may not be exported by a non-Federal purchaser without obtaining a license from the Secretary of State. The quantity exported is small.

TECHNOLOGY

EXPLORATION

The Bureau has never explored for helium by drilling. Instead, it collects and analyzes samples of gas from wells drilled by private operators; and, when a significant helium content is found, the Bureau studies the field concerned to determine its suitability as a source of supply for a helium plant.

The helium content of natural gas from different fields varies from a mere spectroscopic trace to more than 8 percent by volume. Usually, the gas of higher helium content is found in fields that lie over buried granite ridges, such as the deeply buried Amarillo Mountain of the Texas Panhandle and the Nemaha Ridge of Kansas, and in fields closely associated with igneous intrusions, such as the Rattlesnake field of San Juan County, N. Mex.

Although helium-bearing natural gases have been found in many fields, only a few are suitable for helium production in large volumes at reasonable cost.

DEVELOPMENT OF HELIUM-BEARING GAS FIELDS

The Bureau's rights in the Cliffside and Rattlesnake fields were acquired in the initial stages of their development, and further development was necessary and will continue to be necessary to utilize them as sources of supply for the Bureau's helium plants at Amarillo and Navajo (Shiprock), respectively.

The Bureau became interested in the Rattlesnake field when helium-bearing gas was found at a depth of about 7,000 feet in a well drilled for oil by the Continental Oil Co. The Bureau acquired that well and completed it as a gas producer with an open flow of about 34 million cu. ft. in 24 hours and a wellhead pressure of about 3,000 p. s. i. A second well, drilled for the Bureau by a contractor, was completed with an open flow of about 17 million cu. ft. per 24 hours and a wellhead pressure of about 3,000 p. s. i.

The fields that supply the Exell and Otis plants were developed by private companies.

EXTRACTION AND PURIFICATION OF HELIUM

The processes used for extracting and purifying helium have been developed by the Bureau of Mines. Helium is separated from other components of natural gas by cooling the gas to a temperature below the liquefaction point of its ordinary constituents but above its own liquefaction point.

Helium, processed to a purity of 98.2 to 99.5 percent, is then passed through activated charcoal cooled to the temperature of liquid nitrogen. The charcoal adsorbs virtually all remaining impurities and yields helium with a purity exceeding 99.99 percent. One purifier is reactivated by increasing its temperature while another is "on stream." Helium passes directly from the charcoal purifiers into high-pressure containers, in which it is shipped or stored.

STORAGE

Because of its extremely low temperature of liquefaction (−452° F.), the only practicable way to store helium at present is in the gaseous state. It may be stored in low-pressure gas holders, but it is more commonly stored in high-pressure containers to conserve space. High-pressure installations at the helium plants consist of long cylinders made of 2 or more double-length (40-foot) joints of 10-inch heavy-wall pipe. All joints are welded, and the cylinders are manifolded in banks for convenience in filling and discharging. Each plant has storage facilities that hold about 2,250,000 std. cu. ft. (14.7 p. s. i. a. and 70° F.) when filled to a pressure of 1,500 p. s. i. a. Some of the Naval air stations use seamless forged tubes instead of the welded cylinders.

TRANSPORTATION

Helium is shipped in special railway tank cars, automotive semitrailers, and standard compressed-gas cylinders. In the calendar year 1950, when the Amarillo and Exell plants produced a little more than 80 million cu. ft. of helium, the 2 plants handled 1,207 shipments consisting of 64,907 standard cylinders and 305 tank cars. Because Amarillo is a railroad and motor-freight center, shipments in standard cylinders usually are made from the Amarillo plant. The other plants have facilities to fill the cylinders, but they are not now equipped to wash and dry them internally, as is frequently required to prevent contamination of the helium.

A helium tank car is made up of 28 or 30 steel cylinders about 18 inches in external diameter, mounted lengthwise in a specially constructed frame. The capacity of each car is approximately 200,000 std. cu. ft. of helium, when filled to a pressure of 2,200 p. s. i. Each car weighs about 200,000 pounds when empty and about 2,000 pounds heavier when full. Freight charges are based on a minimum carload of 40,000 pounds.

Altogether, the Government owns 90 tank cars which are operated as a pool. No helium tank cars are owned by private industry or by the Bureau of Mines at present; however, the Bureau of Mines has an arrangement through which the cars may be used for deliveries to other Federal agencies and to non-Federal purchasers.

Automotive semitrailers used for transporting helium are specially built similar to the tank cars but smaller. All are owned by the Navy, but the Bureau has arrangements with the Navy under which they may be used to supply other purchasers. These semitrailers transport about 34,000 std. cu. ft. each.

Standard helium cylinders, complying with ICC–3A specifications, are filled to pressures up to 2,400 p. s. i. and have capacities up to about 235 std. cu. ft. of helium. These cylinders are shipped in conventional boxcars and motor trucks. Some purchasers of helium that take deliveries in standard-type cylinders supply their own cylinders; others receive the helium in cylinders supplied by the Bureau of Mines and pay the Bureau for their use.

The amount of helium that can be transported per pound of container by present methods is very small—about 1 or 2 cu. ft. of helium per pound of container. Consequently the cost of shipping helium is high and represents a disproportionate part of the overall cost of helium delivered to the point of use. The freight costs often are several times the cost of the helium at a helium plant. This makes helium transportation a fertile field for attempts at improvement.

UTILIZATION

Changes in helium utilization have had a marked influence on its production. In the early days when helium was produced at Fort Worth and used only as a lifting gas for airships, 92 percent purity was considered to be good. Better design and operation at the Amarillo plant increased the purity to about 98.2 percent, even before recent improvements were made in the purification cycle of the liquefaction process.

This purity was satisfactory for airships but inadequate for some of the newer uses, such as helium-shielded arc welding. Consequently, the Bureau has added charcoal-purification equipment at the plants to remove nitrogen and hydrogen impurities, which are deleterious in some welding and other metallurgical operations. In addition, it was necessary to clean and dry the plant piping and shipping containers to prevent contamination after the helium leaves the charcoal purifier. The purity of helium now produced by the Bureau and made available to Government and private users alike exceeds 99.995 percent.

USES OF HELIUM

Helium is a unique element with many extreme properties. At normal atmospheric pressure and 60° F., 1,000 cu. ft. of helium weighs only 10.54 pounds and provides a lift of more than 65 pounds in air—92.64 percent as much as hydrogen. Helium diffuses more rapidly, flows through a hole faster, conducts heat better, and transmits sound at higher velocity than any other gas except hydrogen. It conducts electricity better than any gas except neon. Helium has a lower solubility in water and other liquids, a lower refractive index, and a lower temperature of liquefaction than any other gas.

Helium is used as a lifting gas in airships and in observation, advertising, meteorological, and toy balloons. Helium-filled blimps escorted 89,000 surface craft in World War II without the loss of a single craft to enemy submarines. The Navy's "Operation Skyhook" has sent helium-filled balloons far into the stratosphere to obtain vital information on cosmic-ray activity and other conditions. Children enjoy safe play with balloons filled with nonflammable helium.

Almost as much helium is used in helium-shielded arc welding and other metallurgical processes as in airships and balloons. Using helium as an inert gaseous shield around an electrode, industry has found that it can arc-weld aluminum, copper, titanium, magnesium, stainless steel, and other alloys without using flux or extensive pretreatment. Helium also is used in degassing molten metals and in producing titanium metal.

Helium is used with oxygen in breathing atmospheres for asthma patients and persons suffering from other respiratory diseases. The helium-oxygen mixture is much lighter than air and will flow through restricted respiratory passages more readily. Consequently, more oxygen can be breathed with the same muscular effort. Helium-oxygen mixtures also are used by divers and caisson workers.

Helium is employed as a tracer element in determining the underground movement of injected gas in oil and gas fields and in detecting leaks in high-pressure and high-vacuum systems. Because of its low refractive index, it is used to fill the space between lenses in range finders and other optical instruments. Alone or mixed with other gases, it is used to obtain various colors in so-called "neon" lights. Helium is mixed with otherwise explosive anaesthetics to make them safe in operating rooms.

Helium is utilized in nuclear physics because it ionizes slowly when bombarded with electrons and has certain other properties. The extreme cold obtainable with liquid and solid helium has opened a broad field of research near the absolute zero of temperature. By studying the properties of the elements at such low temperatures, scientists can infer much about their basic structures and energies.

Helium is no longer a "one-purpose" gas but an important industrial and medical gas, in addition to being a vital military material; and yet, it remains a laboratory curiosity, for in the secrets of its simple atomic structure may be found the answers to many basic questions regarding the world in which we live.

RELATION TO OTHER COMMODITIES

Helium-bearing natural gas is the only known source from which helium can be extracted economically in enough volume to sustain a helium-production plant. Consequently, the production of helium is very closely related to the natural-gas industry. The location, design, and operation of the Bureau's plants, the cost of producing helium, the adequacy of helium reserves for the future—all are affected by natural-gas operations.

All helium-bearing natural gases serving the Bureau's plants were found by private companies in drilling for oil and fuel gas. The occurrence of helium as a minor constituent in the gas was of secondary importance to the companies, but it is the foundation upon which the entire helium industry is built.

The fastest growing use of helium is as an inert-gas shield in welding and other metallurgical processes. These applications require an inert gas that will not combine with molten metal to form impurities. Helium fills the requirements of a good shielding gas, although it is not alone in that respect, because argon also is suitable.

Both helium and argon are being used successfully as an inert-gas shield in welding and in the production of titanium and zirconium. They have different electrical and thermal characteristics that make each especially desirable under certain circumstances; but, in the absence of either one, the other usually can be used.

The two gases are sometimes mixed to provide a composite of their desirable properties, and considerable research is being conducted on the use of gaseous mixtures.

According to the Bureau of the Census, 58 million cu. ft. of argon was produced in the 1950 calendar year and 91 million cu. ft. in 1951. This compares with 81 million cu. ft. of helium in the 1950 calendar year and 109 million in 1951. No breakdown is available to show how much of either gas was used for welding, but probably 35 or 40 percent of the helium was used for that purpose.

It seems, therefore, that helium and argon are more complementary than competitive, although price undoubtedly is a factor in some instances. Helium is more effective than argon in some applications and argon better than helium in others; and for some operations, a mixture of the two gases is more satisfactory than either gas alone.

PRODUCTION

The Bureau of Mines produces helium primarily for use by Federal agencies, but the Helium Act (Public Law 411, 75th Congress), as amended, provides that helium not needed for the Government may be sold for medical, scientific, and commercial use. In recent years the demand has been about two-thirds Federal and one-third private industry and medical. Surveys have shown, however, that at least 65 percent of the helium sold to private customers is consumed on Government contracts. Consequently, about 90 percent of the helium is produced for the Government's benefit. Much of the remainder is utilized in producing defense-supporting items, such as farm tractors, automobiles, electronic devices, and refrigerating equipment and in hospitals. Much of the helium sold to private industry for use on Government contracts is as vital to the Government as some of the Government's direct uses.

Helium is not known to be produced by any other agency or private concern in the world.

Table 1 shows the helium production in the United States from 1921 through the 1954 calendar year.

PRICES

The price was relatively high at the old Fort Worth plant and for the first 3 months of operation at the Amarillo plant. In the first full year at Amarillo, however, the price dropped to $14.30 per 1,000 cu. ft., and it has been less than that to Federal agencies ever since.

In the fiscal year 1943 during World War II, the unit price to Federal agencies was $5.36, which is the lowest yet achieved. Federal agencies paid $12.50 per 1,000 cu. ft. for helium in the fiscal year 1952, including $3.00 per 1,000 cu. ft. for depreciation of Government properties and depletion of Government helium-bearing gas reserves. Depreciation and depletion have been taken into account in establishing the price charged for helium sold to non-Federal purchasers ever since such sales were begun under the act of September 1, 1937.

Federal agencies also are charged for services rendered them in connection with helium shipments. The most important service charges, which are designed to merely reimburse the Bureau, are 40 cents per cylinder for handling and filling standard cylinders and $1 per cylinder per year to cover the cost of maintaining Bureau of Mines cylinders used in the service of Federal agencies. The cylinder maintenance charge does not apply to cylinders furnished by the agencies themselves. Other charges are made for washing and drying cylinders, removing and resetting valves, stenciling and painting cylinders, and similar services when required by the agencies.

Regarding the sale of helium to non-Federal customers, the Helium Act provides

> that helium not needed for Government use may be produced and sold upon payment in advance in quan-

TABLE 1.—*Helium production in the United States, 1921–54*

Year	Active plants	Production (cu. ft.)
1921–January 1929 [1]	Fort Worth, Tex	46, 088, 787
1929 (April)–1942	Amarillo, Tex	164, 867, 140
1943	Amarillo and Exell, Tex	116, 307, 432
1944	Amarillo and Exell, Tex., Otis and Cunningham, Kans., and Navajo (Shiprock), N. Mex.	
1945	Amarillo and Exell, Tex., Otis and Cunningham, Kans	94, 733, 744
1946	do	58, 236, 385
1947	Amarillo and Exell, Tex	70, 297, 700
1948	Exell, Tex	63, 143, 513
1949	do	55, 165, 482
1950	do	81, 394, 416
1951	Amarillo and Exell, Tex., and Otis, Kans	112, 009, 180
1952	do	144, 556, 141
1953	Amarillo and Exell, Tex., Otis, Kans., and Navajo, N. Mex	161, 603, 800
1954	do	190, 928, 300
Total		[2] 1, 486, 265, 150

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[1] No helium was produced at Government helium plants in February or March 1929. The Fort Worth plant was shut down on January 10, 1929, and the Amarillo plant was not put into operation until April.

[2] Includes 87,509,400 cubic feet extracted at the Exell plant and injected into the Government-owned Cliffside gas field for conservation.

> tities and under regulations approved by the President, for medical, scientific, and commercial use * * * at reasonable prices based upon the cost of acquiring, developing, maintaining, and operating the Government properties, and the payment of interest at a rate of not less than 3½ per centum on capital hereafter expended for properties, facilities, and helium-bearing gas lands, as are used for such helium production.

The first regulations, approved by the President January 14, 1938, and revised March 10, 1938, provided that helium should be sold for scientific use at a unit price per 1,000 cu. ft. equal to 105 per centum of its actual cost to the Government and that the price of helium sold for commercial use should be 112 percent of 105 percent of the actual cost. The regulations provided further that the unit price of helium sold for medical use should be such as would, in the judgment of the Secretary of the Interior, permit the general use of helium for medical purposes.

The regulations were amended as of February 16, 1947, to provide, among other things, firm prices for helium. The specified unit prices were $11 for helium sold for medical use and $13 for helium sold for scientific and commercial use. The regulations also provided that an additional charge of $2 per 1,000 cu. ft. should be made whenever the helium was compressed into standard cylinders.

The regulations approved February 2, 1954, set the unit price of all helium sold for medical, scientific, and commercial use at $19 per 1,000 cu. ft. The additional charge of $2 per 1,000 cu. ft. for handling and filling standard cylinders still applies.

The regulations also specify charges for other services and for the use of helium containers supplied by the Bureau for the use of non-Federal customers.

The prices charged by compressed-gas distributors for the helium they purchase from the Bureau of Mines for resale vary considerably from place to place and company to company depending upon the efficiency of the company's operations, the distance from the helium plants and the resultant freight costs, the cost of supplying and handling the containers, and the profit the company wishes to make. The selling prices by one company in 1951, in several major cities in the United States, was as follows:

	Per 1,000 cu. ft.
Chicago, Ill	$57.00
Detroit, Mich	54.30
Boston, Mass	62.30
Portland, Oreg	95.60
Seattle, Wash	95.60
San Francisco, Calif	87.00

The difference between the prices charged by private distributors and that charged by the Government at a helium plant is due in large measure to the freight cost.

CONSERVATION

Helium is difficult to conserve because it usually is found as a constituent of natural gas that has good fuel value, and a greater and more rapid return can be obtained by producing and selling the gas for fuel than by reserving it for helium production. A number of fields that would have been excellent sources of gas for helium production have been depleted by the production of the gas for use as fuel.

As has been discussed previously, the Bureau of Mines has attempted to meet the problem of conservation by acquiring reserves of helium-bearing natural gas that can be held until needed, filling demands insofar as possible by recovering helium from natural gas that is produced for fuel markets, and injecting any excess helium that can be recovered from the fuel gas into the Government-controlled reserves. However, the transportation of helium by rail is costly, so excess helium can be produced and conserved only if a Government reserve suitable for injection is near enough to permit transportation of the helium there by pipeline. The Cliffside field, into which excess helium from the Exell plant is injected, is only about 20 miles from that plant. At the end of 1954 almost 90 million cu. ft. of helium had been conserved by injection into the Cliffside field.

The fields easiest to reserve for future helium production are those that produce gas not valuable as fuel, such as the Rattlesnake field in San Juan County, N. Mex. However, such reserves present economic problems, because the incombustible residue gas that remains after the helium is extracted will not aid in bearing the cost of acquiring, developing, and operating the field.

The Government has set aside lands of the public domain covering the Woodside structure in Emery County, Utah, and Harley dome in Grand County, Utah, as Helium Reserves 1 and 2, respectively. These deposits have not been drawn upon, and their extent is not known. Exploratory wells found gas with a helium content of 1.3 percent in Woodside and 7 percent in Harley dome.

RESEARCH

Helium research can be divided into three main categories:

(1) Research directed toward the assurance, now and in the future, of an adequate supply of helium of adequate purity at reasonable cost.

(2) Research aimed at determining fundamental data on helium.

(3) Research on specific applications of helium.

The Bureau of Mines is concerned largely with the first category, and most of its research is directed toward resources of helium-bearing natural gas, their conservation, and methods of producing, purifying, transporting, and storing helium.

The Bureau has conducted a survey to determine the helium contents of natural gases produced in the United States since the beginning of the Government helium program in 1917. Samples of gas are collected from wells drilled by oil and gas companies and analyzed for their helium content. Engineering studies are made of fields that appear favorable to determine their size and other characteristics.

The Bureau of Mines has also made studies to improve processes and methods of producing, purifying, handling, and analyzing helium. The present helium-extraction process was developed by the Bureau.

A current problem is the liquefaction of helium in quantity, so that it may be transported in liquid form to reduce costs and increase the speed of transportation. This problem involves many technical difficulties but offers promise. Helium is the most difficult of all gases to liquefy—it has a boiling point of −452° F., only 7½° F. above absolute zero. If practical low-cost methods of liquefying helium can be developed and suitable containers can be devised to transport it, the transportation of helium in large quantities by air will become feasible.

The Bureau also conducts research on the utilization of helium for specific purposes, such as welding and other metallurgical operations. A major feature of that research is to determine the quantities, purity, and delivery conditions needed in serving such uses. Other units of the Bureau have developed uses of helium in their research on other minerals. A good example is the use of helium as an inert shield in the production of titanium. The Bureau's research on helium utilization is conducted in close cooperation with interested Government agencies and private industry.

In general, fundamental research on helium is carried on by educational institutions, nonprofit research organizations, and some Government agencies, such as the Bureau of Standards, Atomic Energy Commission, and the Armed Forces. Much of this research is aimed at increasing the fundamental knowledge of helium and its isotopes, in the hope that such knowledge will lead to better understanding of the composition of matter. The unusual properties of helium at extremely low temperatures have received considerable attention.

Research on specific applications of helium is conducted by industrial concerns interested in developing processes and equipment that utilize helium, such as inert-gas-shielded arc welding, titanium production, leak detectors, radio tubes, range finders, and rockets; by hospitals and medical associations interested in the use of helium as anaesthetics, in the treatment of respiratory diseases, and for other purposes; and by the Armed Forces, Atomic Energy Commission, Public Health Service, National Advisory Committee for Aeronautics, and other Government agencies.

Private industry, educational institutions, hospitals, and many Government agencies share the research that will shape the future of the helium industry; however, insofar as the discovery, conservation, and production of helium are concerned, the Bureau of Mines carries almost alone the responsibility of research in those fields.

OUTLOOK

SIGNIFICANT TRENDS

There are encouraging indications that the usefulness of helium will be expanded greatly in the future. Probably the greatest increases in demand in the immediate future will result from the growth of present uses, but new uses undoubtedly will be found.

The present rapid rate of growth is caused primarily by the increased use of helium in welding. The Air Reduction Co., Inc., which has developed the Aircomatic inert-gas-shielded arc-welding machine, predicts that this machine, or adaptations of it, eventually will be used in every welding shop in the country—not only for welding copper, magnesium, aluminum, titanium, stainless steel, and other alloys but also on steel, wrought iron, and similar metals. At present the company is making the machines available as fast as it can manufacture them. Other companies are marketing similar machines under licensing arrangements.

Not all of these machines will use helium; some will use argon as the shielding medium, and others may use mixtures of helium and argon. The choice of gas, or of a mixture of gases, will depend upon performance in the various welding operations, and as helium or a helium-argon mixture serves better in many applications than argon alone, a rapid increase in the demand for helium for use in welding seems assured.

Some other uses of helium are sure to expand. Helium is used in producing titanium and zirconium, which are just now gaining a foothold as light, but strong, structural metals. As the titanium and zirconium industries grow, the demand for helium is expected to increase accordingly.

Helium is used by the Atomic Energy Commission for research and other purposes at all of its major installations. It is used by the military services as a fuel expellant for rockets and guided missiles. All of these applications are expected to expand.

As a lifting gas for airships, helium is unequaled and will continue to be utilized for that purpose. The importance of this use in the future will depend, however, on the future of Navy blimps and possible development of large dirigibles for passenger and freight services.

There appear to be ample prospects for expanded and new uses of helium to confirm the upward trend. For a long time the idea persisted that helium was rare and costly. Industry and the public generally are awakening to the fact that helium is neither scarce nor expensive. There has been no concerted effort to sell helium, yet its use continues to spread, each new user becoming a witness to its availability and reasonable cost.

PROBLEMS

The primary helium problem is to meet present and future demands and especially to fill the extreme demands of wartime without building up an unduly large investment in facilities that must stand idle much of the time. This requires reasonably accurate forecasting of the demands, adequate reserves of helium-bearing natural gas, facilities for producing helium, and trained staffs and crews to operate them to meet current requirements and provide a nucleus for expansion when needed.

The distribution of helium to the points of consumption presents additional problems. The containers that now transport helium as a compressed gas are heavy, holding only 1 or 2 cu. ft. of helium per pound of container. Consequently shipping expenses are high and represent a disproportionate part of the ultimate cost of the helium to the consumer. The Bureau's studies on helium liquefaction may lead to some relief in transportation costs.

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INDIUM

By

John D. Sargent[1]

INDIUM, a metal soft enough to be scratched with the thumbnail, imparts great hardness, strength, and corrosion resistance to other metals on which it may be plated or surface-diffused, or with which it may be alloyed. This lustrous, silver-white metal has been used mostly as bearing material in the aviation, automotive, and diesel fields where an expanding future seems assured.

Summary

The element indium was discovered in 1863 and isolated that same year. For over a half century scientists studied the metal and its compounds but found little practical use for the scarce and expensive substance. Commercial utilization of indium by American industry has been developed only recently.

Indium will plate readily on most metals, other than the ferrous group; when diffused into the metal on which it is plated, the resultant alloy is harder, offers greater corrosion resistance, and has a lower coefficient of friction than the unalloyed metal. These properties proved most valuable in the finishing of bearings, particularly for airplane engines in World War II. Alloys of indium have found increasing use in solders, brazing materials, dental alloys, and metallic seals. Other uses for the metal are as an internal lubricant for engine parts and in the manufacture of junction-type transistors (with the metal germanium) to replace vacuum tubes in electronic equipment.

The element is found widely distributed in the earth's crust, but in minute quantities. No indium minerals are known. Zinc ores and a few rare lead-tin-antimony sulfides have been found to contain up to 1 percent indium. The principal commercial sources of the metal are the residues and dusts produced in refining zinc and lead.

The published price of indium has dropped from $30 per troy ounce in 1942 to the prevailing price of $2.25. The United States is the major producer and consumer of indium and its alloys and compounds. Indium has also been produced in Canada, Peru, and Germany. Current production of indium exceeds demand for the metal. Increased future utilization is indicated in the plating field and in electronics, where consumption is rapidly expanding.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

HISTORY

In 1863, in the old city of Freiburg, in the foothills of the Saxon Erzgebirge, Prof. Ferdinand Reich and his assistant, Hieronymus Theodor Richter, noticed brilliant indigo-blue lines during spectroscopic examination of a specimen of sphalerite. Upon further testing, they were convinced that the blue lines were due to the presence of a new element, which they named "indium," because of the pronounced indigo-blue lines. The indium spectral lines are quite distinct from the two blue lines of cesium and the blue line of strontium. Indium blue can be recognized even without a spectroscope by merely heating a little indium chloride on a platinum wire in a bunsen flame.

Reich and Richter were also the first to isolate indium, which they accomplished in the same year that they discovered the metal. They reduced a sample of indium oxide with charcoal and obtained a small button that resembled tin.

During the following 60 years, various scientists studied the element and its compounds and alloys to define their properties and characteristics.

Indium first attracted attention in the United States about 1915, when a gallium-indium alloy was observed to be sweating out of lead exposed to sunlight; but it was not until 1924 that serious consideration of the metal was given by American industry, when the Oneida Community, Ltd., in Oneida, N. Y., studied its utilization in development of nontarnishable silver. Chemist William S. Murray was assigned to this project and, after a series of investigations with various metals, reached the conclusion that indium should have valuable properties in stabilizing nonferrous metals. The first step was to find an ore that contained economic quantities of indium. Sample ores of zinc, lead, silver, and gold were examined, but only traces of the element were detected.

The technologic staff at the Salt Lake City, Utah, station of the Bureau of Mines cooperated in the search for a commercial source of indium. This effort led to discovery of indium in a zinc-plant flue dust. Shippers to the plant were located and samples of sphalerite obtained. The richest source was the old Bebee property near Kingman, Ariz., in which 35,000 tons of indium-rich ore was blocked out. The complex ore, containing lead, zinc, iron, copper, silver, and gold, analyzed 1.93 ounces of indium per ton. In 1926 an experimental flotation plant was constructed that produced a silver-lead concentrate and a zinc concentrate; the indium was contained in the latter. The zinc concentrate was roasted and the soluble portion dissolved in sulfuric acid. From this solution, the indium was either plated out by metallic displacement or precipitated by neutralization of the acid. The impure indium was redissolved, purified, and again plated out (*2*).[2]

The Indium Corp. of America was formed in 1934 to carry on this early enterprise. From 1934 to 1937 intensive research was conducted, which led to improvement of previously developed processes and the patenting of new ones. By 1939 industry was investigating the advantages of using indium to stabilize nonferrous alloys and to develop and promote uses and sales of the metal.

GENERAL PROPERTIES

Indium (*3*) resembles tin and has a similar characteristic with tin in emitting a high-pitched "cry" when a rod of the pure metal is bent. In the electromotive series, it appears between iron and tin. Indium does not decompose water, even at boiling temperatures. It is stable in air but when heated it burns with a nonluminous blue-red flame, producing In_2O_3. The surface of indium remains bright up to its melting point, but at higher temperatures a film of oxide appears.

The toxic qualities of indium were studied by the Chrysler Corp. Disks of sterling silver, some of which were plated with indium, were implanted in rabbits for 156 days. The results indicated that, within the limits of the experiment, no local or general effects were discernible. No instance of dermatitis or toxic reaction has ever been reported by indium producers or consumers (*2*).

Metallic films of indium on various types of bearings, gears, bushings, and moving parts of machines greatly increase their life because of the corrosion-resistant and lubricating properties of this metal. Indium is soft and slippery and its viscosity changes very slightly over a wide range of temperature. The metal and its alloys with lead, tin, cadmium, and bismuth will adhere to or "wet" glass.

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

OCCURRENCE

Indium is widely distributed in the hydrosphere and lithosphere in minute quantities. Recent estimates give 1×10^{-5} for the abundance of indium in the earth. The metal does not occur native. Dark sphalerite, marmatite, and christophite frequently contain indium. The element has also been detected in small quantities in tin ores, iron carbonates (siderite), manganese, and tungsten ores. Gallium is often present with indium in zinc and tin ores.

The maximum recorded percentage of indium in sphalerite is 1.0 percent. This was noted in specimens from a Russian deposit and also in sphalerites from Pitkaranta, Finland. Other deposits rich in indium are reported in Middle Asia, where the metal content has been determined to be as high as 0.3 percent indium in dark sphalerites. In the United States, the richest recorded concentrations of indium in sphalerites were found in specimens from Georgetown, Leadville, and Nevada City, Colo.; these varied between 0.01 and 0.1 percent indium. The tin-bearing sulfosalts, cylindrite and franckeite, from Bolivia, have been reported to contain up to 1.0 percent indium. Many sulfides of copper, iron, lead, tin, cobalt, and bismuth contain small quantities.

Flue dusts from zinc smelters contain the largest commercial quantities of indium. Flue dust and Cottrell dust from some American plants contain over 1.0 percent indium. Other commercial production is derived from plant residues and dross, which are obtained as byproducts in the refining of zinc, lead, and cadmium. Indium reports with the lead when lead-zinc ores are smelted in vertical retorts.

PRODUCTION AND CONSUMPTION

The industrial use of indium in the United States began before World War II and has increased considerably since then, especially during periods of international strife.

United States shipments of indium (1)

[Troy ounces]

1941	7, 000	1947	13, 908
1942	21, 000	1948	12, 202
1943	59, 568	1949	54, 784
1944	82, 427	1950	125, 777
1945	57, 434	1951	153, 191
1946	9, 667		

Producers' end stocks increased considerably in 1950 and 1951, which resulted in decreased production in 1952; however, production and shipments of indium in 1953 more than doubled those of the previous high output of 1951. Principal United States producers of indium in 1953 were the American Smelting & Refining Co., Denver, Colo., and Perth Amboy, N. J., and the Anaconda Copper Mining Co., Great Falls, Mont. In earlier years production had been reported by the Sherwin Williams Co., Eagle-Picher Co., American Metal Co., American Zinc, Lead & Smelting Co., and Grasselli Chemical Co. (*1*).

Current annual domestic production of about 10 tons of indium could probably be increased to 100 tons if all discarded residues and dusts were processed for indium recovery. Cerro de Pasco Corp., Oroya, produces indium in Peru.

Peruvian production and exports of indium

[Troy ounces]

	Production	*Exports*
1945	1, 333	1, 332
1946	4, 080	5, 005
1947	18, 056	9, 804
1948	40, 348	14, 491
1949	------	20, 751

Canada production

[Troy ounces]

1941–42	437
1949	689
1950	4, 952
1951	582
1952	404

The Consolidated Mining & Smelting Co. of Canada, Ltd., Trail, B. C. (Cominco), is the only Canadian producer of indium. Although present production by this company is small, the potential output is about 35 tons per year. (*3*).

Production of indium has also been reported in the following countries:

Belgium (Société Belgochimie).
Japan (Fumkawa Mining Co.)
Germany (Unterharzer Berg-und Hüttenwerke, Oker).

Many lead, zinc, and tin producers do not extract indium or save plant residues and flue dusts, which contain the metal.

EXTRACTIVE METALLURGY

The recovery of indium from intermediate smelter products has been described by the Cerro de Pasco Copper Corp., as follows (*5*):

> Indium, which is generally found in association with zinc sulfide (sphalerite), enters the lead-smelter feed in the form of concentrates, ores, dusts, and residues that carry minor amounts of zinc. The percentage of indium in the lead-furnace charge is exceedingly low (a few thousandths of 1 percent), and the content of indium in the crude lead bullion is likewise low. However, the crude bullion contains about 0.25 percent tin which is removed in subsequent drossing operations. This also oxidizes the indium which becomes concentrated with the tin in the dross. The dross is then completely reduced and the resultant metal transferred to a melting kettle. Lead chloride is stirred into the molten metal and, since lead is below indium in the electromotive series, the metallic indium in the melt is converted to indium chloride and removed as a "slag."

Indium chloride becomes volatile above 400° C., but the melting point of lead chloride is 501° C. Consequently, the melting point of the chloride slag must be lower to prevent loss of indium by volatilization. This is accomplished by adding sufficient zinc chloride to yield a slag mixture having a melting point of approximately 300° C. The zinc chloride-lead chloride slag is a better solvent for indium than straight lead chloride. The zinc chloride does not react chemically; its effect is physical. The crude slag, containing chlorides of indium, lead, zinc, and tin, is wet-ground with additions of hydrochloric and sulfuric acid. (The addition of sulfuric acid to the pulp is primarily to prevent excessive dissolution of lead. The pulp is filtered and washed, yielding a filtrate containing about 25 grams of indium per liter. This solution is purified by allowing it to stand in contact with metallic indium which causes precipitation of tin, lead, and any other metals lying below indium in the electromotive series.

As zinc has a higher electrode potential than indium, the indium is recovered from the purified filtrate by deposition on zinc rods immersed in the solution. The precipitated indium, in the form of a metallic sponge, is then washed, briquetted, melted under paraffin, and cast into bars. The cast indium has a purity of 99.99 percent.

The recovery of indium from the Oker plant in Germany, which smelts lead-zinc ores from the Rommelsberg mining district, is described in a report from the U. S. Office of Military Government of Germany (*4*):

> When the lead is cupeled, all the indium is found in the first 15 percent of litharge (dross) which is raked out to be ground and then leached in sulfuric acid. Indium is then precipitated as a sponge from the purified solution by zinc and melted under sodium cyanide. The crude metal, containing 1.5 to 2.5 percent zinc and 0.5 percent lead, is refined by blowing steam through the molten metal to oxidize the zinc. The melt is cooled, granulated, and redissolved in acid. The indium is plated out of the solution using platinum anodes and indium foil cathodes. This refined metal is 99.99 percent pure.

USES

Alloys of indium have many promising uses in industry. The addition of small quantities of indium has the general effect of hardening and strengthening the metal with which it is alloyed and of increasing tarnish and corrosion resistance.

Gold-indium alloys, when used in denture castings, are smooth, dense, lustrous, and highly resistant to discoloration. The addition of indium to dental-gold solders provides increased fluidity, improves tensile strength, and gives greater bonding qualities and exceptional durability (*2*).

Lead-indium alloys have been utilized in bearings. The addition of 1 percent of indium to lead will double the hardness of the lead and increase the tensile strength from 1,600 to 3,000 pounds per square inch. An alloy of lead and 8 percent indium has a Brinell hardness of 7.2 (*2*).

The value of indium-treated silver-lead bearings was first recognized by the aviation industry. The unusual conditions imposed on bearings in aviation engines demand that they give long service without failing. Because of the relatively high oil temperatures and the generation of acid in the oil and because of heavy loads and the necessity of high wettability, indium is of prime importance in the silver-lead-indium bearings used in these engines. Each of the three components of this bearing has a service to perform. Silver has the internal properties that resist failure due to fatigue. Externally, silver lacks the quality of "oiliness" needed in a good bearing surface. To fill that requirement, a thin layer of lead is applied to the silver surface. Unfortunately, lead is soluble in the organic acids present or formed in lubricating oils. To offset this difficulty, a thin layer of indium is deposited on and diffused into the layer of lead. This addition of indium to the bearing surface accomplishes three things:

1. It increases the strength of the bearing material into which it is impregnated.
2. It prevents corrosion of the bearing surface without impairing the fatigue resistance or other bearing properties.
3. It permits the bearing surface to retain its oil film more completely by increasing its wettability.

Copper-lead, cadmium-silver, and cadmium-silver-copper bearings treated with 0.2 percent to 0.5 percent indium have been tested by automobile companies under high speeds and bearing temperatures, with excellent results in long life and acid resistance. It has been estimated that the cost of indium metal per connecting rod bearing would be 1 cent ($0.01).

The addition of indium to beryllium-copper alloys increases hardness and tensile strength. It also lowers the melting and heat-treatment temperatures and increases the fluidity of the alloy.

Solders are commonly composed of lead and tin, which have little resistance to alkaline attack. The addition of indium reduces the penetration of alkaline solution into the resultant alloy. A solder of 25 percent indium, 37.5 percent lead, and 37.5 percent tin is strong and highly resistant to alkaline solutions; it has a lower melting range (274° F. solidus) than the lead-tin solders (361° F. solidus). A 50-percent lead and 50-percent indium solder has a solidus temperature of 360° F. (same as lead-tin solder). For higher operating temperatures, a mixture of 25 percent indium and 75 percent lead can be used, as this has a solidus temperature of 446° F. (*2*).

Indium solders have been produced to meet the demand for an alloy with a melting point between lead-tin solders and brazing alloys. One of these has a melting point of 600° F., with a high degree of ductility and a holding strength of 3,500 p. s. i. and a tensile strength of 5,200 p. s. i.

Alloys of indium with copper, silver, and other elements are used as brazing alloys with melting temperatures from 795° to 965° F.

Many indium compounds have been made and studied. These include bromides, chlorides, fluorides, hydroxides, iodides nitrates, oxides, sulfates, and sulfides. Indium has a pronounced tendency to form basic salts that are only sparingly soluble in water. The basic chloride, however, is soluble in boiling water. In general, indium salts behave chemically like corresponding aluminum salts.

Indium plates readily, without any critical temperature or current density. The receptacle for the plating solution should be pitch- or rubber-lined, or it may be lined with glass or enamel (*2*). It is essential to keep the bath free of contaminating metals and materials. Insoluble anodes, such as carbon or platinum, are used, and a current density high enough to give gassing at the cathode is employed. Indium "as plated" is very soft; after diffusion it will be found to have "hardened" or actually to have alloyed with the base metal. The time of diffusion employed with success and without danger of weakening the bond on steel-backed pieces is 2 hours at 340° F. The metals onto which indium is most frequently plated and diffused are lead, cadmium, zinc, tin, gold, silver, and copper. The amount of indium used is very small and in most instances does not exceed 5 milligrams per square inch. The thickness of indium will range from a few ten-thousandths of an inch to 1 or 2 thousandths, depending on surface requirements (*2*).

Zinc-indium has proved satisfactory for coating steel blades of airplane propellers, because it adheres firmly to the steel without chipping or peeling and resists salt spray and wear. Articles such as jewelry cases, bracelets, compacts, and clock faces have been finished with indium to produce an attractive silver-white surface, highly resistant to tarnish and corrosion. The quantity of indium used is very small, probably not over a few cents worth for most articles. Unusual color effects can be obtained in jewelry by means of indium. For example, when gold and indium are plated alternately on silver and then exposed to heat, the article becomes sky-blue, while interdiffusion of layers of silver, indium, and palladium yields a fine, rose-pink alloy. An alloy of 7.5 percent indium with silver has high tarnish resistance and should find increased use in silverware and jewelry.

Indium diffused directly into the surface of harder metals is being used in wire-drawing dies because of a reported 50-percent increase of die life and faster operation in the drawing. Some bare-steel gear teeth refuse to accept lubrication; however, such gears finished with 5 micro-inches of indium accept lubricating oils and eliminate the tendency to stick.

Indium in the form of a proprietary alloy is marketed by Cerro de Pasco Corp. under the name of Cerroseal 35. This contains about equal parts of tin and indium. It is capable of wetting glass and of making glass-to-glass or glass-to-metal seals. Alloys of indium (5 percent or more) with lead, cadmium, or bismuth can also be used to join nonmetallic materials; however, the vacuum-tightness of seals made with an indium-tin eutectic alloy is considerably better.

The use of graphite and indium for lubrication and wear reduction in moving parts of internal-combustion engines, in dies and molds, in metal-disk clutches, and brakes has been reported.

An increasingly important use of indium is with the metal germanium in making junction-type transistors. These electronic devices are expected to replace vacuum tubes in many instances. Only 1 ounce of indium is needed in making 1,000 or more transistors.

Indium is used for motion-picture screens, cathode oscillographs, and mirrors because of its property of reflecting all colors of the spectrum, and indium oxides, sulfides, and other indium salts are used in the electrical industry because of the fluorescence and photoconductivity of these compounds.

The addition of indium sesquioxide to a glass mixture containing sulfur colors the glass yellow in proportion to the indium sesquioxide used. This color appears to be developed by the interaction of indium sesquioxide as the glass is being made, because adding indium sulfide to the glass-forming mixture does not produce the same results. The color is progressive from light canary to dark tangerine-orange. Indium salts are used in treating sleeping sickness.

Low-melting alloys containing bismuth, lead, tin, cadmium and 18 to 26 percent indium have been marketed under the trade name Cerrolow. This metal can, in the fluid state, be brought in contact with the body without causing discomfort. Casts can be rendered permanent by electroplating with silver or copper and melting out the fusible alloy. Cloth or blanket material impregnated with the alloy can be molded in any form when heated. This material sets rigidly upon cooling and can be made pliable again for removal by the application of hot water bottles. The indium alloy with the lowest melting point is a mixture of bismuth, lead, tin, cadmium, indium, mercury, and gallium, which has a melting point of 95° F.

PRICES

The nominal published price of indium in 1954 was $2.25 per troy ounce for electrolytic grade, 99.9 percent. In 1942 the metal was priced at $30 per troy ounce. This was reduced to $15 in 1943, $10 in 1944, $7.50 in 1945, and the current price of $2.25 in 1946. (*1*)

OUTLOOK

Indium consumption probably will increase considerably in the next few years. If indium demand exceeds production, ample stocks are available that can be drawn upon for many years to come.

PROBLEMS

The principal indium problem is to find a market large enough to consume the 100 tons a year of recoverable indium that is contained in ores being mined for other metal values. Much of this potentially valuable natural resource is being wasted.

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IODINE

By

Joseph C. Arundale[1]

IODINE is vital to human and animal life. Most of us have applied "tincture of iodine" to cuts and scratches, most of us use "iodized salt" to prevent goiter, but few realize that iodine has a long list of varied additional uses in industry, agriculture, and medicine.

Summary

Iodine is one of the halogen group of elements and was discovered in seaweed nearly 150 years ago. In 1868 shipments of crude iodine were made from Chile, where it occurs as the mineral lautarite (calcium iodate) associated with nitrate deposits. Later iodine was recovered from waste oil-well brines. These latter two sources now contribute the greater part of the world's supply. In 1953 the United States consumed nearly 1,200,000 pounds of crude iodine, valued at about $2 million, in the manufacture of iodine compounds as well as an undetermined quantity as a catalyst and in the manufacture of products that are not essentially iodine derivatives.

The two domestic producers furnish a substantial part of United States requirements. The remainder is imported from Chile and Japan. Fields of use include medicine, metallurgy, photographic film, photoengraving, human and animal nutrition, rubber, dyes, analytical reagents, synthetic organics, and many others, including sanitation of dairy equipment and food utensils, and drinking-water purification. Although future requirements cannot be predicted accurately, an overall increase in consumption is anticipated.

The problems of the industry include reduction in costs, appraisal of domestic reserves, development of new uses, evaluation of market potential, and the need for more complete knowledge of physical and chemical properties, and economic factors.

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[1] Assistant chief, Branch of Construction and Chemical Materials, Bureau of Mines.

BACKGROUND

Iodine was discovered early in the 19th century in France by Bernard Courtois as a result of his work in recovering potash for use in gunpowder during the Napoleonic wars. The first iodine was recovered from the ash of burned seaweed. Many famous contemporary chemists, including Gay-Lussac and Davy, contributed to the investigation of the properties of iodine. Its importance to chemical science was recognized early, and by the middle of the 19th century substantial quantities were being produced both in France and Great Britain. Kelp was the only commercial source of iodine for many years. Iodine was then discovered in the nitrate deposits of Chile, and shipments were first made in 1868. Later (about 1929), natural brines became an important source, and now iodine recovered from waste oil-well brines in California is one of the principal sources of iodine for the United States.

Iodine is one of the halogen group of elements, which also includes chlorine, bromine, and fluorine. Although iodine is relatively rare in the earth's crust (approximately the 47th most abundant element according to the Federal Geological Survey), it is widely distributed in minute quantities in plants and animals, and in mineral form.

In its mineral form, iodine occurs in about a dozen minerals as an essential constituent, the most important being lautarite, $Ca(IO_3)_2$, the iodine mineral that occurs in the nitrate of soda or so-called "caliche" deposits of Chile. These deposits are in the Provinces of Antofagasta and Tarapaca on the eastern slopes of the coast range in a barren and arid desert. Run-of-mine ore contains about 0.04 percent iodine. Caliche has been one of the world's most important sources of iodine for many years.

OCCURRENCES

In several countries iodine occurs in varying proportions in brines or mineralized waters. One such occurrence is in the area of Long Beach, Calif. Certain oil-well brines in this district contain iodine in concentrations up to 140 parts per million, with the average at about half that concentration. Oil-field brines near Shreveport, La., also have a relatively high iodine content. Similar brines and mineralized waters containing iodine occur in the U.S.S.R., Japan, Italy, Java, Germany, France, and England.

Although iodine occurs only in minute traces in seawater, certain seaweeds have the ability to extract it selectively and this, the first source of iodine, was until recently an important source. The red wracks or "drift kelp" that grow so as always to be submerged by the tide have the greatest concentration of iodine. Although there is reported to be a seasonal and regional variation, the iodine content of this variety of seaweed usually does not exceed about 0.03 percent by weight of the dry weed.

Other types of occurrences seem to have little commercial importance at present. In Germany certain coals were found to have up to about 11 milligrams of iodine per kilogram of coal. Certain soils (for example, the Coastal Plain of the southeastern United States) contain iodine in concentrations up to several parts per million.

RESERVES

Iodine reserves in domestic brines have not been adequately measured but are believed to be large. It is very difficult to estimate the iodine reserves in brines and saline deposits. Iodine is continually being selectively extracted from seawater by seaweeds.

TECHNOLOGY

Iodine is recovered by several methods. In Chile the caliche ore is leached to recover sodium nitrate. The iodate solution becomes increasingly concentrated as the nitrate is removed, and the leach solution is recycled. At intervals the solution is drawn off and treated with sodium bisulfite to precipitate free iodine, which is filtered and sublimed.

In California, two methods are employed in extracting iodine from the brine. In one the brine is acidified. The iodine is liberated by chlorination and blown out by air, and the vapors are collected with a sulfuric-hydriodic acid solution to which sulfur dioxide is added. This reduces the iodine to hydriodic acid and oxidizes the sulfurous acid to sulfuric acid. Iodine is recovered from this mixture by further chlorination and filtration and is purified by melting under sulfuric acid.

In the other process silver nitrate is added to the brine, forming silver iodide, which is coagulated with ferric chloride and filtered. The iron compounds are dissolved with sulfuric acid, and ferrous iodide is obtained from the residue by treatment with metallic iron and water. The ferrous iodide is converted to potassium iodide, the salable product, by reacting with potassium carbonate.

Iodine has been recovered from seaweed by several methods; in one process, the kelp is leached and the filtered liquor partly evaporated to yield sodium sulfate and potassium chloride. After removal of these salts, the mother liquor is treated with sulfuric acid and manganese dioxide to release iodine.

In another process the iodine is precipitated as a cuprous iodide with copper sulfate. Other processes involve leaching a portion of the iodine by washing the fresh weed with water and inoculation with bacteria to cause fermentation of a slurry of weed and water.

PRODUCTION

Iodine is produced in the United States by two companies—Dow Chemical Co. and Deepwater Chemical Co.—both of which recover iodine from waste oil-well brines in California. Together they supply a substantial part of domestic requirements. There seems to be no reason why their rate of production could not be increased substantially if needed.

In Chile the greater proportion of nitrate is produced (see table 1) by two companies—the Anglo-Lautaro Nitrate Corp. and the Cia Salitrera de Tarapaca y Antofagasta. Together these 2 firms produce about 90 percent of the Chilean iodine output. The remaining 10 percent is in the hands of small independent producers. Anglo-Lautaro represents about 60 percent of the industry and is owned principally by American and British capital. Cia Salitrera de Tarapaca y Antofagasta represents principally British, German, and Chilean capital. Chilean producers could recover substantially greater quantities of iodine if markets were available.

All Japanese iodine now is recovered from brine, and that country has a large exportable surplus.

Table 1.—*Production of crude iodine in Chile, 1940–51* [1]

[Pounds]

Year	Pounds	Year	Pounds
1940	2, 799, 000	1946	1, 366, 000
1941	3, 276, 000	1947	2, 784, 000
1942	1, 893, 000	1948	1, 881, 000
1943	1, 807, 000	1949	1, 983, 000
1944	2, 929, 000	1950	1, 197, 000
1945	1, 536, 000	1951	2, 863, 000

[1] From data compiled for the Bureau of Mines by U. S. Foreign Service officers.

FOREIGN TRADE [2]

Table 2.—*Crude iodine imported for consumption in the United States, by countries of origin, 1941–53*

[U. S. Department of Commerce]

Year	Chile		Japan		Total	
	Pounds	Value	Pounds	Value	Pounds	Value
1941	1, 010, 039	$1, 121, 513			1, 010, 039	$1, 121, 513
1942	951, 243	1, 051, 432			951, 243	1, 051, 432
1943	2, 744, 930	3, 041, 609			2, 744, 930	3, 041, 609
1944	1, 204, 303	1, 321, 274			1, 204, 303	1, 321, 274
1945	220, 526	232, 070			220, 526	232, 070
1946	886, 578	976, 190			886, 578	976, 190
1947	2, 260, 506	2, 756, 888			2, 260, 506	2, 756, 888
1948	541, 439	786, 850	50, 697	$60, 902	592, 136	847, 752
1949	382, 344	577, 810	107, 655	141, 948	489, 999	719, 758
1950	582, 562	854, 236	142, 296	201, 710	724, 858	1, 055, 946
1951	667, 426	1, 036, 414	184, 681	283, 914	852, 107	1, 320, 328
1952	471, 077	858, 092	320, 131	504, 817	791, 208	1, 362, 909
1953	681, 484	1, 197, 379	276, 154	408, 645	957, 638	1, 606, 024

The erratic nature of imports (see table 2) does not imply a corresponding variance in actual rate of consumption. Large stocks of iodine normally are held in Chile and New York City and are replenished at irregular intervals as they are depleted by withdrawal for domestic consumption and export requirements.

USES

The so-called crude iodine of commerce usually is about 99.5 percent pure. However, little is consumed in this form. Most is either resublimed to greater purity or converted to iodine compounds. Potassium iodide is the principal compound produced, but scores of other compounds are produced in minor quantities. In 1953, 25 firms consumed iodine in manufacturing a long list of iodine compounds (see table 3).

Iodine and its compounds have a multitude of direct and indirect applications. Perhaps the best known is the popular household antiseptic, "tincture of iodine," which consists of resublimed iodine and sodium iodide dissolved in diluted alcohol. Another familiar use is in "iodized" salt for human and animal nutrition. Iodine has a systemic action in promoting the production of thyroxin in the body and thereby preventing goiter. Such salt contains about 1 part of potassium iodide to 10,000 parts of salt. Potassium iodide, cuprous iodide, or potassium iodate is sometimes added to salt and feed products for animal consumption. Iodine is a constituent of X-ray contrast mediums used as diagnostic aids in examinations of various organs and tissues in the human body. Many other iodine compounds, iodized fats, and fatty acids are used in medicine and pharmacy.

One of the major industrial processes in which iodine and its compounds are employed is wet-plate photoengraving. Photographic

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[2] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

TABLE 3.—*Crude iodine consumed in the United States in 1951–53*

Compound manufactured	1951			1952			1953		
	Number of plants	Crude iodine consumed		Number of plants	Crude iodine consumed		Number of plants	Crude iodine consumed	
		Pounds	Percent of total		Pounds	Percent of total		Pounds	Percent of total
Resublimed iodine	6	137, 918	11	5	78, 222	7	6	149, 405	13
Potassium iodide	9	787, 936	64	10	768, 554	65	10	796, 953	68
Sodium iodide	5	114, 307	9	6	64, 332	5	6	55, 791	5
Other inorganic compounds	8	45, 198	4	8	29, 785	3	8	37, 012	3
Organic compounds	16	152, 563	12	13	232, 981	20	14	131, 261	11
Total	[1] 25	1, 237, 922	100	[1] 25	1, 173, 874	100	[1] 25	1, 170, 422	100

[1] A plant producing over 1 product is counted but once in arriving at total.

film emulsions are large consumers. Iodine has many applications in metallurgy, sanitation, water disinfection, rubber, dyes, analytical reagents, catalysts, etc.

PRICES

Before about 1931 the price of iodine was fixed at about $4 a pound, New York. With development of the American industry, prices were reduced to a low of about 80 cents per pound in the late 1930's. The present price of crude iodine is about $1.15 a pound.

Because of the relatively high price of iodine and its compounds, the other halogens are sometimes substituted where feasible. However, there are many essential uses, and it has been acquired for the National Stockpile.

RESEARCH

Evidence of the enormous amount of research that has been done on various aspects of iodine may be found in the comprehensive bibliographies on the subject which appear in the booklet, "Iodine: Its Properties and Technical Applications" and "Iodine Abstracts and Reviews," published by the Chilean Iodine Educational Bureau, Inc., 120 Broadway, New York, N. Y.

OUTLOOK

It is extremely difficult to forecast future consumption of iodine and its compounds. The many and varied uses are constantly being added to and expanded while substitutes are being found and iodine consumption in some uses has been reduced.

On the plus side, increased use in livestock feeding, sanitation, metallurgy, and fertilizers is predicted by the industry. New processes and products in the field of synthetic organic chemistry promise to increase the use of iodine and its compounds as catalytic agents, as process reagents, and in physiologically active materials.

In inorganic chemistry many new uses have been suggested. In metallurgy much study and research have been done on the utilization of iodine in preparing high-purity metals and alloys. Zirconium and titanium metals are being produced by the "iodide process" on a small scale. Although the cost of the present process deters its large-scale commercial application, considerable research is being conducted in an effort to develop a low-cost continuous process. Other potential applications in metallurgy are in electroplating, casting, production of alkali metals, and powder metallurgy. In medicine recent developments are the use of iodine compounds in conjunction with streptomycin in treating tuberculosis and of radioactive iodine as a diagnostic tool. Other possibilities are in dry cells and lubricants.

Consumption for some purposes may decline. For example, the latest edition of the U. S. Pharmacopoeia describes Iodine Tincture as consisting of 1.8–2.2 grams resublimed iodine and 2.1–2.6 grams sodium iodide dissolved in 100 cc. diluted alcohol. However, several previous editions, through USP XII, specified 7 percent resublimed iodine and 5 percent potassium iodide. Furthermore, the use of iodine as a local antiseptic appears to have declined somewhat in favor of various competitive proprietary antiseptics. The use of stripping film has been displacing that of the wet plate in photoengraving. The United States Government Printing Office has discontinued the use of wet-plate processing.

Indications are that overall consumption of iodine and its compounds will increase, but there seems to be no adequate basis for estimating future requirements.

It is interesting to note that imports of iodine from Japan have increased very rapidly in recent years. Iodine-bearing brines occur throughout the world, and other countries may become important producers. Brines similar to those in California occur at other localities in the United States, and others may be discovered as a result of the greatly expanded oil-well drilling program.

PROBLEMS

Although iodine reserves apparently are adequate for many years, a comprehensive appraisal of iodine reserves has not been made. For long-term guidance of both management and Government agencies concerned with assuring supplies of this commodity, more information on reserves is needed. One of the problems is to find new uses. More extensive knowledge of the physical and chemical properties of iodine and its numerous compounds would be a valuable aid in this search for new uses. Research of this type has been carried on intensively for many years but it is far from completed.

Costs are a problem. The relatively high cost of iodine deters its wider application, and the other halogens are sometimes substituted. There is also lack of detailed information on the quantities consumed in individual end uses and other statistical and economic data.

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IRON

By

W. Holliday [1]

IRON ORE is the basic metallic raw material of modern civilization. Without it, modern farming, manufacturing, construction, armament, and transportation could not have been developed. The high standard of living that the United States enjoys and the present position of this Nation in world affairs have been, in large measure, based upon easily accessible iron ore and coking coal, raw materials needed for iron and steel production.

Summary

The rise in domestic iron-ore production to 118 million gross tons in 1953 parallels improvement in living standards and increasing responsibility in world affairs. The degree of the Nation's good fortune is shown by the fact that the United States, with only 6 percent of the world's population, currently is producing and enjoying the benefits of more than one-third of the world's output of steel.

Iron-bearing minerals are widespread, but because of economic limitations many small or low grade deposits have not been utilized.

In the United States about 80 percent of past iron-ore production has come from the Lake Superior region; and Alabama, New York, Utah, and Texas have contributed most of the remainder; however, because of depletion of high-grade domestic iron ores, there is growing dependence on foreign ores and beneficiated domestic ores. Foreign deposits expected to supply the largest tonnages are in Canada and in Venezuela. Of the beneficiated domestic ores, taconite has perhaps received most attention. Taconite is a very hard, siliceous rock which, under favorable conditions, contains 20 to 30 percent of iron. Other low-grade materials, such as the ferruginous sandstones of Alabama, are also being used or being studied as possible future sources. Beneficiation varies from simple operations like screening and washing to very complex operations involving such steps as fine grinding, roasting, magnetic separation, or flotation.

Mining of iron ore is mainly a problem of handling and transportation. Domestic open-pit operations in 1953 produced 124 million tons of crude ore in addition to removal and disposal of overburden. Underground mining utilizes all known methods but leans toward large-tonnage, low-cost operations. In the Great Lakes region underground mines operate the year round and stockpile in the winter, whereas open-pit mines are seasonal in operation.

The metallurgy of iron consists essentially of the passage of ore, coke, and limestone through the blast furnace. As they progress downward carbon from the coke combines with oxygen from the ore, freeing the iron, which is tapped from the furnace hearth as a liquid.

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[1] Commodity-industry analyst, Bureau of Mines.

Present indications are that as much as 37 percent of the iron-ore supply will be imported by 1975. Of the 37 percent, Canada is expected to furnish about 21 percent, the remaining 16 percent coming from oversea sources. Venezuela is expected to send about 11 percent and Liberia and various other smaller sources only 5 percent. In the above estimates, concentrates from Lake Superior taconites were taken as 30 million gross tons, whereas announced plans of several major producers now place the ultimate projected annual capacity at 43 million gross tons. The two figures agree sufficiently, since in any case a forcast 20 years in the future is conjectural.

Although increasing dependence on foreign sources is, without doubt, a disturbing prospect, importation of iron ore should not necessarily be discouraged. On the contrary, the United States can retard the depletion of domestic reserves by supplementing iron-ore supplies with imported ore.

Domestic iron-ore production over the next 20 years is expected to remain substantially as it was in 1953. To maintain an output of 115 million tons will require continual improvement in plant and methods, and domestic producers are today conducting research in mining and beneficiation. Imports can be expected to supply increases in iron-ore requirements.

BACKGROUND

The vastness of the United States iron and steel industry defies ready comprehension, especially when accessory materials and services are considered. The direct raw materials add up to impressive totals. In 1953 the iron and steel industry produced 75.9 million net tons of pig iron, most of which it used in producing 111.6 million tons of crude steel. Among the materials consumed (*2*)[2] was 127 million tons of iron ore, 71 million tons of coke (100 million tons of coal for all purposes), 40 million tons of limestone and other fluxes, 2.4 billion gallons of fuel oil, 316 million gallons of tar and pitch, 223 billion cubic feet of natural gas, 16.5 million gallons of liquefied petroleum gas, and 40.7 billion kilowatt-hours of electric energy. The industrial organization necessary to supply these materials involves virtually all of the Nation's economy.

Iron was produced in the American Colonies (1645 to 1700) from a number of small charcoal furnaces distributed from Alabama to New England. By 1800 most of the States had ironmaking facilities, using various local iron ores. Later, similar operations were initiated in the Midwest and other newly devloped areas. The hot-blast unit, forerunner of the modern blast furnace, was first tested in 1834 at Oxford, N. J. Blast furnaces at first remained small; in fact, the largest recorded capacity in the mid-19th century was only 150 tons of pig iron a week.

Between 1870 and 1900 the industry grew rapidly. The use of coke from Pennsylvania, West Virginia and Alabama and ore from the Lake Superior district and Alabama expanded to meet the needs of a growing population and increasing industrialization. In 1897 the United States assumed world leadership in pig-iron production.

As the small deposits of high-grade ores first used in localized iron-smelting operations became depleted, as wood for charcoal became less plentiful, and as the use of coal replaced charcoal, the industry tended to centralize near the supplies of coking coal in the Pittsburgh and Birmingham areas. Thereafter, blast furnaces increased in size and complexity. Capital requirements increased until it became difficult for new producers to enter the industry. Producing companies, their locations, and annual capacities are shown in table 1. As of January 1, 1954, there were 34 domestic producers, with a total annual capacity of 82,001,390 tons of pig iron and ferroalloy output (*3*).

Although a major part of the ore and other raw materials for ironmaking are mined by the iron producers, there are many small and a few large independent mining companies. Iron ore distribution and blast furnace output in the United States are shown in figure 1. Most of the pig-iron production capacity is contiguous to domestic coking-coal deposits.

IRON-BEARING MINERALS

Recently, the increasing demand for beneficiated material has occasioned much new study of the properties of minerals and has brought renewed interest to the problem of classifying iron-ore deposits. Important iron minerals are hematite, magnetite, turgite, goethite, limonite, xanthosiderite, siderite, and fayalite. By-product iron ore is obtained from ilmenite, pyrite, and pyrrhotite.

The difficulty of classifying mineral deposits can be seen in the fact that in one district alone "about a hundred" iron-bearing minerals have been identified. These occur as mixtures in all proportions; they are also found in association with a variety of foreign materials that may be either deleterious or beneficial. Common impurities are moisture, silica, alumina, lime, magnesia, sulfur, phosphorus, manganese, and titanium.

IRON-ORE RESOURCES OF THE UNITED STATES

Even after 100 years of iron mining in the Lake Superior district and numerous geologic and engineering studies by reputable scientists, no one can state factually the tonnage of remaining ore. Substantial agreement however, is apparent among the various groups that have estimated domestic reserves. Table 2 is a compiliation of estimates showing direct-shipping and easily beneficiated ores of the United States.

Circumstantial evidence confirms the belief that the principal producers of iron and steel

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

TABLE 1.—*Producing companies, locations, and annual capacities of blast furnaces*

Plant location and operating company	1952		1954	
	Number of stacks	Annual capacity (net tons)	Number of stacks	Annual capacity (net tons)
Alabama (Southern district)				
Birmingham:				
Republic Steel Corp	2	456,000	2	402,000
Sloss-Sheffield Steel & Iron Co	2	281,230	----	----
U. S. Pipe & Foundry Co	----	----	2	281,230
Ensley: Tennessee Coal & Iron Division	6	[3] 1,481,800	6	[4] 1,844,300
Fairfield: Tennessee Coal & Iron Division	3	1,015,700	3	1,218,700
Gadsden: Republic Steel Corp	2	471,000	2	525,000
North Birmingham:				
Sloss Sheffield Steel & Iron Co	2	210,480	----	----
U. S. Pipe & Foundry Co	----	----	2	210,480
Woodward: Woodward Iron Co	4	[3] 772,630	4	772,630
Total	21	[3] 4,688,840	21	[4] 5,254,340
California (Western district)				
Fontana: Kaiser Steel Corp	2	876,000	3	1,314,000
Colorado (Western district)				
Pueblo: Colorado Fuel & Iron Corp	4	918,000	4	882,560
Illinois (Chicago district)				
Chicago:				
Interlake Iron Corp	2	448,020	2	586,620
Granite City Steel Co	2	450,000	2	450,000
South Chicago:				
International Harvester Co	3	754,400	3	731,000
Republic Steel Corp	1	450,000	1	455,000
United States Steel Co	11	4,196,700	----	----
United States Steel Corp. (central operations)	----	----	11	4,196,700
Youngstown Sheet & Tube Co	3	684,000	3	684,000
Total	22	6,983,120	22	7,103,320
Indiana (Chicago district)				
East Chicago: Youngstown Sheet & Tube Co	2	779,200	3	1,292,600
Gary:				
United States Steel Co	12	4,721,400	----	----
United States Steel Corp. (central operations)	----	----	12	4,755,400
Indiana Harbor: Inland Steel Co	8	2,638,950	8	2,638,950
Total	22	8,139,550	23	8,686,950
Kentucky (Pittsburgh-Youngstown district)				
Ashland: Armco Steel Corp	3	777,000	3	802,000
Maryland (Eastern district)				
Sparrows Point: Bethlehem Steel Co	8	3,252,000	9	4,116,000
Massachusetts (Eastern district)				
Everett: Eastern Gas & Fuel Associates	1	200,000	1	191,100
Michigan (Cleveland-Detroit district)				
Dearborn: Ford Motor Co	3	1,102,600	3	1,120,000
River Rouge: Great Lakes Steel Corp	3	1,200,000	4	1,680,000
Total	6	2,302,600	7	2,800,000
Pennsylvania (Pittsburgh-Youngstown district)				
Aliquippa: Jones & Laughlin Steel Corp	5	1,800,000	5	1,800,000
Brackenridge: Allegheny Ludlum Steel Corp	1	96,000	----	----
Braddock:				
United States Steel Co	7	2,602,700	----	----
United States Steel Corp. (central operations)	----	----	7	2,602,700
Clairton:				
United States Steel Co	3	[1] 614,000	----	----
United States Steel Corp. (central operations)	----	----	3	[2] 614,000
Donora: American Steel & Wire Division	2	450,000	2	450,000
Duquesne:				
United States Steel Co	6	[5] 1,451,400	----	----
United States Steel Corp (central operations)	----	----	6	[6] 1,262,700
Etna: United States Steel Co	2	[7] 145,000	----	----
Farrell: Sharon Steel Corp	2	504,180	2	561,000
McKeesport: National Tube Division	4	1,280,300	4	1,280,300
Midland: Crucible Steel Co. of America	2	535,000	3	895,000
Monessen: Pittsburgh Steel Co	3	954,000	3	954,000
Neville Island: Pittsburgh Coke & Chemical Co	1	280,000	2	654,000
Pittsburgh: Jones & Laughlin Steel Corp	6	1,872,000	6	1,927,000
Rankin:				
United States Steel Co	6	2,133,000	----	----
United States Steel Corp. (central operations)	----	----	6	2,133,000
Sharpsville: Shenango Furnace Co	2	417,300	2	445,450
Subtotal	52	15,134,880	51	15,579,150
Pennsylvania (Eastern district)				
Bethlehem: Bethlehem Steel Co	7	2,160,000	7	2,708,000
Birdsboro:				
Brooke Iron Division E. & G	1	151,200	----	----
Colorado Fuel & Iron Corp	----	----	1	151,200
Chester: Chester Blast Furnace, Inc	1	200,000	1	200,000
Erie: Interlake Iron Corp	1	167,580	1	270,750
Fairless Hills: United States Steel Corp. central operations	----	----	2	1,134,000
Johnstown: Bethlehem Steel Co	7	[8] 1,674,000	7	[9] 1,664,000
Palmerton: New Jersey Zinc Co	2	[10] 112,000	2	[11] 112,000
Sheridan: Lavino & Co., E. J	1	[12] 56,000	1	[13] 56,000
Steelton: Bethlehem Steel Co	3	842,000	3	876,000
Swedeland: Alan Wood Steel Co	2	454,800	2	454,800
Subtotal	25	5,817,580	27	7,626,750
Total Pennsylvania	77	[14] 20,952,460	78	[15] 23,205,900
Tennessee (Southern district)				
Lyles Wrigley: Tennessee Products & Chemical Corp	1	36,300	1	36,300
Rockwood: Tennessee Products & Chemical Corp	2	173,880	2	181,440
Total	3	210,180	3	217,740
Texas (Southern district)				
Houston: Sheffield Steel Corp	1	300,000	1	360,000
Lone Star: Lone Star Steel Co	1	366,600	1	385,000
Rusk: Valencia Iron & Chemical Corp	1	27,000	----	----
Total	3	693,600	2	745,000

Minnesota (Chicago district)				
Duluth:				
American Steel & Wire Division	2	449, 400	2	449, 400
Interlake Iron Corp	1	131, 670	1	131, 580
Total	3	581, 070	3	580, 980
New York (Eastern district)				
Buffalo:				
Hanna Furnace Corp	4	[16] 800, 000	4	850, 000
Republic Steel Corp	2	618, 000	2	618, 000
Lackawanna: Bethlehem Steel Co	6	2, 472, 000	7	3, 036, 000
North Tonawanda: Tonawanda Iron Division	1	171, 000	1	171, 000
Tonawanda: Colorado Fuel & Iron Corp	2	390, 000	2	390, 000
Troy: Republic Steel Corp	1	263, 000	1	263, 000
Total	16	4, 714, 000	17	5, 328, 000
Ohio (Pittsburgh-Youngstown district)				
Campbell: Youngstown Sheet & Tube Co	4	1, 450, 800	4	1, 450, 800
Canton: Republic Steel Corp	1	235, 000	1	235, 000
Hamilton: Armco Steel Corp	2	564, 000	------	------
Hubbard: Youngstown Sheet & Tube Co	1	200, 400	1	200, 400
Jackson:				
Globe Iron Co	1	[17] 100, 000	1	100, 000
Jackson Iron and Steel Co	1	[18] 93, 000	1	95, 000
Lowellville: Sharon Steel Corp	1	148, 620	1	148, 620
Martins Ferry:				
Louis Berkman Co	------	------	1	136, 800
Newport Steel Corp	1	136, 800	------	------
Massillon: Republic Steel Corp	1	238, 000	1	238, 000
Middletown: Armco Steel Corp	------	------	1	502, 000
New Miami: Armco Steel Corp	------	------	2	524, 000
Portsmouth: Detroit Steel Corp	1	284, 800	2	768, 700
Steubenville: Wheeling Steel Corp	5	1, 444, 500	5	1, 554, 000
Struthers: Pittsburgh Coke & Chemical Co	1	182, 500	1	182, 500
Warren: Republic Steel Corp	1	516, 000	1	516, 000
Youngstown:				
Republic Steel Corp	5	1, 723, 000	5	1, 717, 000
United States Steel Co	6	2, 003, 700	------	------
United States Steel Corp. (central operations)	------	------	6	2, 003, 700
Youngstown Sheet & Tube Co	2	501, 600	2	501, 600
Subtotal	34	9, 822, 720	36	10, 874, 120
Ohio (Cleveland-Detroit district)				
Cleveland:				
American Steel & Wire Division	2	530, 000	2	530, 000
Jones & Laughlin Steel Corp	2	540, 000	2	740, 000
Republic Steel Corp	5	1, 735, 000	6	2, 251, 000
Lorain: National Tube Division	5	1, 842, 500	5	1, 842, 500
Toledo: Interlake Iron Corp	2	551, 000	2	551, 180
Subtotal	16	5, 198, 500	17	5, 914, 680
Total Ohio	50	[21] 15, 021, 220	53	16, 788, 800
Utah (Western district)				
Geneva: Columbia-Geneva Steel Division	3	1, 200, 000	3	1, 200, 000
Ironton: Columbia-Geneva Steel Division	2	482, 700	2	482, 700
Total	5	1, 682, 700	5	1, 682, 700
Virginia (Southern district)				
Lynchburg: Lavino & Co., E. J	1	[12] 56, 000	1	[13] 56, 000
West Virginia (Pittsburgh-Youngstown district)				
Benwood: Wheeling Steel Corp	1	234, 000	1	246, 000
Weirton: Weirton Steel Company	3	1, 500, 000	4	2, 000, 000
Total	4	1, 734, 000	5	2, 246, 000
Grand total	251	[19] 73, 782, 340	260	[20] 82, 001, 390
Distribution by districts:				
Eastern	40	13, 983, 580	54	17, 261, 850
Pittsburgh-Youngstown	93	27, 468, 600	95	29, 501, 270
Cleveland-Detroit	22	7, 501, 100	24	8, 714, 680
Chicago	47	15, 703, 740	48	16, 371, 250
Southern	28	5, 648, 620	27	6, 273, 080
Western	11	3, 476, 700	12	3, 879, 260
Total	251	73, 782, 340	260	82, 001, 390

Source: American Iron and Steel Institute, Iron and Steel Works Directory of the United States and Canada: 1954, 497 pp.

[1] Includes 85,000 tons ferroalloys capacity.
[2] Includes 85,200 tons ferroalloys capacity.
[3] Includes 29.000 tons ferroalloys capacity.
[4] Includes 35,000 tons ferroalloys capacity.
[5] Includes 105,000 tons ferroalloys capacity.
[6] Includes 237,600 tons ferroalloys capacity.
[7] Includes 145,000 tons ferroalloys capacity.
[8] Includes 180,000 tons ferroalloys capacity.
[9] Includes 216,000 tons ferroalloys capacity.
[10] Includes 112,000 tons ferroalloys capacity.
[11] Spiegeleisen only.
[12] Includes 56,000 tons ferroalloys capacity.
[13] Ferromanganese only.
[14] Includes 683,000 tons ferroalloys capacity.
[15] Includes 706,800 tons ferroalloys capacity.
[16] Includes 120,000 tons ferroalloys capacity.
[17] Includes 100,000 tons ferroalloys capacity.
[18] Includes 93,000 tons ferroalloys capacity.
[19] Includes 1,081,400 tons ferroalloys capacity.
[20] Includes 797,800 tons ferroalloys capacity.
[21] Includes 193,000 tons ferroalloys capacity.

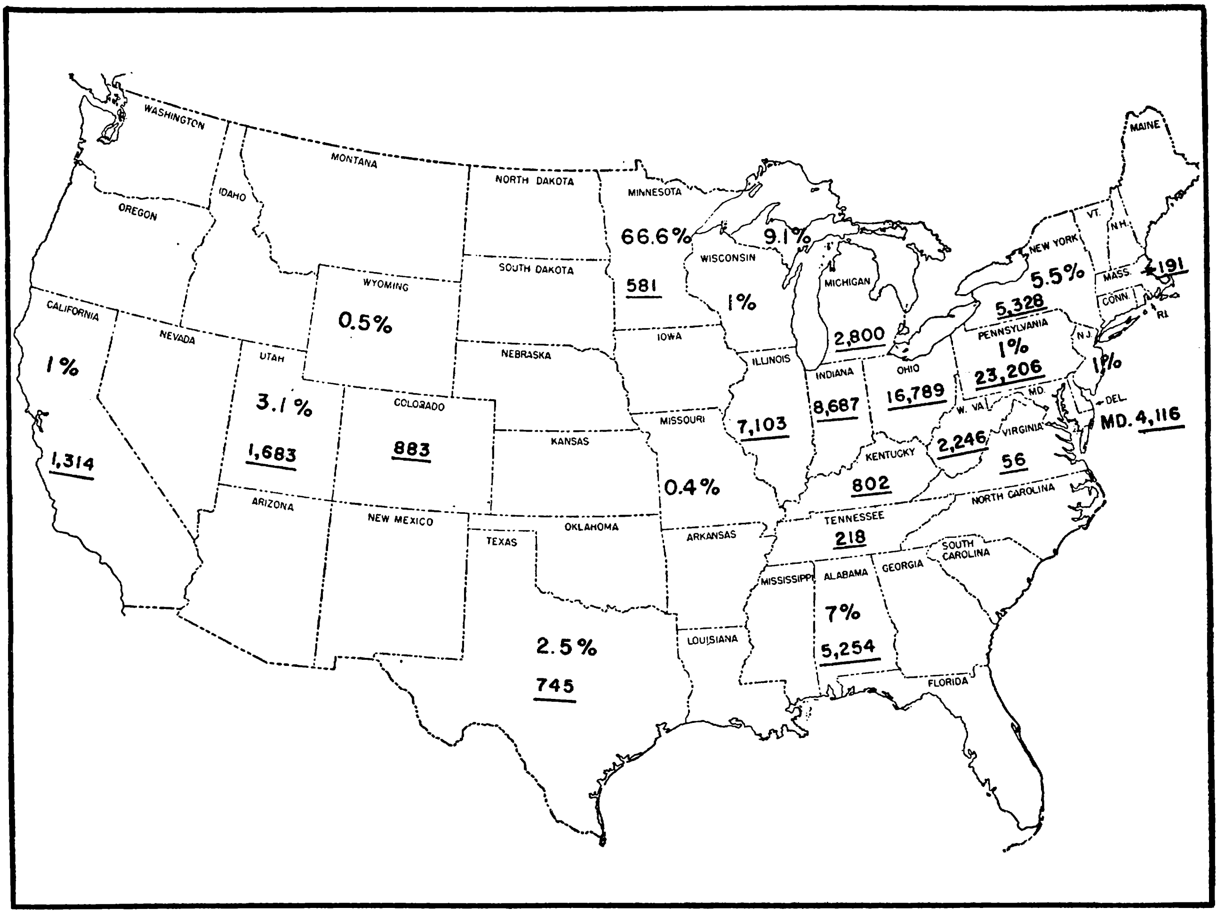

FIGURE 1.—Distribution of Iron-Ore Production for 1953 and Blast-Furnace-Capacity as of January 1, 1954, by States.

o/o, percent of United States total (crude) iron-ore production. Numbers underlined, thousand net tons of pig-iron capacity.

have no private knowledge of the existence of large, undeveloped domestic reserves. Of the listed reserves of direct-shipping ore on the Mesabi, Cuyuna, and Vermilion ranges, over three-fourths are controlled by the United States Steel Corp. This largest producer and consumer, admittedly in anticipation of restricted domestic supply is developing deposits in Venezuela. Five other steel companies that have depended upon the Lake Superior region for ore are now associated in developing the Quebec-Labrador deposits, and in Minnesota production of taconite concentrate is well underway.

LAKE SUPERIOR DISTRICT

The Lake Superior district supplied 80 percent of the 3,560 million tons of domestic output through 1952. Shipments from United States mines in the district approached 100 million tons in 1953 and can be expected to continue, as in the past, to provide the greater part of domestic supply.

SOUTHEASTERN STATES

The Southeastern States tonnage of available ore (table 5) is adequate to support the present scale of consumption for nearly 100 years. Expansion of ore output, however, can be accomplished only with difficulty.

Industry, aware that blast-furnace capacity can be increased by using richer ores, is blending high-grade imports with local red ores. Brown ores are also used in Birmingham furnaces to supplement and enrich the red ores. Extensive beneficiation of the local ores could minimize the need for imports, but the degree to which they can compete is limited. Small, independent mines in Georgia, Alabama, and eastern Tennessee ship ore in quantities ranging from one carlot to 40,000 tons a year.

All brown ores are concentrated by washing or gravity methods. The deposits are numerous, scattered, and irregular, so that reserve data in general would be difficult to obtain. Aggregate brown-ore output constitutes a substantial portion of the supply.

Table 2.—*Estimates of iron-ore reserves of the United States, by geographical regions as of January 1, 1950;*[1] *direct-shipping ore and concentrates by present methods of beneficiation*

[Million gross tons]

Region and State	1 Measured	2 Measured and indicated (includes column 1)	3 Measured, indicated, and inferred (includes column 2)
Lake Superior region: Minnesota, Michigan, Wisconsin: Total	[2] 1, 124	1, 720	[3] 3, 605
Northeastern region:[4] New York, New Jersey, Pennsylvania: Total	----------	300	380
Southeastern region:[5]			
Alabama and Georgia	----------	968	968
Tennessee, Virginia, North Carolina, West Virginia, Kentucky	----------	[6] 160	[6] 659
Total	----------	1, 128	1, 627
Central and Gulf region:			
Missouri [7]	----------	2	4
Texas [8]	17	27	60
Total	17	29	64
Western region:			
Utah [9]	----------	100	350
Wyoming [10]	----------	25	25
California [11]	28	33	54
New Mexico [12]	----------	19	39
Nevada [13]	----------	11	16
Total	28	188	484
United States	1, 169	3, 365	6, 160

[1] Compiled by Geological Survey and Bureau of Mines.
[2] From column 3 in table 3.
[3] (*71*); typographical error in reference corrected to above.
[4] See footnote 4, table 6.
[5] Measured and indicated not differentiated.
[6] Possible commercial production from this material will require thorough appraisal as to availability and value of deposits.
[7] (*9*).
[8] Geological Survey and Bureau of Mines, 1950; unpublished compilation.
[9] (*1*); (*54*).
[10] Measured and indicated not differentiated.
[11] (*36* and *39*).
[12] (*49*); excludes manganiferous iron ore and other deposits of marginal grade or less than 10,000-ton reserve.
[13] Bureau of Mines, 1950; unpublished compilation.

Table 3.—*Estimates of iron-ore reserves of the Lake Superior region as of January 1, 1950;*[1] *direct-shipping ore and concentrates by present methods of beneficiation*

(Does not include taconite or iron-formation requiring similar treatment)

[Million gross tons]

	Measured ore; estimates for taxation		
	Direct shipping	Concentrates	Total
Mesabi	[2] 751	[2] 161	912
Minnesota (excluding Mesabi)	[2] 48	[2] 8	56
Michigan	[3] 150	--------	150
Wisconsin	[3] 6	--------	6
Total	955	169	1, 124

[1] Compiled by the Geological Survey.
[2] Minnesota Department of Taxation.
[3] (*71*).

Table 4.—*Estimated reserves of marginal ore of the Lake Superior region*[1]

(Concentration techniques not perfected)

[Million gross tons]

	Crude material	Concentrate product
Mesabi district:		
Magnetic taconite [2]	5, 000 (20–27% Fe)	1, 700 (63% Fe)
Taconite [3] (including magnetic taconite)	60, 000	20, 000
Gogebic range:		
Wisconsin [4]	1, 050	263
Michigan [5]	300	120
Iron County (Mich.)[5]	120	48
Dickinson County (Mich.)[5]	525	210
Marquette district (Mich.)[5]	865	346

[1] Compiled by Geological Survey.
[2] (*38*).
[3] (*66*).
[4] (*7*).
[5] (*60*).

TABLE 5.—*Tentative estimate of reserves, Birmingham Basin (revised July 1, 1952)* [1]

[Gross tons]

	Virgin deposits	Idle mines	Operating mines	Total
Total by classes:				
Ferruginous sandstone	248, 311, 000	209, 021, 000	575, 860, 000	1, 033, 192, 000
Marginal ore	204, 189, 000	117, 771, 000	257, 432, 000	579, 392, 000
Commercial ore	237, 820, 000	25, 398, 000	284, 667, 000	547, 885, 000
Total	690, 320, 000	352, 190, 000	1, 117, 959, 000	2, 160, 469, 000
Total by seams:				
Upper Ferruginous	124, 240, 000	5, 350, 000	426, 200, 000	555, 790, 000
Ida		35, 310, 000	10, 040, 000	45, 350, 000
Middle Ferruginous	124, 071, 000	109, 171, 000	142, 400, 000	375, 642, 000
Upper Big	128, 997, 000	28, 299, 000	219, 213, 000	376, 509, 000
Lower Big	237, 329, 000	120, 150, 000	240, 563, 000	598, 042, 000
Irondale	75, 683, 000	53, 910, 000	79, 543, 000	209, 136, 000
Total	690, 320, 000	352, 190, 000	1, 117, 959, 000	2, 160, 469, 000
Classes by seams:				
Ferruginous sandstone:				
Upper Ferruginous	124, 240, 000	5, 350, 000	426, 200, 000	555, 790, 000
Ida			7, 260, 000	7, 260, 000
Middle Ferruginous	124, 071, 000	109, 171, 000	142, 400, 000	375, 642, 000
Lower Big		94, 500, 000		94, 500, 000
Total	248, 311, 000	209, 021, 000	575, 860, 000	1, 033, 192, 000
Marginal ore:				
Ida		35, 310, 000	2, 780, 000	38, 090, 000
Upper Big	11, 997, 000	18, 299, 000	12, 719, 000	43, 015, 000
Lower Big	122, 959, 000	25, 650, 000	174, 056, 000	322, 665, 000
Irondale	69, 233, 000	38, 512, 000	67, 877, 000	175, 622, 000
Total	204, 189, 000	117, 771, 000	257, 432, 000	579, 392, 000
Commercial ore:				
Upper Big	117, 000, 000	10, 000, 000	206, 494, 000	333, 494, 000
Lower Big	114, 370, 000		66, 507, 000	180, 877, 000
Irondale	6, 450, 000	15, 398, 000	11, 666, 000	33, 514, 000
Total	237, 820, 000	25, 398, 000	284, 667, 000	547, 885, 000

[1] (*68*).

NORTHEASTERN STATES

The Northeastern States are not considered capable of large increases beyond the present 5-million-ton output. This supply is important, nevertheless, because it is composed of lump ore and high-grade concentrates, which averaged 62 percent iron content in 1953.

Open pits supply nearly half of current output, but underground mines are more numerous. All of the ore now mined occurs in metamorphic rocks. Open pits are generally of low grade, but the combination of low-cost surface mining, magnetic concentration, and relatively large output permits an economic operation. Underground mines exploit similar deposits of higher grade. Although more than one level may be worked simultaneously, expansion of output is limited by hoisting facilities and other factors, mechanical and economic. Lump ore for open-hearth consumption is taken from the richer portions of some deposits, and lower grade material is subjected to concentration.

OTHER IRON-ORE RESOURCES

In *northeast Texas*, an area of several counties has deposits of brown ore aggregating an estimated 60 million tons after concentration. The annual output is slightly over 1 million tons. The ore carries a high percentage of water and varies widely in iron content and response to beneficiation. Many individual deposits not now exploited could be brought into production by the application of suitable beneficiation methods. Other brown-ore deposits in *Arkansas* and *Missouri* support a modest output, and one mine produces small quantities of magnetite.

The most important producing area in the *Western States* is southern *Utah*, which supplies furnaces at Geneva and Provo, Utah; Pueblo, Colo.; and Fontana, Calif. An inferred 350 million tons of partly altered magnetite occurs in large, wedge-shaped masses mined by open-pit methods. In recent years, much of the ore exported to Japan has come from these deposits.

Principal mines in other Western States include the Eagle Mountain mine near Desert Center, *Calif.*, operated by the Kaiser Steel Corp., and the Sunrise mine in *Wyoming*, operated by Colorado Fuel & Iron Corp. Reserves are estimated in table 2 to be 54 and 25 million tons, respectively. Further exploration should reveal additional tonnages in these States. *Montana*, *Oregon*, and *Washington* have partly explored deposits of significant size and varying quality. Technologic advances eventually will bring one or more of them into operation as economic growth in the Northwest demands additional iron and steel.

During the last 25 years the average grade of usable ore (direct ore, concentrates, and sinter) in the United States has averaged slightly over 50 percent iron content. The only area in which the grade has been lower is the Southeastern district. There, the principal production of ore averaged around 37 percent iron but was usable due to its self-fluxing character.

From table 2 it can be seen that domestic, high-grade ores total 1,169 million tons of measured reserves. At the rate of 100 million tons a year, these would last about 11 years from 1950.

NEW SOURCES OF IRON ORE

Because of the foresight of American private industry, a new pattern of iron-ore supply has begun to emerge. Major sources of foreign ore have been and are being developed. Taconite concentrates are being produced in large amount, and various of the simpler beneficiation techniques are being more widely used. Research along many lines is being undertaken.

TACONITE CONCENTRATES

The reserves of low-grade ores in the Lake Superior district appear as unlimited with respect to present-day requirements as did the high-grade ores in the early days, with the important difference that the low-grade ores will be more expensive. This applies in terms of time as well as in terms of manpower, supplies, and capital investment. One estimate states that "to add 20 million tons of capacity would take 2 or 3 years." For this reason, taconite production cannot be relied upon for rapid expansion in an emergency. A question has also been raised as to the maximum possible rate of production. Lack of water, excessive overburden, or some other factor may limit annual production to something at or near the announced plans for 43 million tons.

According to the Minnesota Department of Taxation:

> Taconite is defined as a ferruginous chert or ferruginous slate in the form of compact siliceous rock in which the iron oxide is so finely disseminated that substantially all of the iron-bearing particles of merchantable grade are smaller than 20-mesh. Taconite may be further defined as iron-ore-bearing rock which is not merchantable as iron ore in its natural state and which cannot be made merchantable by simple methods of beneficiation involving only crushing, screening, washing, jigging, drying, or any combination thereof.

Similar formations known by various names exist in other parts of the world, but those of the Lake Superior district (including the Michigan jaspers) have undoubtedly received most study. Table 4 shows that the large tonnages are on the Mesabi.

FOREIGN SOURCES

Among the foreign iron-ore deposits of immediate importance to the United States, *Quebec-Labrador* is prominent because of its magnitude, availability, and potential flexibility. Exploration has measured 418 million tons of direct-shipping, open-pit ore. It is anticipated that substantial additional reserves will be located in the course of development and operation in the area. The measured tonnage was considered the minimum to support the planned scale of mining. Output from here is tentatively scheduled to be 6 million tons in 1955 and 10 million tons in 1956. In addition to the high-grade, direct-shipping ores, at least 1 billion tons of lower grade material averaging about 40 percent iron and amenable to gravity concentration is indicated by reconnaissance mapping close to the recently completed railroad.

Atikokan district of Canada (Steep Rock), a part of the Lake Superior region, listed total proved and probable ore reserves at the end of 1953 at approximately 90 million tons on the A, B, and G ore zones and 100 million tons on the C ore zone (8). Ultimate reserves far exceed this quantity, but great depth and mining difficulties will restrict production to a few million tons a year. The ore bodies are upturned, and highly fractured, thus causing a serious problem of water drainage. Output in 1953 was 1.3 million tons.

Newfoundland deposits, though large, are too high in phosphorus to attract American customers. However, substantial production is being exported to Europe where that impurity is less objectionable. Newfoundland ore could be used in the United States if basic Bessemer converters were installed to reject the phosphorus.

The iron-ore deposits at El Pao, *Venezuela* are steeply inclined tabular masses of hard hematite with some magnetite. A minimum of 60 million tons of open-hearth grade minable by open-pit methods has been estimated (53). However, an earlier investigator (20) believed that the deposits contained 100 million tons of ore averaging 68 percent iron. El Pao began ore shipments to Bethlehem's Sparrows Point plant in 1951 and expects to expand according to needs and as Chilean output declines.

The deposits at *Cerro Bolivar* are 50 miles south of Cuidad Bolivar, Venezuela, on the Orinoco River and 65 miles west of El Pao. The ore bodies cap a mountain and extend about 5 miles in length and 300 to 3,500 feet in width. A pinch near the middle of the largest ore body divides it into 2 parts, each about 1¼ miles long; 1 has a maximum width of 2,000 feet and the other 3,500 feet. Another large ore body, very irregular in outline, is about 1½ miles long, with a maximum width of 3,500 feet. Three smaller ore bodies are lenses about ½ to ¾ mile long and 300 to 500 feet wide. The ore averages 230 feet in thickness, with a maximum of 550 feet thus far. Cerro Bolivar approximately equals the Mahoning-Hull-Rust pit on the Mesabi range in area. The ore is a mixture of hematite and limonite, with minor amounts of magnetite; it occurs as lenses in closely folded ore-bearing layers. The average grade of the deposit on a dry basis is 63.0 percent Fe, 0.13 percent P, 0.80 percent SiO_2, 0.10 percent Mn, 1.80 percent Al_2O_3, 6.50 percent loss on ignition, and 8 percent moisture; the ore contains a maximum of 0.04 percent sulfur. Overburden is light, and the ore is mined by open-pit methods. Shipment requires a 90-mile rail haul to Puerto Ordaz on the Orinoco River, where the ore is loaded into ocean-going vessels.

Although other deposits of less extent have been discovered nearby, only the deposits on Cerro Bolivar have been explored thoroughly enough to permit estimates to be made. The figure given for Cerro Bolivar is 500 million tons (53). The total tonnage, including adjacent deposits in the Cerro Bolivar area, is thought to exceed 1,000 million tons.

Iron-ore deposits in *Brazil* are numerous and widely distributed, but the information needed for estimating reserves is inadequate for most of them. The principal deposits are in two areas in the southern part of the country. The richest and best known area is in the State of Minas Geraes; the other is in the State of Matto Grosso near Corumba.

Most iron-ore production in Brazil has come from Minas Geraes, where iron ore is produced in three important districts. The Itabira district, with Caue Peak as the most important producing mine, is the source of most of the export material. This ore is shipped via the Vitoria e Minas Railroad 350 miles east to Vitoria for export mainly to the United States and the United Kingdom. Iron ore for the Volta Redondo iron and steel plant in Brazil is obtained from the Casa do Pedro mine near Congonhas do Campo, the second most important Brazilian district. At Belo Horizonte, iron ore is produced mainly from the Sabara mine for other Brazilian plants.

The deposits at Corumba are rich in iron (55 percent) but are too inaccessible and too high in silica (up to 20 percent) to be in a favorable economic position at present. It has been estimated (*25*) that 1,310 million tons of ore is available. One deposit of detrital ore that was examined would probably yield a screened product of 64 percent iron and 4 percent silica.

The principal iron-ore deposits of *Chile*, near El Tofo, are masses of compact hematite with minor quantities of magnetite. The ore contains about 58 percent Fe, 0.01 percent S, 0.011 to 0.5 percent P, and 2.5 percent SiO_2. These deposits continue to supply ore to the United States, but future production for export will decline as Venezuelan output increases. Iron-ore lenses at Romeral are similar to the Tofo ores. It has been estimated that these deposits (Romeral) contain 20 million tons of ore with 60 percent iron (*29*).

Cuba is an important potential foreign supplier of iron-ore concentrates to the United States. Aside from several small deposits of varying character, the huge deposits of lateritic iron ore along the north coast constitute one of the larger groups of ore bodies in the world. Table 6 shows 3 billion tons of reserves, but the total may reach 3 or 4 times that figure.

The ore forms a blanket that consists mainly of soft, earthy iron oxides containing a large percentage of water and significant quantities of nickel, chromium, and cobalt. Currently, nickel is being extracted by a complex process, and the other minerals are discarded.

Because Cuba is the nearest of the overseas sources and the tonnage of reserves is so large, the development of means to utilize these ores is of first importance. Development of a successful method for beneficiating lateritic iron ores would make deposits of similar nature in the United States as well as other parts of the world available for consumption.

Liberia began to ship premium-grade lump ore to the United States in 1951 from the Bomi Hills deposit about 40 miles north of Monrovia.

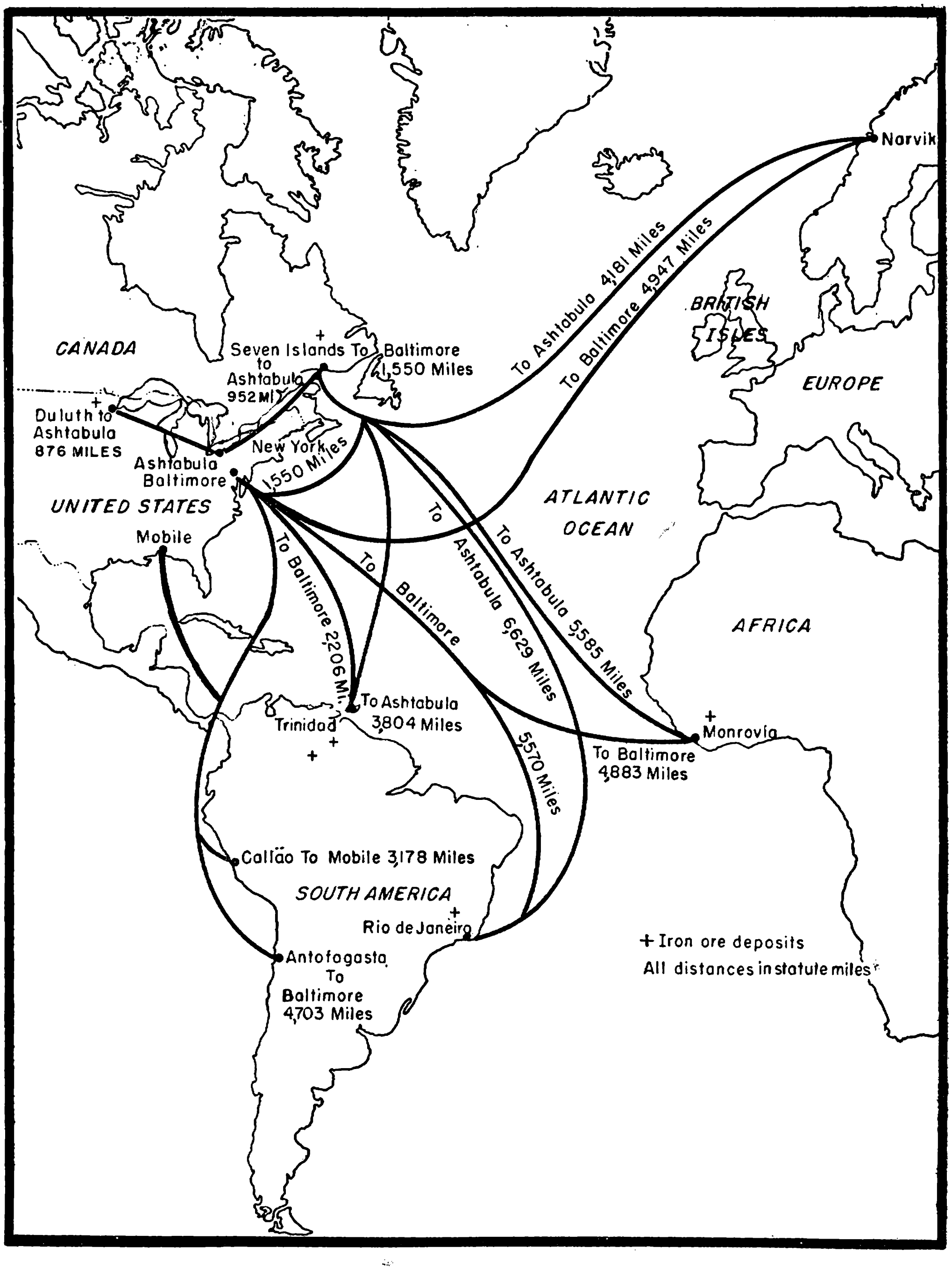

FIGURE 2.—Locations and Distances of Foreign Sources of Iron Ore.

The ore bodies at Bomi Hills form a series of massive lenses arranged mainly around 2 sides of an irregular basin about 1½ miles long. The main ore body is about 2,500 feet long and at depth dips under the basin. The measured and indicated reserves of high-grade ore calculated by the Federal Geological Survey are about 20 million tons. The ore averages 66 to 68 percent Fe, 2 to 3 percent SiO_2, 0.03 to 0.05 percent S, and 0.06 to 0.08 percent P; a few million tons of hard weathered ore averages 0.02 to 0.04 percent P. There are 100 million tons or more that average about 42 percent Fe and 37 percent SiO_2.

The most productive iron-ore deposits of *Algeria* are in the southeastern part of the country; they are lenticular masses of hematite with 50 to 60 percent iron and low phosphorus, sulfur, and silica. Similar deposits occur in Tunisia and possibly in Morocco and Tripoli. Estimated reserves in *Algeria* and *Tunisia* are 100 to 150 million tons.

Iron-ore deposits are present in several parts of *Sweden*. The most important of these are 20 to 90 miles from the northern border. The deposits at Kiruna are exposed in an area a little less than 2 miles long and 100 to 500 feet wide. The ore bodies are steeply inclined, tabular masses of magnetite and have been drilled to a depth of over 1,000 feet. The usable ore has a composition of 56 to 69 percent Fe, 0.19 to 2.76 percent P, 0.017 to 0.05 percent S, and 1.67 to 10.5 percent SiO_2. The reserves of the Kiruna area have been conservatively estimated as 750 million tons (metric) and may total 2,000 million tons (*46*).

Somewhat similar deposits are at Gellivare, 50 miles south of Kiruna. The material is coarse-grained magnetite in large lenses that partly overlap on parallel and curving trends. A reserve of 350 million tons has been estimated by the Bureau of Mines.

A belt of iron-ore deposits in the southern part of the Province of Kopparberg contains six principal deposits as lenses of hematite and magnetite. The irregular ore bodies are as much as 3,300 feet long and 65 to 310 feet wide. The crude ore contains 45 to 50 percent iron, but concentration produces material that contains 59 to 64 percent iron and 0.77 to 1.0 percent phosphorus. Reserves to a depth of 1,600 feet have been estimated at 150 million tons (*62*).

Table 6 summarizes selected reserve data classified by grades, and figure 2 shows the location of various foreign sources with the distances to United States ports. Ashtabula, Ohio, an unloading port on Lake Erie for Lake Superior iron ore, is also a probable unloading port for Quebec-Labrador ore.

TABLE 6.—*Summary of iron-ore resources of the United States and selected foreign countries*[1]

[Million gross tons]

Country	Grade of ore (percent Fe)		
	Over 50	35–50	25–35
United States:			
Lake Superior	[2] 3,605		[3] 60,000
Northeastern		[4] 1,000	
Southeastern	[5] 105	1,127	[6] 1,033
Central and Gulf		[6] 63	
Western	460	25	
Total	4,170	2,215	61,033
Canada:			
Labrador-Quebec	1,000	1,000	
Newfoundland	3,500		
Ontario	70		100
Total	4,570	1,000	100
Venezuela	1,060		
Brazil	1,500	3,500	
Chile	20		
Cuba		3,000	
Algeria and Tunisia	125		
Liberia	20	100	
Sweden	2,300	150	

[1] Measured, indicated, and inferred.
[2] (*71*) typographical error in reference. Corrected to above.
[3] Taconite.
[4] 25–45 percent ore will probably yield 380 million tons of concentrates.
[5] Concentrates.
[6] (*68*).

SECONDARY SOURCES AND RECOVERY

Byproduct or coproduct sources of primary material for use in ironmaking are limited at present. The major supply of this type material consists of an iron oxide residue remaining from the roasting of pyrites and pyrrhotite in the manufacture of sulfuric acid. This source now produces approximately 500,000 gross tons (averaging about 66 percent iron) of furnace feed a year. Some companies ship the residue in its natural powdery form (from roasted flotation concentrates), while others sinter it before shipment. The burned pyrite is a premium product because of its high grade and because, after agglomeration, it can be used to enrich most blast-furnace and open-hearth feeds. It is being produced as a byproduct iron ore in Virginia and Tennessee from domestic ores and in Delaware from foreign as well as domestic ores. Ilmenite will doubtless gain importance as a source of byproduct iron "ore."

Other secondary sources of iron-bearing material for blast-furnace consumption include recycled materials, such as flue dust (ore, coke, and stone fines that are carried out of the furnace top with gases and recaptured); mill scale (iron oxide formed on cooling ingots and during hot-forming operations); and open-hearth furnace slag containing iron oxides and entrained metal. Scrap iron and steel unsuitable for steelmaking (amounting to nearly 4.8 million tons in 1953) is charged back into blast furnaces with ore to produce new pig iron.

Stockpiling, although not a source of iron ore, can be considered in that category because of its function, which would be to provide immediate access to large reserves in case of an emergency. It is currently being advocated for taconite concentrates and might in future be considered because of the time required for constructing beneficiating plants.

Utilization of smaller deposits is another development that should be considered as should new inventions, new smelting methods, or lowered transportation costs.

MINING

Owing to the large tonnages involved, iron-ore production is largely a problem of materials handling. This applies particularly to open-pit mining operations which, in 1953, supplied 124 million tons or 79 percent of the domestic crude (unbeneficiated) ore output. Open-pit operations predominate in the Mesabi and Cuyuna ranges of Minnesota, in the Western States except Wyoming, and in the brown-iron-ore operations of Alabama, Georgia, Minnesota, and Texas. This type of operation also will be used at the new mines in Venezuela, Labrador, and Liberia. Underground mining is practiced to a smaller degree but predominates in Michigan, Wisconsin, the Eastern States, and Wyoming. In a number of instances both types of mining are employed on the same ore body.

Taconite mining can perhaps be summarized in the same terms as open-pit mining in the past—development drilling, removal of overburden, blasting, loading, and haulage; but hardness of material, variations in composition, and vastly greater tonnage have brought many changes. Development drilling will encounter a greater percentage of hard rock, so that the more economical chopping or "structure" drilling of the district will lose ground in favor of diamond drilling. Overburden will consist not only of glacial drift but of unusable taconite or other rock; therefore the dip of the formations will be an increasingly important consideration. Drilling blast holes in taconite has been so troublesome that new drilling methods are being evolved. About 3 tons of taconite for each ton of concentrate, in addition to overburden, will have to be moved by the loading and haulage equipment.

Underground mining accounts for most of the production from the Marquette, Gogebic, and Menominee ranges of Michigan and Wisconsin, the Vermilion range in Minnesota, and for some production from the Mesabi and Cuyuna ranges in Minnesota. The various Lake Superior underground iron mines provide approximately 60 percent of total domestic underground output. The deposits vary appreciably both in inclination and in depth. In the Mesabi area several of the mines are worked jointly as open pit underground operations, while mines in the Marquette range are as much as 3,000 feet deep. During the history of the Lake Superior district, almost all standard mining methods have been used; shrinkage stopes, block caving, top slicing, sublevel stoping, open stopes, and filled stopes. At the present time most production is by sublevel stoping, being largely governed by the type of ore. Mechanization of operations and improved methods in recent years have greatly increased the production per man-shift.

Although four open-pit operations in New York and Pennsylvania provide nearly half of the usable ore output from Northeastern States, underground mining still predominates in number of operations and total output. Ore is generally mined by sublevel or open-stope systems.

In the Birmingham district hematite beds dip at an angle of about 15° below the outcrop and become essentially horizontal at depth. Mining originally (1890–1900) was near the surface, but most operations are at present under 1,500 to 1,700 feet of cover. With but one exception the ore is hoisted through inclined slopes which follow the old workings. In certain instances these haulageways are as much as 10,000 feet long, and it seems likely that there will be some further conversion to vertical shaft hoisting. An open-stope room-and-pillar system of mining is used to recover ore from the slightly inclined to flat ore beds. About 50 to 60 percent of the ore is mined in the primary operation; and total recovery, after mining of pillars, approximates 75 percent.

In Wyoming the Sunrise mine produces hematite ore by the block-caving method. The block being caved is first undercut with drifts and crosscuts and finally caved by blasting out pillars left between them. The ore then falls of its own weight and is drawn off through chutes at the bottom of the block. Preliminary costs of development for this type of mining are relatively high; but overall costs are low, and the method permits a very high tonnage output per man.

BENEFICIATION

Beneficiation of domestic subgrade iron ores offers at least a partial alternative to increasing dependence upon foreign sources for an adequate supply. No clear-cut choice is possible, but a compromise based on strategic and economic considerations is indicated.

The trend in favor of one or another source will shift according to future developments with respect to security, economics, and technology. Of these, technology offers the highest degree of control. Progress has been made, and much of the ore now consumed without treatment could advantageously be subjected to beneficiation in some form. In its simpler forms the treatment may include only washing to reject mud or slime, crushing and sizing, or sizing alone. For example, in the Birmingham district, it was shown some years ago that sizing and layering sizes in the blast furnaces, rather than indiscriminate charging of sizes as formerly practiced, increased the iron output of the furnace by at least 10 percent. The treatment of ores to improve physical characteristics is an important avenue of research among milling problems, but such material is not normally included in statistics of beneficiated ores.

From the simple forms of beneficiation mentioned, the required treatment for various ores may progress, as needed, through many procedures of increasing complexity until a product is obtained that can be exploited commercially. The cost of the treatment versus the value of the product will remain the deciding factor as regards the feasibility of the methods used; but, if the consumption of concentrated material continues to increase percentagewise, it seems inevitable that domestic iron-ore production will rise in relative cost until importation of ore prevents further rise in cost.

A rough classification of iron ores can be based on the particle size at which the iron mineral is broken away or liberated from the gangue with which it is associated.

Crushing of iron-bearing materials produces some fines that usually are discarded, sintered, or otherwise reagglomerated without concentration or treated by the methods required for beneficiating finely divided ores.

Direct-shipping ore is mined, crushed to sizes (minus-4-inch) that are suitable for charging to the blast furnaces, and shipped without further treatment. The minus-¼-inch material produced in crushing may be sintered or otherwise agglomerated but usually is shipped with the coarse ore. In some instances the coarse ore is sized into several fractions and charged separately into the blast furnaces.

A second type of ore consists of iron-bearing materials that require physical concentration by rejection of gangue diluents but that can be treated usually in minus-3-inch to ⅛-inch sizes. The method most commonly used to treat such materials is heavy-medium separation. In this process ores are thoroughly washed (actually scrubbed by self-abrasion when necessary), wet-screened on about ¼-inch screens to reject slimes and fine granular material that does not respond to this method, and passed through separating vessels of various types that contain a water suspension of powdered ferrosilicon (a rust-resistant alloy of iron and silicon). The heavier iron minerals sink through the suspension to the bottom of the separating vessel, while the gangue of lighter specific gravity floats on the surface and overflows to a washing screen and drainage. The sink product is concurrently removed from the separating vessel and similarly drained and washed on a vibrating screen. The ferrosilicon medium is continuously recovered and recirculated in the system. The separation is rapid (up to several hundred tons per hour can be treated in a single unit); inexpensive (treatment costs rarely exceed 25 cents per ton of feed, including amortization); and flexible, in that the effective specific gravity of the ferrosilicon medium can be adjusted to produce any grade of product desired, provided the iron minerals in the feed treated are well liberated from gangue and a satisfactory recovery of iron is obtained.

The heavy-medium-separation method described is being used in Canada, Minnesota, and Alabama on hematite ores and in Alabama and Texas on brown or limonite-type iron ore. Recently the method has been put into operation at Eagle Mountain, Calif.

A third type of material, in the Mesabi area of Minnesota, is known as wash ore. This material is washed and the particles scrubbed together (in coarse sizes) to reject what is locally known as paint rock—a ferruginous material high in insoluble matter. Slimes are rejected to waste. Some brown ores in Georgia and Texas are similarly subjected to scrubbing in log washers, the slimes and fine sizes rejected, and the coarse sizes hand-picked on conveyor belts to reject coarse chert or mud-ball gangue.

Many beneficiation processes have been proposed and tested for recovering iron minerals in the minus-¼-inch to 100-mesh size range. Full descriptions of the methods are not presented here, but additional information on the processes can be obtained from references cited in the bibliography (*35*, *62*, *65*, *66*). Processes that have been used for treating iron-bearing materials in this size range have been based essentially on the fact that the specific gravity of the various iron minerals is appreciably greater than that of the gangue minerals that are rejected. Jigging, tabling, hydraulic classification, Humphreys spiral concentration, and various combinations and modifications of the procedures have been used to recover iron minerals from low-grade feeds in this size range. Most of these methods have had limited capacity; but a process now under plant trial (treatment of minus-¼-inch, plus-35-mesh ore in a cyclone-type separator, using powdered magnetite or ferrosilicon as a suspending medium) promises to solve the problem of treating intermediate-size ore in a high-capacity operation if certain technical difficulties can be solved.

At present, a few jig plants are in use in Minnesota concentrators; several plants are using Humphreys spiral concentrators for minus-35-, plus-150-mesh hematite; several others are using modified forms of hydraulic classification to reject fine silica; and 2 plants are experimenting with heavy-medium separation in cyclone-type separators.

One plant in Alabama crushes ore to pass ⅛-inch and rejects shale and silica in the minus-⅛-inch to 100-mesh sizes by hydraulic classification and tabling. In a second installation, minus-¼-inch to 100-mesh fines from a heavy-medium concentrator are being treated by jigging to reject shale.

Iron-bearing materials that must be crushed to pass 65-mesh or finer sizes for liberation include the large reserves of magnetic and nonmagnetic taconites, as well as the Michigan jaspers and eastern ironbearing sandstones. These are the sources that will supply the huge requirements of future generations, and some of them are beginning to contribute to the current supply.

Of the many variations in the Lake Superior taconites, two principal classes are identified—slaty taconite and cherty taconite. As implied by the name, slaty taconite is extremely fine grained and requires grinding to micron sizes before iron minerals are adequately liberated from silica. Such fine grinding is excessively high in cost and eventually may be responsible for a shift toward chemical or pyrometallurgical methods of treatment. Conversely, the cherty taconites, which must be divided further into magnetic and nonmagnetic types, are relatively coarse grained and usually require grinding to pass only 100- to 150-mesh screens for liberation. The facilities now being constructed on the Mesabi are designed to treat the coarser grained magnetic portion of the taconites. The concentration process is simple magnetic separation, but the treatment becomes involved as a result of the numerous steps connected with crushing and grinding and the difficulties of agglomerating the fine concentrates into a product of suitable physical structure.

The total output of taconite concentrate is estimated to reach 21 million tons annually by 1958. Results achieved by that time may support further expansion. However, to attain this yearly output, many problems remain to be solved, aside from the construction of plant facilities. Continuing research will be required to develop the most economical means of mining a material difficult to penetrate and break. Crushing and grinding procedures will be subject to much experimentation and revision in an effort to reduce wear and breakage of equipment. After the iron concentrate is produced, the fine particles must be agglomerated into a suitable structure for furnace consumption. This latter problem has received much attention recently with considerable success. Pelletizing, nodulizing, briquetting, and sintering have been tried, but the choice of the most satisfactory method has not been unanimous.

Fine-grained nonmagnetic (hematite, limonite, siderite) taconites, which comprise about 80 percent of the estimated taconite reserves, as well as the ferruginous sandstones of other States, present much more difficult concentration problems. Companies embarking on magnetic taconite concentration have operating data from commercial experience in the magnetic concentration of ores from New York, New Jersey, and Pennsylvania mills to guide their planning. A similar guide is not available for application to possible processes for recovering iron from nonmagnetic taconites.

There are two schools of thought as to the best and most economical method of recovering iron from fine-grained, nonmagnetic materials. One proposes the reduction roasting of the crude ore to produce artificial magnetite, which is then to be recovered by magnetic separation. The other favors flotation of iron minerals from water suspensions by means of surface-active chemical reagents or, conversely, depression of iron minerals in a flotation unit while floating and rejecting gangue particles, particularly silica. In either process, the iron products would require filtration and some form of agglomeration (sintering, briquetting, nodulizing, extrusion, or pelletizing) to prepare the fine-size product for use in the blast furnace. Figure 3 shows highly simplified flowsheets for the two types of processing.

With many minerals, rough concentration tests can be made, cost estimates prepared (perhaps allowing 50 to 100 percent for contingencies), and a commercial unit constructed. More comprehensive studies must precede commercial beneficiation of iron ores (particularly with non-magnetic iron ores) because of the multiple costs involved and the narrow margin of profit for any method used. Thus, in the roasting and magnetic separation process, reduction roasting, grinding, and agglomeration of concentrates are the major costs. In flotation the grinding, chemical reagents, and agglomeration of concentrates are the major costs. In both instances mining is also a major cost, inasmuch as 3 to 4 tons of ore, depending on the grade of the natural iron, is required to produce 1 ton of concentrate.

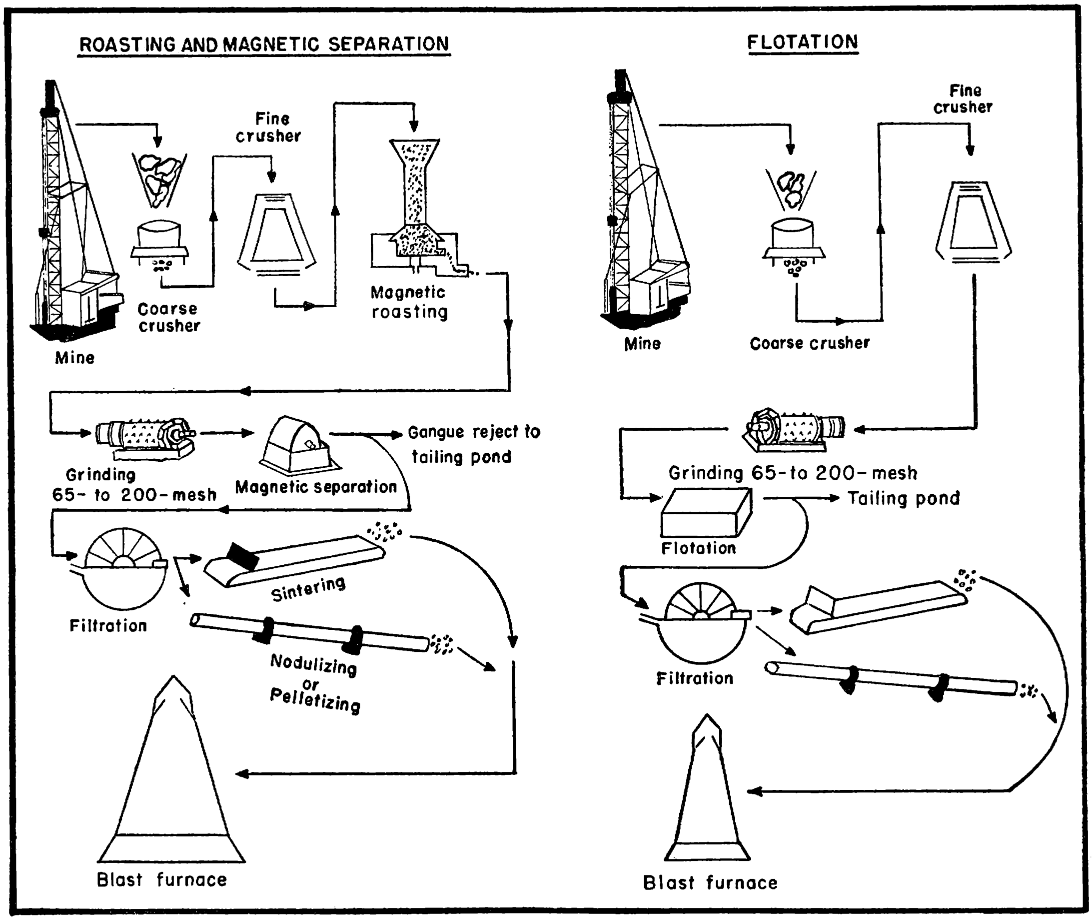

FIGURE 3.—Simplified Flowsheet Showing Different Methods of Treating Nonmagnetic, Low-Grade Iron Ores.

METALLURGY

The earliest processes for smelting iron, such as those used in China and medieval England, employed low-temperature reduction of iron oxide ores to form a spongy mass containing metallic particles. The particles of metal were sorted out and worked into usable objects by means of alternate heating and hammering. In most instances the product was relatively impure due to attachments and inclusions of other materials. It was many years later that improved furnaces and blowing apparatus first permitted the production of metal that could be drawn from the furnace in a molten state and cast. This advance yielded iron of higher purity and was due to the higher temperatures achieved in the furnace and the combination of carbon from the charcoal with iron, which afforded a reduction in the melting point of the metal. Before the 14th century, when furnace fusion was first accomplished, essentially all iron was equivalent to the wrought iron of today.

The smelting of iron remained a relatively small-scale operation until the 19th century, when the development of railroads greatly increased the demand. Progress was still slow until Sir Henry Bessemer developed the process (1856) that bears his name. The Bessemer converter provided a means of producing large quantities of steel through rapid elimination of carbon from molten iron. Since that time much additional knowledge of iron and steel processing has been gained and many improvements of the operations effected, but the basic principles are essentially the same as in 1856.

The modern blast furnace has become a large unit, expensive to construct, but by far the most economically operated of the various iron producing methods. Its size has resulted largely from efforts to economize on the consumption of coke, the most expensive component of the charge. Loss of heat and gas generated from coke is much greater in the smaller furnaces, and high thermal efficiency is the prime goal of today's blast-furnace operators.

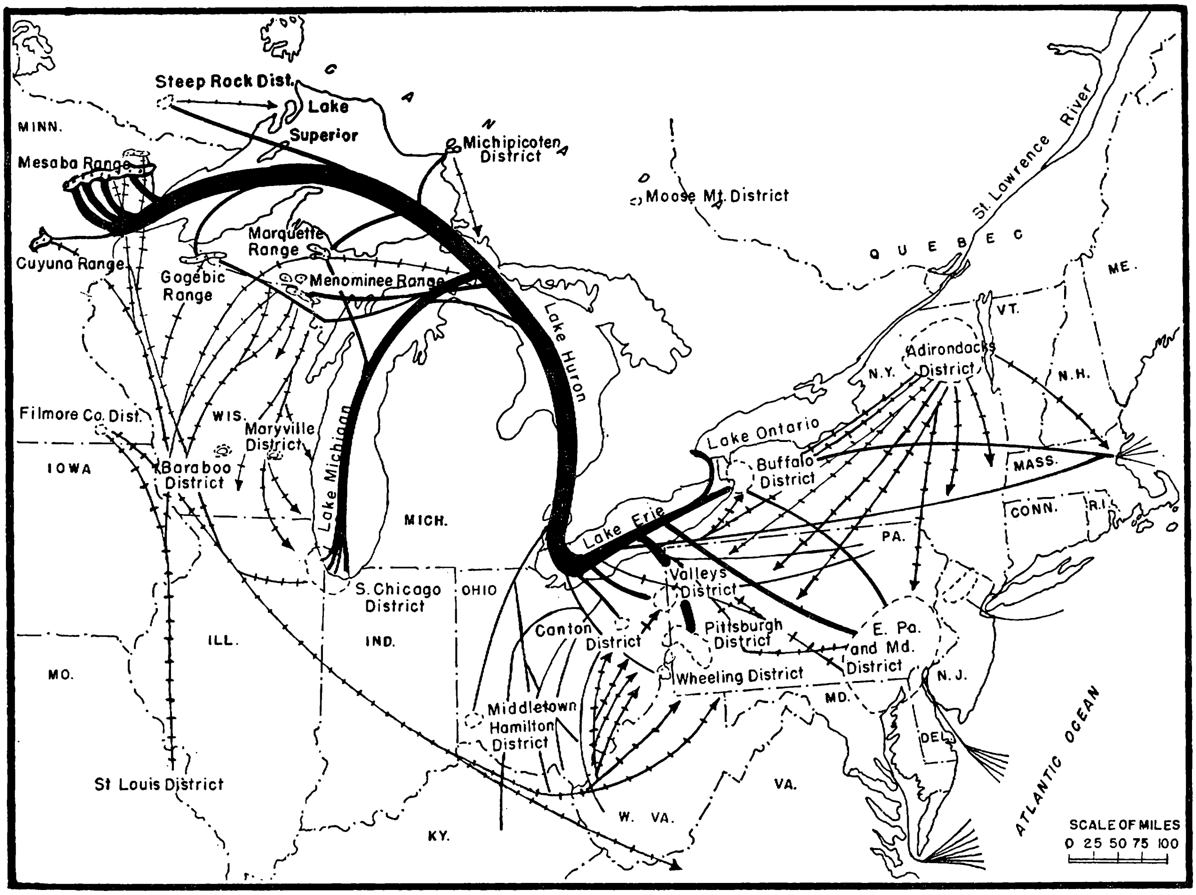

FIGURE 4.—Flow Map Showing Movement of Lake Superior Iron Ores for 1951.

(Courtesy, Lake Superior Iron Ore Association, 1400 Hanna Bldg., Cleveland 15, Ohio.)

Essentially, a blast furnace receives through its top a continuing series of charges composed of ore, coke, and limestone. These move slowly downward through the stack, and molten pig iron and slag are discharged at the bottom and gas at the top. Ore is converted to metal by carbon reduction of the iron. In effect, carbon from the coke combines with oxygen from the ore, freeing the iron to accumulate at the bottom of the furnace. Coke has the important additional function of supplying heat, and stone is necessary to form a slag and eliminate impurities. Moreover, the formation of slag progressively reduces the temperature necessary to melt the ore and stone. Mixed limestone and silica will melt at a lower temperature than either material alone. This is the fluxing action. Figure 5 is a diagram of a modern blast-furnace plant.

A blast furnace using Lake Superior ore of approximately 50 percent iron content, operating under average conditions, will use the following average quantities of raw materials per ton of pig iron product:

Ore, sinter, etc.	lb.	3,800.
Limestone	do.	800–1,200.
Coke	do.	1,700–1,900.
Air	cu. ft.	116,000.

In addition to the pig iron, 1,100 or more pounds of slag and 200 to 400 pounds of flue dust will be produced. The overall recovery of iron, considering only the metal content of the pig iron, is about 93 percent. Most producers now market their slag as a byproduct for use as aggregate in road building and, to some extent, for agricultural purposes.

Top gas from the blast furnace, containing appreciable quantities of carbon monoxide, is usually burned during passage through one of the stoves. Fresh air to be blown into the blast furnace is then drawn through the heated stove to be preheated before entrance into the furnace. The pig iron produced in the furnace is the basic raw material for refining processes used in the manufacture of steel as well as metal for gray-iron castings.

Over the years, research on furnace methods has been directed toward improving the physical nature of feeds; increasing reduction rates and efficiencies by pressurizing the furnace to provide better gas contact; developing improved methods of charging raw materials into the unit; better furnace design; improving blast generation; oxygen enrichment of blast; and other details of the complex operation. Almost all work has been directed primarily toward reduction of the coke requirements of the processes. However, modern blast furnaces are so large and expensive that experimentation proceeds with difficulty. It is logical that furnace foremen tend to resist any radical changes in procedure, because a miscalculation or any other unexpected development can create great difficulties in operation, perhaps taking a furnace out of production several days.

Any major changes in furnace design probably will result from a decline in coking-coal supplies. Technicians in some foreign countries, notably Germany and England, have been investigating the possibilites of a shorter column blast furnace of different design which can use noncoking coal or other carbonaceous materials as the reducing agent. Economic application of such a unit would permit some decentralization of ironmaking in the United States, inasmuch as this type of furnace need not be so large and could be supported by a smaller market.

A number of units and processes other than the blast furnace have been tested in past years for the production of iron. It is technically feasible to produce iron in a number of ways; and in some countries, where special conditions exist, units other than blast furnaces are in operation. In Norway, Sweden, Finland, Italy, and Japan, electric furnaces of several designs have produced appreciable tonnages of pig iron. The process requires only about half the coke used in blast furnaces because electrical energy replaces coke for heating purposes. Installation costs on a per-ton-of-pig-iron basis are smaller than those for blast furnaces. The difficult economic features are electric power costs and moderately increased labor costs. Electric furnaces can compete with blast furnaces when ample supplies of electric power are available at low rates—usually from hydroelectric sources—but have not succeeded with steam-generated electricity.

Notwithstanding this situation, as large high-grade ore deposits are depleted and coke and transportation costs continue to rise, the production of electric-furnace pig iron may, in future, become economic in some parts of the United States.

For many years there has been much interest in development of new and economic methods for producing iron by direct reduction—that is, conversion of iron from the oxide form of the ore to the metallic state without actual melting. Bureau of Mines experiments with such processes have indicated that, when relatively pure iron oxide ores or concentrates are used, this type of smelting yields high-grade metal suitable for fabrication or for a scrap substitute in steelmaking. On the other hand, if lower grade ores are used, serious complications arise in the production process as well as in utilization of the iron.

Probably the most effective low-temperature method that has progressed to commercial-scale operation is the Swedish Wiberg-Soderfors sponge-iron process. In this process, ore or concentrate travels by gravity through a shaft-type unit in a manner similar to blast-furnace operation, but the product is removed from the bottom of the shaft in a metallic but unmelted form. As the ore travels downward, carbon monoxide gas generated in a separate unit passes upward and effects reduction; this gas is recirculated and regenerated. Coke and flux are not introduced into the furnace. All impurities in the ore, except sulfur occur in the final product, but the carbon content of the reduced metal remains low, inasmuch as the sponge iron does not have an opportunity to absorb carbon or sulfur from coke, as is the case in blast-furnace operation.

The unit can be operated under special conditions but, as was the case with electric furnaces, has not proved to be directly competitive with the blast furnace (*16*). Table 7 compares costs and operational data, indicating that, at the time of the study (1949), pig-iron production in blast furnaces cost $28.43 per ton; in the Tysland-Hole electric furnace, $35.32; and in the Wiberg-Soderfors sponge-iron furnace, $35.02.

Iron can be produced at low temperatures in brick kilns, in rotary kilns, and by fluidized-solids methods. Some methods continue to promise eventual commercial application but at present remain largely noncompetitive with blast furnaces. One great problem of producing such iron on a continuous basis is the difficulty of preventing reoxidation of the finely divided iron until it can be cooled to a temperature where oxidation of iron progresses slowly.

Another type of direct-reduction method that should be mentioned is the Krupp-Renn, a German process that has been commercially installed in a number of foreign countries. This process, using a rotary kiln, treats a mixture of fine ore, coal or coke, and fluxing material. In practice this mixture is heated until iron is reduced and actually melted to form globules of metal. These remain coated with slag, and reoxidation is thus prevented. After being cooled to normal temperature, the kiln discharge is crushed to free iron from the slag, and metal is recovered by magnetic

TABLE 7.—*Comparative production costs and investment requirements of the standard blast furnace, electric smelting furnace, and sponge-iron furnace* [1]

Item	Blast furnace	Electric smelting furnace [2]	Sponge-iron furnace [2]
PLANT AND FURNACE CAPACITY			
Total yearly output ... tons	280, 000	110, 000	45, 000
Number of furnaces	1	3	2
Daily output per furnace ... tons	800	100	64
PRODUCTION COSTS (DOLLARS) PER TON OF IRON [3]			
Coke	[4] 9. 35	[5] 5. 12	[6] 3. 60
Ore	[7] 14. 50	[7] 14. 50	[8] 20. 75
Limestone at $2.25 per ton	. 97	. 88	. 09
Electrodes at $134 per ton		1. 80	. 40
Power at 0.3 cents per kw.-hr	----------	7. 50	2. 70
Cooling water at 0.003 cent per gallon	. 42	. 33	. 06
Labor at $1.20 per hour	. 94	2. 19	2. 52
Repairs and maintenance	. 50	. 95	1. 65
Overhead and miscellaneous (including casting)	3. 65	3. 80	3. 25
Production cost	30. 33	37. 07	35. 02
Gas credit at 25 cents per million B. T. U	1. 90	1. 75	----------
Net production cost	28. 43	35. 32	35. 02
TOTAL INVESTMENT (THOUSAND DOLLARS) [9]			
Furnaces and auxiliary equipment	12, 000	5, 000	1, 300
Buildings	2, 000	1, 000	400
Ore and material-handling equipment	2, 000	1, 000	300
Coke ovens	6, 000	----------	----------
Working capital (6 months)	4, 000	2, 000	750
Engineering and contingencies	2, 000	750	250
Total	28, 000	9, 750	3, 000
INVESTMENT PER TON-YEAR (DOLLARS)			
Total	100	89	67

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[1] (*16*).

[2] Tysland-Hole electric smelting furnace and Wiberg-Soderfors sponge-iron furnace.

[3] Production cost estimates based on typical operating conditions, reflecting 1948 cost of production factors in Canada. Production costs do not include profits or fixed charges on investment.

[4] Coke made in own plant at $11.25 per ton: 0.83 ton.

[5] Purchased mixed coke (metallurgical coke at $16.50 per ton, fines at $8.25 per ton): 0.45 ton.

[6] Purchased coke at $16.50 per ton: 0.22 ton.

[7] 50 percent iron content at $7.25 per ton: 20 tons.

[8] 56 percent iron content at $11.50 per ton: 1.8 tons (selected lump Steep Rock ore).

[9] Investment costs are not intended to be exact nor to apply to any particular location. They are representative costs in Canada in 1948 and are intended only for purposes of comparison.

separation. The fine-size iron particles must then be melted or fused together in some manner before use as a substitute for scrap iron in steelmaking furnaces. Various reports indicate that the method, though workable, has not proved to be competitive with blast furnaces in reducing iron ore.

The blast-furnace method continues to be the most economical one for producing pig iron, but economic changes in the individual component costs of iron production may eventually make operation of smaller scale units of other types feasible in special circumstances. Such developments promise to provide means of conserving natural resources by enabling localized utilization of domestic ore deposits that are not amenable to centralized blast-furnace operations.

STATISTICAL REPORTING

Data on the iron industry are reported annually in the Minerals Yearbook series of the Bureau of Mines. Ore statistics and industry notes are presented in the chapter Iron Ore and for pig iron in the chapter Iron and Steel. The information is compiled from questionnaires returned to the Bureau of Mines from each operating company. Coverage is virtually 100 percent complete.

The University of Minnesota, Mines Experiment Station (H. H. Wade, acting director) publishes yearly the Mining Directory of Minnesota, giving data on individual mining operations within the State. Additional information is presented for the Lake Superior region. Lake shipments and iron-ore consumption by those firms depending upon the Lake Superior region are reported by the Lake Superior Iron Ore Association.

United States production of iron and steel, with corollary statistics, are compiled and published by the American Iron and Steel Institute. Various State agencies publish statistical data on activities within their States from independent surveys or in cooperation with the Bureau of Mines.

Statistical data in connection with foreign trade are collected by the Bureau of the Census, United States Department of Commerce, and published by both that agency and the Bureau of Mines. Data on foreign production and trade are compiled by the Bureau of Mines from official foreign Government publications and from consular reports supplied by the Division of Foreign Reporting, United States Department of State.

PRICES

Published prices for iron ore include values for certain imported ores, eastern concentrates, and the Lake Erie base prices for Lake Superior ores.

Foreign ores are quoted in cents per long-ton unit (each 22.4 pounds of iron contained) delivered at United States ports. Most of the higher prices listed are for premium grades for consumption in steel furnaces.

Lake Erie base prices are established each year by the publication of a major contract between a prominent producer and a steel corporation. Other companies follow this lead and negotiate contracts according to their own best interests.

Notwithstanding published prices for imported and domestic iron ore, most requirements are supplied from sources wholly or partly integrated with the consuming companies, and direct purchases are estimated to comprise less than 25 percent of the total. In the Lake Superior district, trade channels are complicated by intercompany trading to obtain blends suitable for a consistent furnace feed. Most of the ore changing hands and subject to Lake Erie base prices probably is in this category.

Outside of the Lake Superior region, a larger proportion of the output is through integrated operations, but a number of independent producers also contribute to the supply. Sales are subject to individual contracts, and prices are published only for northeastern magnetites.

Pertinent information on the Lake Erie pricing system and mining cost data for Michigan and Minnesota are quoted from the Mining Directory of Minnesota and the Annual Statistical Report, Michigan Department of Conservation.

Ore prices for varying iron content calculation of Lake Erie selling values [1]

[According to formula adopted in 1925, and still in use]

Standard Lake Erie selling values for iron ore, as quoted in trade journals and ore sales contracts, are per gross tons of 2,240 pounds, delivered at rail of vessel at Lower Lake Ports and are based on the following classification and guaranteed base analyses:

Old Range Bessemer, 51.50% iron natural, 0.045% Phos.
Old Range Non-Bessemer, 51.50% iron natural.
Mesabi Bessemer, 51.50% iron natural, 0.045% Phos.
Mesabi Non-Bessemer, 51.50% iron natural.
High-Phosphorus, 51.50% iron natural, 0.180% Phos.

Price adjustments for iron content above or below the guarantee; all grades

Selling values of ores of different iron content than the base ores are determined as follows: The base price is divided by 51.50, the number of units in the base ore. The resulting quotient is the base unit value, used to determine additions to or subtractions from the base price, for iron contents above or below the base analysis, as follows:

When less than 51.50% and not less than 50.00% Iron: From the base price deduct, for each unit or fraction of a unit of iron less than 51.50% iron, at the rate of the base unit value.

When less than 50.00% and not less than 49.00% Iron: From the price computed for 50.00% iron deduct, for the unit or fraction of a unit of iron less than 50.00% iron, at the rate of one and one-half times the base unit value.

When less than 49.00% Iron: From the price computed for 49.00% iron deduct, for each unit or fraction of a unit of iron less than 49.00% iron, at the rate of two times the base unit value.

When over 51.50% Iron: To the base price add, for each unit or fraction of a unit of iron more than 51.50% iron, at the rate of the base unit value.

Price adjustment for phosphorus

All ores containing 0.045% phosphorus, or less, are classed as Bessemer. Phosphorus content lower than 0.045% commands a premium, determined in accordance with the standard table of phosphorus values. All ores containing more than 0.045% phosphorus are classed as Non-Bessemer. Ores containing more than 0.180% phosphorus are classed as High Phosphorus.

Penalties

In addition to the standard deductions applied for iron contents of less than 50%, which are computed as above, arbitrary penalties are also exacted for high silica and for fine structure.

Premiums for lump structure and high manganese content

Hard ores of high-iron, low-silica contents are often sold as lump grade, generally being priced as Old Range Non-Bessemer plus premiums for lump structure.

Ores containing in excess of 5% natural manganese are recognized as standard manganiferous iron ores and are generally priced as Old Range Non-Bessemer on the combined natural iron and manganese content, plus a premium for the natural manganese in excess of 5%. Ores containing between 2% and 5% of natural manganese are also sometimes marketed as manganiferous at prices which recognize some small value for the manganese content.

Premiums for lump structure and high manganese content vary and are determined by negotiation between buyer and seller.

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[1] (74.)

TABLE 8.—*Lake Erie base prices of iron ore* [1] *and valley prices of Bessemer and No. 2 foundry pig iron at date of ore-buying movement* [2]

Season	Date buying movement	Old Range Bessemer	Old Range Non-Bessemer	Mesabi Bessemer	Mesabi Non-Bessemer	High phosphorus	Bessemer pig iron	No. 2 foundry pig iron
1930	Apr. 1, 1930	$4. 80	$4. 65	$4. 65	$4. 50	$4. 40	$19. 00	$18. 50
1931	Apr. 15, 1931	4. 80	4. 65	4. 65	4. 50	4. 40	17. 00	17. 00
1932	June 3, 1932	4. 80	4. 65	4. 65	4. 50	4. 40	14. 50	14. 50
1933	June 7, 1933	4. 80	4. 65	4. 65	4. 50	4. 40	16. 00	15. 50
1934	May 21–26, 1934	4. 80	4. 65	4. 65	4. 50	4. 40	19. 00	18. 50
1935	Apr. 23, 1935	4. 80	4. 65	4. 65	4. 50	4. 40	19. 00	18. 50
1936	Apr. 1, 1936	4. 80	4. 65	4. 65	4. 50	4. 40	20. 00	19. 50
1937	Mar. 8, 1937	5. 25	5. 10	5. 10	4. 95	4. 85	24. 50	24. 00
1938	May 23, 1938	5. 25	5. 10	5. 10	4. 95	4. 85	24. 50	24. 00
1939	May 3, 1939	5. 25	5. 10	5. 10	4. 95	4. 85	21. 50	21. 00
1940	Apr. 16, 1940	4. 75	4. 60	4. 60	4. 45	4. 35	23. 50	23. 00
1941	Apr. 17, 1941	4. 75	4. 60	4. 60	4. 45	4. 35	24. 50	24. 00
1942 [3]	Apr. 10, 1942	4. 75	4. 60	4. 60	4. 45	4. 35	24. 50	24. 00
1943 [3]		4. 75	4. 60	4. 60	4. 45	4. 35	24. 50	24. 00
1944 [3]		4. 75	4. 60	4. 60	4. 45	4. 35	24. 50	24. 00
1945 [3]		4. 95	4. 80	4. 70	4. 55	4. 55	25. 50	[4] 25. 00
1946 [3]		5. 45	5. 30	5. 20	5. 05	5. 05	27. 00	[4] 26. 50
1947	Jan. 25, 1947	5. 95	5. 80	5. 70	5. 55	5. 55	31. 00	30. 50
1948	Mar. 27, 1948	6. 60	6. 45	6. 35	6. 20	6. 20	40. 00	39. 50
1949	Dec. 30, 1948	[5] 7. 60	[5] 7. 45	[5] 7. 35	[5] 7. 20	[5] 7. 20	47. 00	46. 50
1950	Jan. 26, 1950	8. 10	7. 95	7. 85	7. 70	7. 70	47. 00	46. 50
1951 [6]	Dec. 2, 1950	8. 70	8. 55	8. 45	8. 30	8. 30	53. 00	52. 50
1952 [6]	July 26, 1952	9. 45	9. 30	9. 20	9. 05	9. 05	55. 50	55. 00
1953 [6]	Feb. 12, 1953	10. 10	9. 95	9. 85	9. 70	9. 70	55. 50	55. 00
1953 [6]	July 1, 1953	10. 30	10. 15	10. 05	9. 90	9. 90	55. 50	55. 00
1954 [6]		10. 30	10. 15	10. 05	9. 90	9. 90	57. 00	56. 50

[1] Based on following analysis: Bessemer, 51.50 percent Fe (natural) and 0.045 percent phosphorus (dry); Non-Bessemer, 51.50 percent Fe (natural).

[2] *(74)*.

[3] Prices controlled by the Office of Price Administration.

[4] Maximum per gross ton, established by Office of Price Administration.

[5] 6 percent increase in dock-unloading charge of $0.18, or $0.0108, added to buyers' account, effective January 11, 1949.

[6] Iron-ore prices subject to adjustment for changes in ore transportation and handling costs from mines to rail of vessel at lower Lake ports, including rail, dock and vessel charges and transportation taxes thereon, as follows: 1951 and 1952 prices by the amount of any change after Dec. 1, 1950; Feb. 12, 1953, prices by the amount of any change after Dec. 31, 1952: July 1, 1953 and 1954 prices by the amount of any change after June 24, 1953.

TABLE 9.—*Minnesota iron ore ad valorem taxes levied, 1914–52* [1]

	Ad valorem taxes			
	State	County	Local	Total
1914–15	$2, 422, 416	$2, 649, 422	$8, 863, 364	$13, 935, 202
1916–20	7, 525, 564	11, 743, 432	50, 899, 138	70, 168, 134
1921	1, 203, 473	3, 040, 145	13, 941, 538	18, 185, 156
1922	1, 161, 288	2, 951, 031	14, 299, 181	18, 411, 500
1923	2, 298, 710	3, 300, 036	14, 056, 522	19, 655, 268
1924	1, 682, 383	3, 143, 135	13, 910, 838	18, 736, 356
1925	2, 149, 882	2, 984, 651	13, 436, 296	18, 570, 829
1926	1, 458, 007	2, 912, 173	12, 897, 499	17, 267, 679
1927	1, 972, 268	3, 167, 651	12, 202, 463	17, 342, 382
1928	1, 347, 033	3, 129, 570	12, 367, 746	16, 844, 349
1929	1, 592, 537	3, 290, 144	12, 369, 019	17, 251, 700
1930	1, 366, 684	3, 262, 329	12, 456, 632	17, 085, 645
1931	1, 883, 194	3, 382, 985	11, 351, 038	16, 617, 217
1932	1, 959, 006	3, 201, 138	10, 697, 346	15, 857, 490
1933	2, 643, 812	3, 247, 220	10, 691, 097	16, 582, 129
1934	2, 762, 996	4, 059, 152	10, 843, 984	17, 666, 132
1935	3, 062, 746	3, 931, 227	10, 329, 856	17, 323, 829
1936	2, 798, 071	4, 459, 946	10, 754, 161	18, 012, 178
1937	2, 024, 419	4, 009, 528	11, 235, 620	17, 269, 567
1938	2, 004, 850	4, 123, 766	10, 126, 596	16, 255, 212
1939	1, 953, 413	4, 601, 422	9, 876, 478	16, 431, 322
1940	1, 810, 014	4, 374, 856	9, 394, 986	15, 579, 856
1941	1, 507, 775	3, 951, 242	9, 105, 236	14, 564, 253
1942	1, 451, 024	3, 506, 085	8, 286, 928	13, 244, 037
1943	893, 996	3, 677, 474	8, 728, 633	13, 300, 103
1944	662, 625	3, 462, 913	8, 351, 732	12, 477, 270
1945	1, 019, 654	3, 291, 772	8, 276, 887	12, 588, 313
1946	1, 026, 087	3, 714, 909	7, 991, 773	12, 732, 769
1947	888, 768	5, 125, 429	7, 909, 331	13, 923, 528
1948	914, 255	4, 823, 156	7, 520, 417	13, 257, 828
1949	1, 141, 709	5, 195, 204	8, 564, 674	14, 901, 587
1950	1, 319, 986	5, 908, 781	8, 766, 200	15, 994, 967
1951	1, 106, 126	5, 656, 363	9, 697, 886	16, 460, 375
1952	1, 113, 353	6, 535, 064	10, 109, 842	17, 758, 259
Total	62, 128, 124	139, 813, 351	400, 310, 946	602, 252, 421

AUTHORITY: Minnesota Department of Taxation.

[1] *(74)*.

TABLE 10.—*Average production costs of open-pit and underground ore produced in Minnesota* [1]

Year	Total tonnage mined	Tota lcost of development, royalty, and mining	Average cost per ton of development	Average cost per ton of mining and beneficiation					Average cost per ton of royalty paid	Average cost per ton of all preceding items
				Labor	Supplies	Total labor and supplies	Other items (including beneficiation)	Total		
Open-pit operations:										
1940	44, 008, 093	$44, 640, 364	$0. 217	$0. 108	$0. 109	$0. 217	$0. 184	$0. 401	$0. 397	$1. 015
1941	58, 771, 355	60, 547, 192	. 218	. 138	. 109	. 247	. 149	. 396	. 418	1. 032
1942	64, 951, 827	72, 290, 635	. 202	. 154	. 131	. 285	. 232	. 517	. 394	1. 113
1943	63, 761, 539	75, 491, 717	. 221	. 195	. 152	. 347	. 267	. 614	. 352	1. 187
1944	61, 177, 038	75, 309, 811	. 246	. 185	. 170	. 355	. 279	. 634	. 351	1. 231
1945	59, 012, 981	72, 960, 183	. 217	. 183	. 175	. 358	. 320	. 678	. 341	1. 236
1946	47, 312, 655	61, 036, 079	. 232	. 199	. 188	. 387	. 325	. 712	. 346	1. 290
1947	56, 648, 191	77, 761, 752	. 266	. 217	. 232	. 449	. 331	. 780	. 327	1. 373
1948	61, 075, 597	93, 888, 374	. 313	. 219	. 251	. 470	. 405	. 875	. 349	1. 537
1949	51, 804, 480	88, 647, 173	. 360	. 260	. 258	. 518	. 500	1. 018	. 333	1. 711
1950	61, 098, 092	111, 225, 426	. 416	. 292	. 221	. 513	. 531	. 044	. 360	1. 820
1951	74, 382, 213	148, 105, 427	. 507			. 570	. 570	1. 140	. 344	1. 991
Underground operations:										
1940	4, 296, 565	10, 140, 522	. 040	. 947	. 487	1. 434	. 507	1. 941	. 381	2. 362
1941	4, 964, 992	11, 466, 023	. 060	1. 033	. 501	1. 534	. 335	1. 869	. 380	2. 309
1942	5, 096, 889	12, 877, 388	. 054	1. 238	. 543	1. 781	. 347	2. 128	. 344	2. 526
1943	5, 242, 922	13, 655, 699	. 064	1. 353	. 550	1. 903	. 293	2. 196	. 343	2. 603
1944	3, 896, 438	10, 847, 052	. 043	1. 321	. 628	1. 949	. 425	2. 374	. 367	2. 784
1945	3, 469, 065	10, 139, 631	. 050	1. 403	. 637	2. 040	. 392	2. 432	. 441	2. 923
1946	2, 337, 701	7, 622, 325	. 044	1. 734	. 780	2. 514	. 324	2. 838	. 379	3. 261
1947	3, 319, 570	11, 542, 070	. 043	1. 787	. 797	2. 584	. 441	3. 025	. 409	3. 477
1948	3, 938, 109	13, 845, 709	. 055	1. 697	. 808	2. 505	. 390	2. 895	. 566	3. 516
1949	3, 383, 391	12, 853, 923	. 047	1. 896	. 846	2. 742	. 380	3. 122	. 630	3. 799
1950	3, 694, 127	15, 511, 552	. 048	2. 112	. 668	2. 780	. 726	3. 506	. 645	4. 199
1951	3, 925, 073	17, 749, 167	. 040			3. 077	. 764	3. 841	. 640	4. 521

[1] Tonnage of all ore mined in Minnesota in years 1940–51, inclusive (excluding taconite production); comparison of total costs and costs per ton for development and other costs incurred in mining, as between open-pit and underground operations (*56* and *74*).

TABLE 11.—*Total Michigan underground mines* [1]

[Average costs per ton]

	1950	1951	5-year average
Cost of mining:			
Labor	$1. 9298	$2. 3185	$1. 9474
Supplies	. 8522	1. 0097	. 8457
	2. 7820	3. 3282	2. 7931
Deferred costs	. 1810	. 2966	. 1884
Taxes:			
General property	. 1818	. 1908	. 1772
State corporation	. 0073	. 0089	. 0075
Social Security	. 0488	. 0518	. 0451
	. 2379	. 2515	. 2298
General overhead:			
General office	. 0986	. 0915	. 0799
General superintendence	. 0420	. 0429	. 0368
General insurance	. 0039	. 0038	. 0037
Other expense	. 0089	. 0147	. 0054
Employee insurance	. 1254	. 1119	. 0541
Depreciation	. 1163	. 1533	. 1045
	. 3951	. 4181	. 2844
Transportation:			
Rail freight	. 9760	. 9629	. 9294
Boat freight	1. 3544	1. 4320	1. 2719
Cargo insurance	. 0025	. 0025	. 0025
	2. 3329	2. 3974	2. 2038
Marketing:			
Selling	$0. 0468	$0. 0431	$0. 0466
Analysis	. 0053	. 0054	. 0061
	. 0521	. 0485	. 0527
Total direct ore costs	5. 9810	6. 7403	5. 7522
Other reported costs:			
Royalty	. 3330	. 3529	. 3142
Federal income tax	. 5284	. 6004	. 4190
	. 8614	. 9533	. 7332
Total	6. 8424	7. 6936	6. 4854
Lake Erie value per ton [2]	7. 8096	8. 3632	7. 1539

[1] Michigan Department of Conservation, Geological Survey Division.
[2] Computed Lake Erie price adjusted for sulfur analyses.

TRANSPORTATION

The general flow of Lake Superior ore is shown in figure 4. Great Lakes and Canadian ore movements will be mainly completed during an 8-month season although "all rail" transportation may occasionally be used during winter months. Open sea routes farther south, figure 2, will not be seasonal but will perhaps be more vulnerable to interruption in event of war.

Of the planned 10-million ton production from Quebec-Labrador deposits 7 to 8 million tons is expected to go to Atlantic ports and 2 to 3 million tons to be transhipped at Montreal to Great Lakes ports in carriers small enough to negotiate present canals.

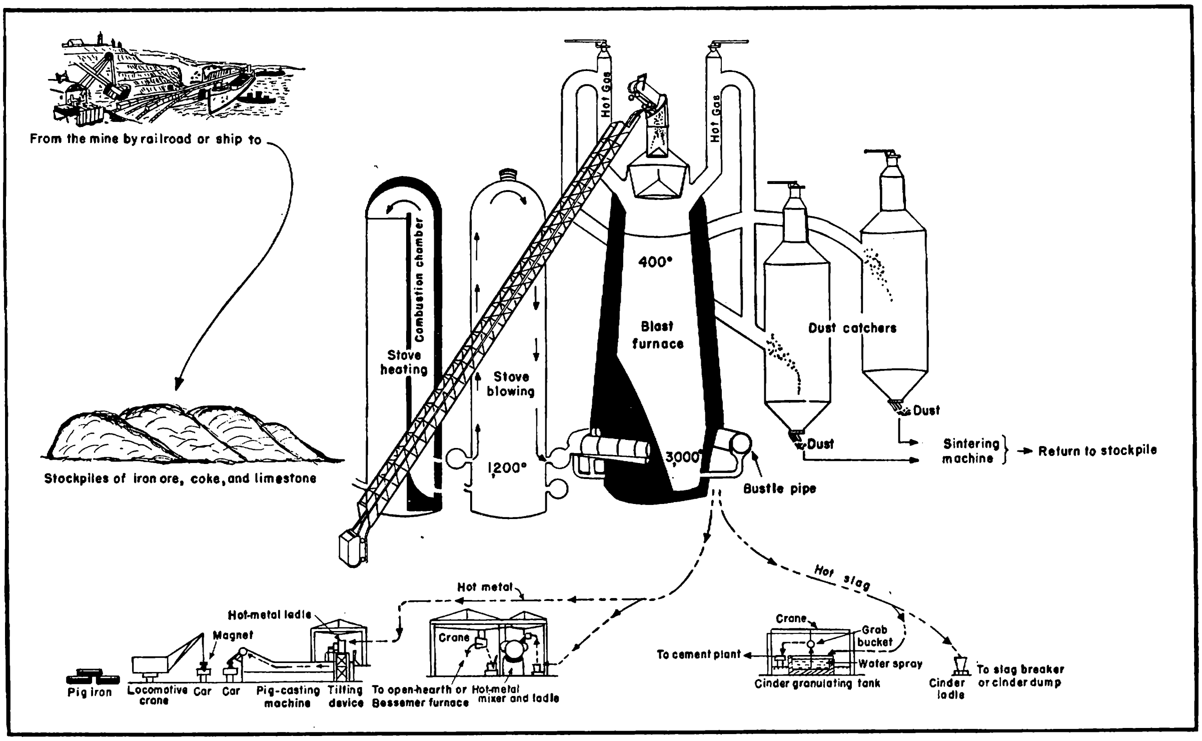

FIGURE 5.—Manufacture of Pig Iron.

OUTLOOK

Steel capacity, as of January 1, 1954, was 124,330,000 net tons; and, because of lowered demand during the year, it is unlikely that the immediate future will bring any great increase. Table 12 provides a projection over a longer period. An average increase of only 1.9 million tons capacity per year was estimated during the 20 year interval 1956–75. Since the average annual increase during the preceding 20 years was 2.2 million tons, the estimate of 160 million tons capacity for 1975 may be too conservative.

Iron-ore requirements as presented in table 12 are calculated from projected steel capacity. In table 13 the total domestic supply in 1975 is estimated at 115 million tons, which very nearly equals the 118 million tons produced in 1953. If domestic production and consumption have been correctly estimated, it would appear that, over the next 20 years, increasing imports will approximately equal increases in requirements. Such factors as loss of domestic ore reserves or plant because of inability to compete economically with foreign ores could of course upset this prediction.

IRON-ORE SUPPLY

Two important inferences may be drawn after examining tables 12 and 13:

(1) The indicated potential iron-ore supply is adequate to support United States iron and steel production at projected capacity rates until 1975.

(2) The proportion of iron ore from foreign sources increases to 37 percent by 1970. Two paramount problems require attention:

(*a*) National security as related to the increasing dependence upon foreign supplies.

(*b*) Adequate supply after 1975.

TABLE 12.—*United States crude-steel capacity and iron-ore requirements at full operation*

[Million gross tons]

	1950	1951	1953	1955	1960	1965	1970	1975
Steel capacity (ingots and steel for castings net tons Jan. 1)	99. 4	104. 2	118	126	130	140	150	160
Iron-ore requirements:								
Domestic use [1] gross tons	106. 6	117. 7	132	140	146	157	168	179
Export do	2. 5	4. 3	4	4	4	4	4	4
Total do	109. 1	122. 0	136	144	150	161	172	183

[1] Assumes that adequate supplies of scrap will be available.

TABLE 13.—*United States potential iron-ore supply apparent from present industry plans and other available information*

[Million gross tons]

	1950	1951	1953	1955	1960	1965	1970	1975
Domestic—Lake Superior:								
Direct ores:								
Open-pit	42. 8	53. 6	52	50	30	10	5	5
Underground	17. 5	18. 2	18	20	20	20	20	20
Concentrates:								
Taconite	--------	. 2	26	2	15	20	25	30
Other	18. 8	22. 0		30	35	35	30	25
Total Lake Superior	79. 1	94. 0	96	102	100	85	80	80
Southeastern States:								
Direct ore	5. 4	6. 0	8	7	7	7	7	7
Concentrates:								
Red ore	2. 2	2. 4		3	3	3	3	3
Brown ore								
Northeastern States (virtually all concentrates)	4. 6	5. 1	5	6	6	6	6	6
Western States:								
Utah (direct ore)	3. 1	4. 7	9	6	7	10	12	12
Other:								
Direct ore	1. 6	2. 6		3	3	3	3	3
Concentrates	1. 2	1. 0		1	2	2	2	2
Byproduct ore	. 6	. 6	. 6	1	1	2	2	2
Total domestic supply	97. 8	116. 4	118	129	129	118	115	115
Total concentrates included	27. 4	31. 3	38	43	62	68	68	68
Concentrates, percent	28	27	30	33	48	58	59	59
Foreign supply:								
Canada:								
Quebec-Labrador	--------	--------	------	5	15	20	30	30
Steep Rock	1. 9	2. 0	2	3	5	5	5	5
Michipicoten				1	2	2	2	2
Other				1	1	1	1	1
Latin America:								
Brazil	. 7	1. 0	------	1	2	2	2	2
Chile	2. 5	2. 8	6	-----	------	------	------	------
Venezuela	--------	. 6		6	13	15	20	20
Other	. 2	. 2	------	1	1	1	1	1
Africa:								
Algeria and Tunisia	. 7	. 6	1	1	1	1	1	1
Liberia and British West Africa	. 2	. 4	1	2	3	4	4	4
Europe:								
Sweden and Norway	2. 0	2. 5	2	2	2	2	2	2
Total foreign supply	8. 2	10. 1	12	23	45	53	68	68
Total iron-ore supply	106. 0	126. 5	130	152	174	171	183	183
Foreign ore, percent of total supply	8	8	9	15	26	31	37	37
Total foreign supply other than Canada	6. 3	8. 1	9	13	22	25	30	30
Foreign supply other than Canada, percent of total	6	6	6	9	13	15	16	16

TABLE 14.—*1950 production of iron ore, pig iron, and steel in important countries* [1]

[Thousands metric tons except per capita in kilograms]

Country	Iron ore			Pig iron [2]			Steel [3]			Ratio: steel-pig iron
	Amount	Percent of world	Per capita	Amount	Percent of world	Per capita	Amount	Percent of world	Per capita	
Algeria	2,573	1.05	292	-------	-------	-------	-------	-------	-------	-------
Australia	2,403	.98	293	[5] 1,101	0.83	134	[5] 1,400	0.75	171	1.271
Austria	1,859	.76	258	883	.66	123	947	.51	132	1.073
Belgium	46	.02	5	3,693	2.78	427	3,788	2.04	438	1.026
Brazil	[4] 1,900	.78	36	704	.52	13	764	.41	15	1.085
Canada [7]	3,309	1.35	239	2,260	1.70	163	3,070	1.65	222	1.358
Chile	2,976	1.21	511	12	.01	2	65	.03	11	5.417
China	n.a.		-------	[4] 1,022	.77	2	[4] 540	.29	1	.528
Czechoslovakia	[4] 1,600	.65	127	1,883	1.42	150	2,736	1.47	218	1.453
France [8]	30,203	12.33	729	9,526	7.16	227	10,548	5.67	252	1.107
Germany:										
West	10,882	4.44	228	9,480	7.13	199	12,121	6.52	254	1.279
East	[4] 328	.13	19	288	.22	16	1,155	.62	66	4.010
Hungary	368	.15	39	[4] 500	.38	53	1,022	.55	109	2.044
India	[4] 3,000	1.22	8	1,689	1.27	5	1,437	.77	4	.851
Italy	442	.18	10	570	.43	12	2,362	1.27	51	4.144
Japan	910	.37	11	2,286	1.72	28	4,848	2.61	58	2.121
Luxembourg	3,845	1.57	13,034	2,499	1.88	8,470	2,449	1.32	8,300	.980
Mexico	420	.17	16	249	.19	10	[4] 320	.17	13	1.285
Morocco	1,179	.48	118	-------	-------	-------	-------	-------	-------	-------
Netherlands	-------	-------	-------	454	.35	45	490	.26	49	1.079
Poland	790	.32	32	[4] 1,250	.94	50	[6] 2,305	[6] 1.45	92	1.854
Rumania	[4] 395	.16	24	[4] 243	.18	15	[4] 558	.30	35	2.296
Sierra Leone	1,185	.48	593	-------	-------	-------	-------	-------	-------	-------
Spain	2,079	.85	73	680	.51	24	779	.42	28	1.146
Sweden	13,927	5.68	1,990	848	.64	121	1,438	.77	206	1.696
Tunisia	758	.31	230	-------	-------	-------	-------	-------	-------	-------
Union of South Africa	1,189	.49	97	733	.55	60	755	.41	61	1.030
United Kingdom	13,145	5.37	263	9,785	7.36	196	16,555	8.90	331	1.691
United States	99,740	40.71	662	60,217	45.28	400	87,848	47.23	583	1.459
U. S. S. R	n.a.	-------	-------	[4] 19,500	14.66	98	[4] 27,000	14.52	135	1.385
Yugoslavia	[4] 800	.33	49	[4] 210	.16	13	[4] 420	.23	26	2.000
World (estimate)	245,000	-------	102	133,000	-------	55	186,000	-------	77	1.399

[1] Sources: Bureau of Mines Minerals Yearbooks and World Almanac (population).

[2] Including ferroalloys.

[3] Ingots and castings.

[4] Estimate.

[5] Fiscal year ended June 30, 1950.

[6] 1949 production not included in total.

[7] Includes Newfoundland.

[8] Includes Saar.

DOMESTIC IRON-ORE SUPPLY

The Lake Superior region will continue to be the principal source, with an estimated annual output of 100 million tons through 1960; however, during that period the United States will have lost one of its important strategic assets—the ability to expand and contract production in quick response to demand. Volume production from new sources in Canada and Venezuela will relieve the pressure for further expansion of domestic output. Nevertheless, despite the 1954 decline, demand is expected to remain strong enough to maintain domestic production at a high level, and the longer this high level is maintained, the more rapid will be the subsequent decline.

If the demand for domestic iron ore should increase soon as a result of interrupted foreign supply, the open-pit ores will respond again but with increasing difficulty and in decreasing quantities as the reserves decline; higher output for any particular year is always at the expense of the ability to expand in subsequent years.

Concentrates from lean ores are expected to make up the greater part of the Lake Superior production after the decline of open-pit ores and before concentrates from taconite expand beyond the 25-million-ton level. Reserve information on these ores is uncertain, and it is expected that output cannot be maintained high for a long period.

Southeastern States, principally Alabama, are not expected to increase their output beyond 10 million tons a year, because imported ore is counted upon to supply additional requirements. However, good possibilities exist for developing economic means to beneficiate the important reserves of low-grade materials and of discovering additional large tonnages of self-fluxing or partly self-fluxing ores similar to those now being used.

Northeastern States, already producing premium-grade iron-ore concentrates, are expected ot maintain an output approximating 6 million tons per year. A sustained high demand will bring a few additional small mines into production, but output in quantity from the many small deposits of siderite and brown ore awaits a considerable increase in the value of iron ore.

West of the Mississippi River the principal output has come from Texas, California, Utah, and Wyoming, although iron-ore deposits are known to occur in several other States. Distances, freight costs, and increasing population tend to encourage the growth of an iron producing industry, however, evaluation of the possibilities is difficult.

FOREIGN SOURCES

Inasmuch as it appears necessary for United States industry to go abroad for supplemental supplies of iron ore, it is fortunate that a good part of these supplies may be obtained from nearby deposits. Canada may be supplying nearly 40 million tons annually by 1975.

The Great Lakes-St. Lawrence Seaway is an important factor in the possible tonnage available from the Quebec-Labrador deposits. A target date of Spring 1959 has been set for opening the seaway to shipping.

Nearby, yet subject to open-sea transportation, are the rich iron-ore deposits in Venezuela. In magnitude they appear to be comparable to those of Quebec-Labrador. Security considerations favor the Canadian deposits, but year-round operation and premium grade favor Venezuela. The first shipment from Cerro Bolivar arrived at Morrisville, Pa., on January 20, 1954.

Chile, Peru, Brazil, and Cuba will provide additional supplies, but ores from Europe and Africa (with the possible exception of Liberia) will more probably find markets in Europe.

PROBLEMS

Chief among the problems of the industry are those stemming from the approaching depletion of readily accessible, high-grade domestic ores. The factors involved include:

(1) The problem of determining the safe minimum level of domestic production. This requires consideration of domestic and foreign ore reserves, economic, political, and technical data regarding foreign sources, and transportation to the United States during both peace and war.

(2) The problem of maintaining domestic production at or above the safe minimum.

Secondary problems include adjustments in transportation and furnace practice.

Development of furnace techniques to accommodate beneficiated, imported, and blended ores is hampered by the great cost of experimenting with large units.

Upgrading of ore and ore preparation, such as sintering, by adding another operation, lead to an additional cost, which must be defrayed at least in part by increased efficiency.

Expanding utilization of taconites introduces new problems to every phase of the industry. Greater tonnages of ore are required, and the ore is harder and must be finely ground, concentrated, and reagglomerated.

Beneficiation of various ores other than taconites, such as the laterites of Cuba, the brown ores of Texas, the ferruginous sandstones of Alabama, and the lean ores of Minnesota presents many problems.

Other raw materials—particularly fuels—must be supplied in proportion to the demand for iron ore. The problems of cost, quality, and transportation of these raw materials are increasingly difficult.

Development of economic methods of producing iron on a small scale would permit some decentralization of the industry and foster utilization of smaller deposits.

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JEWEL BEARINGS

By

Robert D. Thomson [1]

TINY JEWEL BEARINGS, so small that about 18,000 can fit in a thimble, have become a part of our everyday life. Millions are consumed each year in making instruments of measurement, regulation, and control—from the meter that measures with precision the flow of electricity into homes to the watch that accurately keeps the time of day. These bearings guide our civilian and military aircraft and are indispensable in military precision instruments.

Summary

Jewel bearings manufactured from synthetic sapphire and ruby have largely replaced those from natural minerals for use in watches and precision instruments.

Synthetic crystals (boule and rod) are made from aluminum oxide, which under intense heat acquires the physical and chemical properties of natural stones. The manufacture of jewel bearings involves meticulous detail, and as many as 60 operations are required to shape, drill, and polish boules into marketable jewel bearings. The work is done to close tolerances and requires skilled labor with high finger dexterity, muscular coordination, and patience; diamond powder of various specifications; and trade techniques of the industry, many of which are secret.

Virtually all of the domestic supply of jewel bearings is obtained from Switzerland. Cost differentials resulting from a large skilled labor force and lower wages in Switzerland compared with the United States hinder the domestic jewel-bearings industry. However, the United States industry has made progress in commercial production of those industrial jewel bearings that are larger and more amenable to mass-production methods. A jewel-bearing plant near Rolla, N. Dak., is being operated under the direction of Army Ordnance Corps to provide a nucleus of skilled workmen capable of instructing others if it becomes necessary to expand production of jewel bearings for full mobilization. Some domestic jeweled-watch companies have a nucleus of skilled personnel capable of instructing others in the manufacture of jewel bearings.

Jewel bearings have been stockpiled; however, the large variety of shapes and sizes required by industry makes it difficult to determine future requirements.

Methods of safeguarding domestic security include an adequate stockpile of certain standard jewel bearings and special tools and machinery, research on substitute materials, and a small standby domestic industry capable of expanding to full-scale production in an emergency.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

Jewel bearings are used where it is necessary to minimize friction and wear between small moving parts. The tiny bearings are shaped, drilled, polished, and cupped to make them usable. The materials normally used for making jewel bearings are natural sapphire, synthetic sapphire, and synthetic ruby. The manufacture of synthetic corundum was disappointing until shortly after the turn of the century, when a Frenchman, Verneuil, developed a process for producing low-cost boule. This soon was adopted by the Swiss watch industry as a raw material for use in manufacturing jewel bearings. Each year larger quantities were used until eventually natural sapphire was replaced entirely.

The manufacture of jewel bearings involves scrupulous attention to detail. In the past the United States has relied almost entirely on imports for its supply of jewel bearings; however, there are facilities in the United States for limited production of even the most intricate bearings, but high labor costs and a shortage of skilled workmen have hindered expansion of the United States industry.

RAW MATERIALS

Jewel bearings are made principally from natural or synthetic sapphire and ruby; in fact, synthetic sapphire and ruby bearings have largely replaced natural mineral bearings. Glass and synthetic spinel are used in making bearings where maximum hardness and shock resistance are not essential.

In the United States natural sapphire came principally from the American Gem Co. mine at Philipsburg, Mont., and Perry Schroeder Mining Co. mine at Helena, Mont. Several domestic manufacturers of jewel bearings used the stones produced by these companies; but, with domestic production of synthetic corundum in the early 1940's, the majority of the firms began using synthetic material only. Each year the demand for natural stones decreased, and the two companies ceased operations in 1943.

In manufacturing synthetic sapphire boule, the principal raw material is aluminum oxide in the form of a fine white powder. Small quantities of metallic oxide can be added for pigmentary purposes—for instance, 2.5 percent of chromic oxide for ruby and about 10 percent of magnesium oxide for spinel.

Synthetic corundum has essentially the same chemical and physical properties as natural corundum but is of greater purity and homogeneity (*7, 10*).[2] Synthetic stones are free from flaws and less expensive than the natural material and can be utilized to a further extent. Natural stones are irregular and often contain cracks, layers, or inclusions that cause difficulty in processing. Flawless natural stones usually are made into gems and because of their high price can be used only for special purposes by the jewel-bearings industry. Synthetic stones have a lower coefficient of friction than natural ruby (natural, 0.287; synthetic, 0.158).

There is no shortage of raw materials for producing the boule or rod. The process requires large quantities of cheap electric power to make hydrogen and oxygen gas, which are used as fuel in the furnaces. The power derived from hydroelectric sources in Switzerland costs only about 9 mils per kilowatt-hour.

TECHNOLOGY

The production of jewel bearings can be discussed in two phases: (1) The production of the boule or rod and (2) the production of the jewel bearing.

The production of a synthetic corundum boule by the Verneuil (flame-fusion) process consists in growing a single crystal in a simple, inverted, oxyhydrogen blowpipe. Very pure, finely ground aluminum oxide is charged at the top of a small furnace and passes through a hydrogen-oxygen flame in a combustion chamber. The burning hydrogen-oxygen gas creates a temperature of approximately 4,000° F., and the molten alumina solidifies on a fire-clay peg as a carrot-shaped crystal. This crystal or boule is usually 0.5 to 1.0 inch in diameter, 2 to 4 inches long, and weighs 75 to 250 carats. The boule forms in 2 to 4 hours and cools in about 1 hour. Raw boule will produce 4 to 7 watch jewels or 1 industrial jewel per carat.

The United States boule-manufacturing process differs from the European in that small, cylindrical rods are grown, in addition to the

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[2] Italicized figures in parentheses refer to items in the bibliography at end of this chapter.

carrot-shaped boule. The principal use of rods is in manufacturing end stones. However, most jewel bearings used in the United States are made from boule because of lower overall cost.

As many as 60 operations are required to shape and polish a boule to form jewel bearings to extremely close tolerances, usually in the order of 0.0001 inch. The production stages can be divided into three main groups: (1) Rough preparation—sawing, rondeling, and flat grinding; (2) intermediate steps—drilling and opening; and (3) finishing operations—turning down, polishing, bombéing, oliving, recessing, and chamfering (*3*, *10*). Essentially, the boule is split longitudinally, sawed into blanks, ground into rondelles, drilled, cut to various shapes, and polished and glazed into marketable jewel bearings. The customary sequence of operations is shown in figure 1.

The industry has developed, where possible,

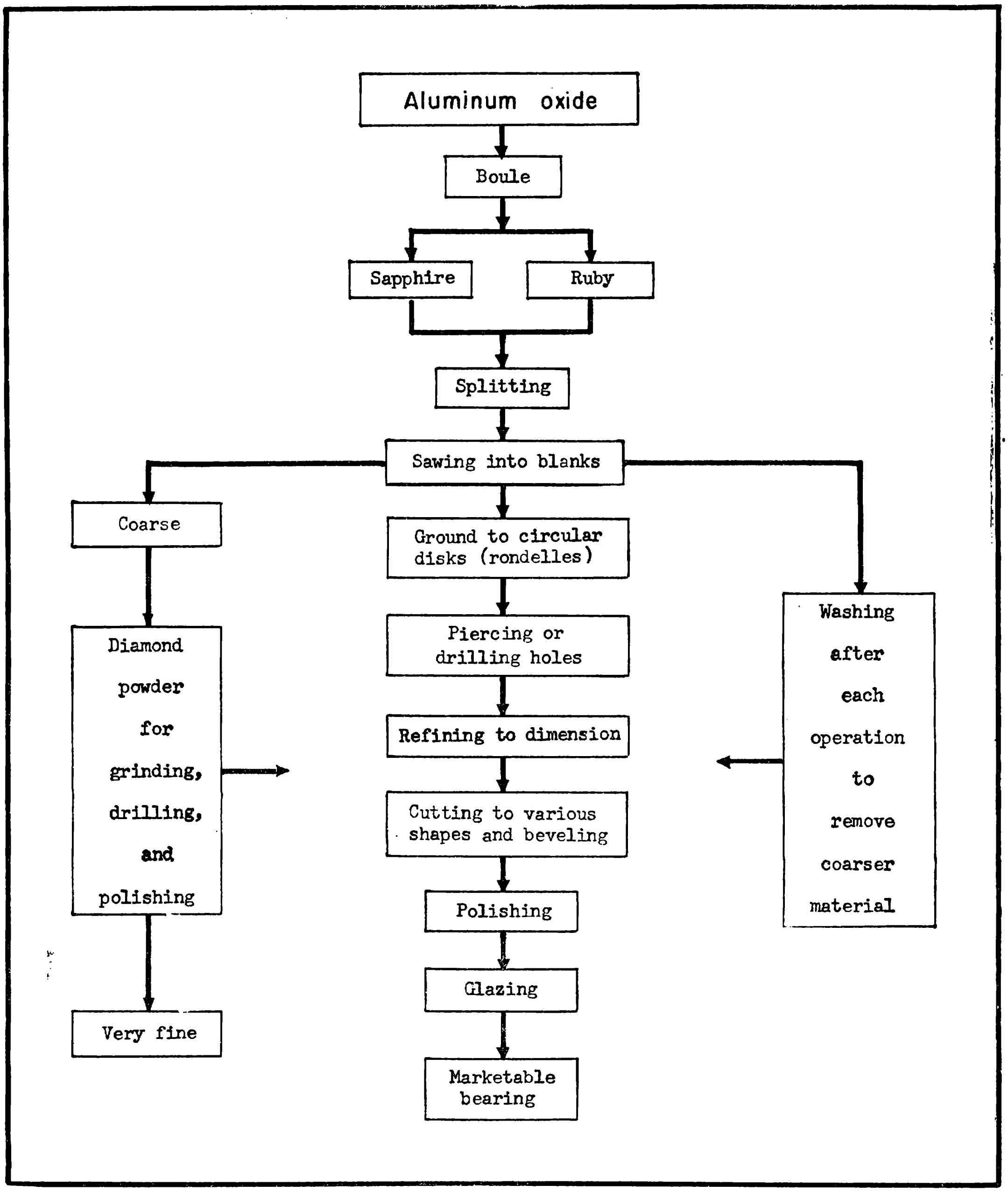

FIGURE 1.—Sequence of Operations in Manufacturing Jewel Bearings.

mass-production techniques, where numerous stones can be machined in a single operation. However, certain stages must be conducted on a single-operation basis, such as piercing and recessing. The special machines used for the different stages depend on personal preference, vary from factory to factory, and usually are manufactured by the Swiss producer of jewel bearings. In 1953 there were no manufacturers of jewel-bearings equipment in the United States. Although production has been mechanized as much as possible, manual methods must be used for many operations, such as fixing the stones, washing, and inspection.

Diamond powder is an essential item in the production of jewel bearings. It is second to labor as a cost factor and amounts to about 25 percent of the production costs for industrial bearings and 5 to 10 percent of the costs for producing watch bearings. The consumption of diamond powder used either embedded in metal bodies or in loose form on laps depends on the arrangement of the sawing blade, operating conditions, and machinability of the synthetic corundum. One empirical formula indicates that diamond consumption by the Swiss industry, in carats per jewel, equals the jewel diameter, in millimeters, divided by 1,000 (*2*). Table 1 shows the percentage of diamond powder consumed in each of the processing operations (*2*). Only small quantities of the diamond powder are salvaged, which necessitates rigid controls on the quantities of powder used. The Swiss industry consumes about 1 carat of diamond powder to produce 1,000 watch jewels and about 5 carats per 1,000 industrial jewels. If all of the 1953 domestic consumption of jewel bearings had been produced domestically, about 200,000 carats of diamond powder would have been required.

SHAPES AND SIZES

Jewel bearings are made in a great variety of shapes and sizes for consumption as watch or industrial bearings. Little progress has been made in establishing standard shapes and sizes, and the majority of the jewels supplied are for special applications (*2*). The selection of the proper raw material for producing the jewel bearings depends on the ultimate use. Synthetic sapphire is preferred for manufacturing industrial jewel bearings, whereas synthetic ruby is preferred for watch jewels. Glass bearings have been used as "vee" jewels in place of jewel bearings for light loads and low-shock-resistance applications.

Watch jewels are used on the parts that move the fastest and require the finest adjustments. The most common varieties of watch bearings in a 7-jewel watch are pallet stones (1 on each arm of the Y-shaped lever), roller pin (on a roller table affixed to the shaft of the balance wheel), cap jewel (1 each on the upper and lower end of the balance staff), and hole jewels (1 each on the upper and lower end of the balance staff). An estimated distribution of the number of the various kinds of jewels used in timepieces is shown in table 2.

TABLE 1.—*Consumption of diamond powder in the production of ring jewels*

Main operation	Consumption of diamond powder (percent)	Individual operation	Consumption of diamond powder (percent)
Rough preparation	43	Sawing	25
		Plain grinding	9
		Rondeling	9
Intermediate steps	19	Piercing	7
		Enlarging	12
Finishing operations	38	Turning down	5
		Chamfering	6
		Recessing	3
		Polishing recess	1
		Plain polishing	12
		Bombé polishing	10
		Angling and oliving	1
	100		100

The principal industrial jewel bearings are: Cup (nonperforated disk in shape of cup which holds oil), vee (nonperforated circular disk with conical indenture in one side), ring (perforated disk or hole jewel), and cap (flat-surface endstone). Industrial jewels require a larger quantity of raw materials than the watch jewels.

PRODUCTION

The jewel-bearings industry in the United States consists of companies having the production of jewel bearings as their principal function and of the jeweled-watch industry. Essentially the industry is composed of firms that produce boule or rod; that produce finished jewel bearings directly from boule or rod; that adapt, finish, and mount imported jewels into metal rings or bushings; and that assemble or mount jewel bearings, essentially imports. The major portion of the commercial firms fall into the last two categories and are concerned principally with finishing and mounting imported jewel bearings, while others are concerned only

TABLE 2.—*Kind and number of jewels used in timepieces*

Kind of jewel	Timepiece			
	7-jewel	15-jewel	17-jewel	21-jewe
Hole	2	10	12	12
Cap	2	2	2	6
Pallet stone	2	2	2	2
Roller pin	1	1	1	1

TABLE 3.—*Salient statistics of the jewel-bearings industry in the United States, 1945–53*

[Number of jewel bearings]

	1945	1946	1947	1948	1949	1950	1951	1952	1953
Production:									
Blanks	1, 058, 804	271, 500	600, 700	680, 400	249, 600	795, 400	1, 200, 503	1, 907, 301	6, 043, 886
Finished jewels [1]	[2] 1, 232, 471	[2] 1, 873, 906	[2] 2, 626, 968	[2] 2, 576, 095	2, 725, 103	3, 327, 206	9, 876, 654	10, 637, 206	15, 666, 908
Consumption:									
Blanks	5, 096, 135	7, 392, 663	4, 816, 626	7, 503, 199	6, 678, 922	7, 008, 289	11, 415, 514	9, 062, 893	7, 939, 130
Semifabricated jewels	921, 061	1, 878, 297	2, 436, 150	1, 729, 100	1, 603, 900	3, 331, 500	7, 884, 500	1, 892, 000	1, 900, 000
Finished jewels [1]	38, 491, 139	50, 731, 953	65, 226, 137	66, 212, 629	68, 322, 111	71, 126, 700	85, 030, 037	77, 311, 999	70, 936, 923
Shipments:									
Blanks	655, 896	267, 244	55, 700	125, 400	29, 100	85, 400	75, 503	5, 391	8, 189, 821
Semifabricated jewels	58, 313	21, 236	3, 189	2, 069	1, 771	2, 414	561	1, 439	30, 000
Finished jewels [1]	[2] 4, 830, 375	[2] 13, 066, 857	[2] 8, 835, 021	28, 816, 351	24, 645, 548	6, 976, 608	14, 031, 386	28, 795, 001	36, 772, 885
Stocks on hand Dec. 31:									
Blanks	5, 545, 069	1, 469, 491	5, 490, 286	7, 297, 087	7, 684, 765	5, 796, 014	2, 618, 650	4, 327, 957	1, 413, 951
Semifabricated jewels	2, 489, 378	929, 294	578, 105	405, 225	243, 454	529, 540	710, 479	1, 054, 886	2, 134, 040
Finished jewels [1]	53, 224, 685	42, 468, 092	52, 927, 900	72, 945, 750	98, 213, 655	107, 432, 348	97, 390, 081	104, 169, 041	97, 545, 593

[1] Includes finished jewels made from glass.

[2] Does not include jewels made from glass.

with the production of glass bearings. Major obstacles to the establishment of a domestic industry have been high labor costs, shortage of skilled laborers, shortage of specialized machinery or a domestic firm producing them, lack of funds to conduct research and develop programs necessary to build a domestic industry competitive with foreign producers, and competition from Swiss bearings of a higher quality, lower cost, and plentiful supply.

Domestic production of jewel bearings supplies only a small portion of the annual domestic consumption. Essentially, no watch jewels are made in the United States by private enterprise, except for a small quantity of pallet stones. However, hole and cap jewels suitable for use in watches were produced in 1954 at the Government-subsidized Turtle Mountain Ordnance Plant. The only synthetic sapphire jewel bearings manufactured domestically in any appreciable quantities are cup jewels. Sizable quantities of "vee" jewels are produced annually, but these are mainly glass bearings.

In 1953, 10 companies and the Turtle Mountain Ordnance Plant reported production of jewel bearings, including bearings made from glass, as shown in table 3. However, the major portion of this production was Swiss jewel bearings, polished and mounted in the United States, and glass "vee" bearings. Geographically the jewel bearings were reported as manufactured at plants at Elgin, Ill.; Waltham and West Lynn, Mass.; Newark, Perth Amboy, and Trenton, N. J.; Woodside, N. Y., Rolla, N. Dak.; and Lancaster and Morrisville, Pa.

Synthetic sapphire and ruby crystals are manufactured by Linde Air Products Co., a division of Union Carbide & Carbon Corp., New York, N. Y., at plants in Chicago, Ill., and East Chicago, Ind. Linde Air Products Co., with a capacity sufficient to supply all United States requirements, is equipped to manufacture both boule and rod (*11*). In 1953 the crystals were sold to the fabricating trade for manufacture into gem stones and industrial articles, but virtually none of Linde Air Products Co. product was used in the manufacture of jewel bearings.

CONSUMPTION

Jewel bearings are used to reduce friction and to make a hard, smooth surface that will resist the wear caused by pivot action in precision instruments and similar devices and are essential to the life and accuracy of the instrument. Before 1940 jewel bearings were used only in electrical measuring instruments and timepieces. The trend to increased instrumentation which began in World War II has continued. Jewel bearings find universal application in watches and other timepieces, gages, meters, indicators, sensitive scales and balances, textile thread guides, fire-control devices (range finders, etc.), and other instruments. Jewel bearings are used widely in civilian and military equipment.

Consumption of blanks and finished jewels is shown in tables 3 and 4. Forty-one companies in 12 States reported consumption of finished jewels in 1953 (table 5). About 38 percent of the number of jewels used in the manufacture of end-use products was consumed by 6 companies in New Jersey and 8 companies in New York and Pennsylvania.

FOREIGN TRADE

The major portion of the United States supply of jewel bearings has been imported. Switzerland is the world's principal source of jewel bearings, both for watches and for industrial uses. Some jewel bearings have been produced intermittently in Italy, France, Germany, Japan, and Canada, and during World War II Russia and England produced limited quantities.

Table 4.—*Consumption and shipments of finished jewels*[1] *in the United States, 1953, by uses*

Use	Consumption		Shipments	
	Quantity (number of jewels)	Market value	Quantity (number of jewels)	Market value
Vees:				
Glass	6, 702, 462	$957, 578	8, 179, 633	$357, 575
Other	2, 186, 962	356, 035	6, 203, 609	735, 787
Instrument rings	21, 805, 016	1, 340, 316	10, 713, 766	1, 469, 547
Watch rings	7, 027, 528	406, 145	788, 617	88, 310
Cups	6, 460, 485	509, 915	4, 527, 216	403, 729
Endstones	7, 079, 447	165, 918	3, 378, 423	178, 642
Caps	10, 196, 097	439, 450	----------	----------
Hole	1, 467, 187	109, 101	2, 108	475
Pallet stones	5, 127, 097	163, 857	401, 635	26, 042
Jewel pins	1, 700, 063	50, 623	647, 500	78, 200
Locked stones	----------	----------	1, 869, 000	224, 280
Roller pins	674, 905	17, 008	78	12
Other	509, 674	122, 322	61, 300	36, 644
Total	70, 936, 923	4, 638, 268	36, 772, 885	3, 599, 243

[1] Includes finished jewels made from glass.

The Swiss jewel-bearings industry is very complex, and only 1 or 2 firms are integrated and equipped to do all of the operations necessary to produce a finished product from the alumina to the marketable bearings. Most manufacturers perform only 1 or 2 of the specialized operations and contract to have the remaining operations done for them. The manufacturing industry in Switzerland for the production of watch jewels is controlled by a closely knit union organization and the foreign trade by a manufacturers' association. The trade associations in the watch-jewel branch of the industry exercise close control on wage rates, number of employees, kind and type of equipment to be used, prices, and sales and export quotas.

Over 150 companies make up the Swiss watch-jewel industry, consisting of a few large producers and many small firms (cottage industry) that are highly specialized in one phase of the production. Nearly all of the labor is skilled, and three-fourths of the labor supply is women because they are better adapted to the painstaking skilled work. One to three years of apprentice training is required, and the number of skilled workers is controlled by the trade associations.

Table 5.—*Consumption of finished jewel bearings*[1] *in the United States, 1953, by States*

State	Number of consumers	Quantity (number of jewels)
Connecticut	3	2, 186, 505
Massachusetts	6	3, 727, 149
Michigan	2	39, 751
New Hampshire	3	880, 859
New Jersey	4	6, 720, 792
New York and Pennsylvania	8	20, 270, 120
Ohio	5	2, 415, 315
Other States [2]	10	34, 696, 432
Total	41	70, 936, 923

[1] Includes finished jewels made from glass.

[2] Includes Illinois, Indiana, Minnesota, and Wisconsin.

Table 6.—*Jewel bearings (not assembled in units) imported for consumption in the United States, 1940–53*[1]

Year	Number	Value
1940	98, 771, 042	$1, 831, 007
1941	92, 547, 236	2, 007, 012
1942	88, 650, 286	2, 770, 866
1943	65, 166, 357	2, 601, 256
1944	38, 324, 422	1, 376, 675
1945	36, 340, 820	1, 467, 547
1946	58, 896, 065	2, 076, 391
1947	114, 089, 168	4, 016, 072
1948	138, 229, 491	5, 614, 287
1949	140, 742, 977	5, 117, 341
1950	87, 939, 766	3, 737, 979
1951	92, 396, 053	3, 965, 983
1952	98, 021, 914	4, 226, 948
1953	89, 892, 637	3, 708, 027

[1] U. S. Department of Commerce.

The industrial-bearings industry represents about one-fifth of the total value of the output of jewel bearings and is not subject to control by an association. The Swiss industrial-bearings industry is concentrated in 12 plants, of which 4 produce over 80 percent of the output. Industrial-bearings labor can be less skilled than the watch-jewel-bearings labor, and thus the industry is more amenable to expansion. Industrial bearings cost more to produce than watch-jewel bearings, not because they are larger, but because more diamond powder is required in their manufacture. The average cost ranges from 8 to 9 cents each for the smaller sizes and somewhat higher for the larger units.

TARIFF

The Tariff Act of 1922 established a 10-percent ad valorem duty on jewel bearings, and this rate was still in effect in 1954. The act defines jewel bearings as "jewels, suitable for use in any movement, mechanism, device, or instrument, or in any meter or compass." Duty is not sufficient to offset the higher cost incurred in the production of domestic bearings as compared with the cost of Swiss bearings production. The tariff rate on synthetic sapphire boule is 30 percent ad valorem.

WORLD WAR II PROGRAMS

With the fall of France in June 1940 the United States virtually was cut off from its supply of Swiss jewel bearings. The German Government compelled Swiss exporters to obtain transit permits but was reluctant to issue any. Because of pressure from the German Government, Switzerland enforced rigid mailing restrictions on envelopes and packages going to the United States in an effort to restrict further the export of jewel bearings. However, quantities of jewel bearings from Switzerland were smuggled into the United States, but these receipts were erratic and undependable. The stoppage of imports through regular channels alerted the United States to the precarious position of precision-instrument-manufacturing operations. Steps were taken by the United States Government to develop and increase the domestic production of boule and jewel bearings. In October 1941 the Defense Plant Corporation issued a contract for production of 10 sets of jewel-bearings-manufacturing machinery; and, with subsequent contracts, a total of 25 sets of ring and vee machinery was produced; these sets were leased to the jewel-bearings manufacturers. Industry also anticipated the vulnerability of the Swiss jewel-bearings supply. As a result of privately sponsored research, additional jewel-bearings-making equipment was designed and built. Furthermore, full-scale production facilities for boule were built, capable of satisfying the entire domestic requirement. Certain companies were assisted through Government funds advanced to aid in establishing new plants.

The military requirements for jewel bearings increased to such proportions that Conservation Order M–50 was issued in January 1942 placing jewel bearings under complete allocation. Contracts for the production of jewel bearings were not issued until the latter months of 1942. During the transition period—from meager production before World War II to volume production—manufacturers utilized stockpiles of low-price jewels previously received by "smuggling" methods. By mid-1943, the domestic jewel-bearings industry reached mass-production levels. Jewels made in the United States were higher in price because of training costs, higher labor costs, and expensive machinery. Industry was reluctant to consume large quantities of the domestic-produced jewels while stocks of less expensive imported jewels were available. By the time the domestic industry achieved mass production, requirements had decreased considerably, and imports increased. In 1 year the situation changed from a dire need for a mass domestic industry to the question of what to do with the domestic industry as the supply, principally from imports, greatly exceeded requirements.

Within this short period, the United States industry had built machinery; trained personnel in the delicate, precise, and meticulous details of jewel manufacture; and developed feasible production methods, expanding from a small nucleus of several jewel-bearings companies and the jeweled-watch industry. In spite of extreme difficulties, domestic industry produced excellent-quality jewel bearings from domestic boule. The United States industry contributed greatly to the war effort.

During World War II, industry tried to develop suitable substitutes for jewel bearings. Many materials were tested and a few substitute materials, such as garnet, glass, and metal-on-metal, were used, all inferior to corundum and limited in application. Industry developed a glass jewel that could be molded to close tolerances. As a result, the glass "vee" became an important factor as a substitute material but was restricted to use where moving elements weigh less than 1 gram and in applications where they are not subjected to severe vibration or shock.

POST-WORLD-WAR ACTIVITIES

In line with increased automation and more complex instrumentation in civilian and military equipment, the United States Air Force realized the need for a substantial domestic industry and contracted with Elgin National Watch Co. for a research project on the methods of jewel manufacturing *(11, 12)*. Under the contract, Elgin was to investigate the development of automatic methods for producing jewel bearings. The actual production to be developed under the contract was a completely integrated pilot production line consisting of 26 different types of automatic or semiautomatic machines. By 1954 Elgin had constructed at least 1 each of the 26 different types of the machines. Reports show that under this contract progress was made in reducing skilled labor and labor input and increasing the automatic machine operations for the production of jewel bearings *(11)*.

The Army also has been greatly concerned with the problems of domestic jewel bearings, and in June 1952 the Army Ordnance Corps awarded a contract to Bulova Watch Co. to establish and operate a jewel-bearings plant at Rolla, N. Dak. The plant, known as Turtle Mountain Ordnance Plant, utilizes the skill (finger dexterity and muscular coordination) of the Chippewa Indians from the Turtle Mountain Indian Reservation. The purpose of this plant is to provide a nucleus of skilled labor for the production of finished jewel bearings and develop and perfect conventional production methods. Efforts of about 90 persons have been concentrated on the production of watch-hole jewels, one of the more difficult jewel bearings to produce. The cost of jewel bearings produced in this plant has been estimated to be about three times that of imported jewel bearings. It is anticipated that this program will provide a labor supply capable of instructing others to expand a domestic jewel-bearings industry as required in periods of emergency.

Jewel bearings are in Group I on the list of strategic and critical materials. Three different classifications—(1) jewel bearings, instrument jewels, except vee jewels; (2) jewel bearings, sapphire and ruby vee jewels; and (3) jewel bearings, watch and timepiece device jewels—have been stockpiled, but purchases were suspended in 1950.

Jewel bearings are readily stockpiled because they require a small storage space and are not subject to deterioration. The multiplicity of shapes and sizes makes it difficult to foresee what shape or size would be required to establish realistic stockpile objectives. It has been reported that certain types of jewel bearings purchased during World War II now are obsolete. However, a segment of safety can be gained through a stockpile adequate in size to permit mass development of the domestic industry in an emergency. Representatives of industry have expressed the opinion that it would be of value to stockpile special tools and machinery for producing jewel bearings instead of the finished jewel bearings unless standardization can be adopted. A limited number of the basic tools and machinery could be stockpiled, but without potential productive capacity for utilizing the machinery with a minimum of delay or laborers skilled in producing jewels, the full value of the machinery could not be realized.

OUTLOOK

World War II and the years succeeding have shown a definite trend toward more complex instrumentation, with increased dependence on jewel bearings. More and more military and civilian instruments are using jewel bearings. High labor costs have accelerated the use of instrumentation, resulting in a sharp increase in the demand for industrial jewel bearings. New applications for jewel bearings will continue as long as new equipment is being designed. Two significant new uses for synthetic corundum have recently developed and may have considerable importance. The first is the manufacture of ball and roller bearings, utilizing spheres and cylinders of synthetic ruby or sapphire. The second use is the manufacture of phonograph needles, windows for infrared equipment, and mortars and pestles.

So far, a commercially successful large-scale jewel-bearings industry has not been established in the United States. It is unlikely that a domestic commercial organization can be developed that will be competitive with the Swiss industry because of high production costs. During peacetime, it is anticipated that the supply, principally imports, and requirements will remain in balance. If an emergency should occur, the situation might become extremely critical before domestic mass production could be developed. Companies now producing or processing jewels and the jeweled-watch industry have maintained, in some degree, personnel and activities that could serve as a nucleus for rapid expansion in an emergency. However, it appears that, without subsidization of the existing industry or expansion of facilities comparable to the Turtle Mountain Ordnance Plant in peacetime, a productive capacity adequate to meet expanded requirements might not be obtained rapidly enough in an emergency.

If at any time the supply of jewel bearings from Switzerland again were curtailed, the United States would be forced to resort to a previously acquired stockpile, substitutes, or a domestic industry.

PROBLEMS

The major problems of a United States jewel-bearings industry are (1) the almost complete lack of production facilities and (2) the difficulty of maintaining a standby industry in the United States against competition of the well-established Swiss industry. Major obstacles to establishment of a domestic industry are high labor costs and a shortage of skilled workmen. Another problem of paramount importance is the impracticability of establishing realistic stockpile objectives for the vast number of sizes and styles now being used or for those that may be in demand in time of full mobilization. The prospect of relying on Swiss imports in emergencies presents the biggest problem facing the American jewel-bearings industry.

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KYANITE AND RELATED MINERALS

By

Brooke L. Gunsallus[1]

THE RELATIONSHIP of refractories to modern heat-utilizing industries is analogous to the classic race between armorplate and projectiles, except that refractories can never win. When better refractories are made in any price range and service class, industrial applications are already waiting to strain the capabilities of the new refractory to the utmost. The mullite-forming materials—kyanite, sillimanite, andalusite, dumortierite, topaz, and synthetic mullite—constitute an important type of refractory used in the metallurgical and glass industries.

Summary

Mullite is rarely found in nature, but when any aluminum silicate minerals (kyanite, andalusite, sillimanite, dumortierite, or topaz) are heated to appropriate temperatures they are converted to mixtures of mullite and a form of free silica. Mullite of any desired purity also may be produced by sintering or fusing mixtures of alumina and either kaolin or silica in stoichiometric proportions. In common with all refractories, however, correct chemical composition is not the only essential criterion. Mineralogical constitution and grain structure also have vital significance in their influence on the ability of the raw materials to produce finished refractories.

Before World War II most of the kyanite-mullite refractories were made using imported Indian kyanite as the main constituent. However, by 1953 economic conditions and the uncertainty of obtaining Indian kyanite and other high-grade materials from abroad had increased the demand for high-grade domestic aluminum silicate minerals and synthetic mullite.

The results of prospecting and exploration to date indicate the improbability of finding adequate domestic sources of massive mullite-forming minerals. Disseminated deposits from which mullite-forming minerals can be obtained only as fine concentrates provide a product that can be used directly for some refractory purposes. Two companies, in Virginia and South Carolina, produce kyanite concentrates. Through application of present techniques of pelletization, however, such concentrates can serve some of the purposes for which massive materials would normally be used.

The development of synthetic mullite offers the best source for fulfilling future needs. Furthermore, synthetic material can be produced by sintering from nonstrategic raw materials, comparable in quality and at a cost about equal to mullite produced from massive Indian kyanite. A superior product is produced by fusing, but at a considerably higher cost.

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[1] Commodity-industry analyst, Bureau of Mines.

Domestic reserves of kyanite and sillimanite from disseminated ores are large, on the order of tens of millions of tons, but domestic reserves of massive kyanite are lacking. Reserves of low-iron siliceous bauxite are adequate for short-range requirements, but data on the quantity and quality are meager.

Reserves of good-quality Indian kyanite are assumed to be adequate.

Domestic consumption of kyanite and synthetic mullite has ranged from 30,000 short tons in 1950 to 45,000 short tons in 1953, all for mullite refractories. About 90 percent of the mullite refractories were used by the metallurgical and glass industries and the balance for various industrial applications.

The outlook is for expanded use of mullite refractories as the temperatures at which metallurgical processes are conducted become higher.

The problems of the kyanite-mullite refractories industry include lack of more adequate raw-materials specifications, lack of enough low-cost electrical energy for future expansion of fused synthetic mullite, and lack of detailed information on the quantity and quality of deposits of mullite-forming materials. In addition, there is need to develop new uses for domestic kyanite concentrates and to develop improved beneficiation methods to recover kyanite from its ores.

BACKGROUND

MINERALOGY

Kyanite, andalusite, sillimanite, dumortierite, and topaz are aluminum silicate minerals that may be used to produce mullite-containing refractories. The first three minerals, in their pure forms, have the formula $Al_2O_3 \cdot SiO_2$ but differ in crystal structure; the formula for dumortierite is $8Al_2O_3 \cdot 6SiO_2 \cdot B_2O_3 \cdot H_2O$; the formula for topaz is $2Al_2O_3 \cdot 2Al(F \cdot OH)_3 \cdot 3SiO_2$. All these minerals all bave the property, when heated to the proper temperatures, of transforming to mullite ($3Al_2O_3 \cdot 2SiO_2$ and SiO_2). At or below the temperature of inversion, the boron of dumortierite and the fluorine of topaz are volatilized. The SiO_2 appears as a glass or some form of quartz, depending upon how much flux is present and upon the heat treatment. The mineral mullite rarely occurs in nature.

The most important of the kyanite group of minerals for refractory purposes at present is kyanite. Kyanite exhibits considerable expansion when heated between 2,200° and 2,400° F. and changes from a specific gravity of 3.6 to 2.6. Kyanite is completely changed to mullite and glass at about 2,450° F. Because of the great expansion of raw kyanite when heated to its transformation point, it is common practice to precalcine the kyanite ore before using it in refractory bodies.

TECHNOLOGY

Until recent years, most of the mullite-forming refractory material used in the United States was obtained by calcining high-grade massive Indian kyanite. Material having the favorable characteristics of good-grade massive Indian kyanite has not been found in appreciable quantities in the United States. The present domestic output comes from disseminated deposits in South Carolina and Virginia and is recovered as flotation concentrates. These concentrates cannot be considered as a suitable replacement for the Indian kyanite for two reasons: (1) The concentrates are too fine grained for refractory brick without relatively expensive nodulation; and (2) the calcined material is quite friable and has less strength than desirable.

Mullite obtained from the other minerals of the kyanite group would be satisfactory for manufacturing mullite refractories, but so far no large deposits of massive ore have been discovered. From 1939 through 1946 small quantities of andalusite from California and dumortierite from Nevada were produced from small deposits by selective mining.

During the last few years, research by the Bureau of Mines and private industry, utilizing Western Hemisphere raw materials, has resulted in development of synthetic mullite products equal or superior to those derived from high-grade Indian kyanite. Synthetic mullite is manufactured by sintering or fusing intimate mixtures of siliceous and aluminous materials to form products consisting of mullite and free silica.

The highest quality synthetic mullite is produced by using Bayer-process alumina made from Caribbean bauxite and siliceous materials either by the fusing process or by sintering at high temperatures.

The suitability for refractories of all these raw materials, whether obtained from natural ores or from synthetic processing, depends on grain size, density of the grain, and performance of the finished refractories under operating conditions.

Sintered synthetic mullite is produced by sintering properly proportioned mixtures of aluminous and siliceous raw materials, such as bauxite, alumina, kaolin, kyanite concentrates, and flint or other forms of silica, in periodic or rotary kilns and is a microcrystalline, granular product. The cost of sintered synthetic mullite is comparable to that of mullite produced from imported kyanite. Most sintered synthetic mullite is being produced from low-iron siliceous bauxite occurring in the southeastern United States.

A completely fused material is produced from exceptionally high-quality raw materials in an electric-arc furnace. When crushed and sized, mullite produced by this process is a superior product because of its crystalline structure, uniform composition, and small content of impurities. The cost of fused synthetic mullite is much higher than that of sintered synthetic mullite or mullite produced by calcining imported lump kyanite, but for certain applications its superior properties have proved it to be economical.

Sintered synthetic mullite is being produced in the midwestern United States in a plant formerly used to manufacture high-grade fireclay refractories. Any plant of this type can be converted at low cost to produce sintered synthetic mullite refractories. Some sintered synthetic mullite is being produced as an additional product, and on a small scale, in other plants that formerly produced only high-grade refractory fire brick.

One of the accepted methods of producing sintered synthetic mullite is as follows: Domestic low-iron siliceous bauxite is ground, screened, pugged with water, and extruded through a die to form a column. The extruded column is cut into convenient lengths, known as dobbies or slugs, which are calcined in periodic or rotary kilns. The calcined material is crushed, ground, and screened to specifications based on desired end products. The prepared material is bonded and formed into brick and shapes by power pressing. In most operations a few brick and shapes are formed by hand. After drying, the mullite brick and shapes can be fired in several types of kilns.

A company in the eastern United States has reported that it is producing a very good grade of sintered synthetic mullite, using mixtures of siliceous and aluminous materials. The dobbies or slugs are produced in the conventional way and fired in a pebble-type heat-exchange kiln with a capacity of 120 tons. The time required to set, fire, cool, and draw this kiln is about 1 month. The minimum temperature used in sintering is 3,200° F., using fuel oil. According to the reported chemical analysis and pyrometric-cone equivalent, this material approaches the range of Government specifications set up for electrically fused synthetic mullite.

The development of synthetic mullite assures the United States of self-sufficiency in raw materials for mullite-refractories manufacture, and in 1954 kyanite was removed from the list of critical and strategic materials for stockpiling. The stockpile objective for kyanite was reached in 1952.

RESERVES

A small deposit of high-grade fibrous kyanite occurs on Willis Mountain, central Virginia. Estimated reserves of high-grade lump kyanite are 5,000 tons. This represents about 1 month's supply at the 1954 rate of consumption. No other deposits of massive kyanite are known in the United States.

Reserves of massive Indian kyanite, 55–59 percent grade (Al_2O_3), are believed to be adequate for export requirements in the foreseeable future, but detailed information is not available.

In the opinion of the Federal Geological Survey, the reserves of disseminated kyanite ores in Virginia, North Carolina, South Carolina, and Georgia are of the order of tens of millions of tons of ore containing 20 to 30 percent kyanite.

Another potential source of fine-grained kyanite is the tailings from the concentration of monazite, zircon, rutile, and ilmenite from Florida beach sands, which are known to contain 20 to 30 percent of a mixture of kyanite and sillimanite associated with staurolite and small quantities of other minerals. In 1954 accumulation of these tailings from past operations was of the order of 40,000 tons of kyanite-sillimanite content, and the average accumulation of tailings from current operations is several hundred tons a month.

The adequacy of the low-iron siliceous bauxite reserves for producing synthetic mullite may be questioned; however, industry and Government reports indicate that reserves are adequate for meeting short-range requirements, provided that in an emergency production is allocated principally for mullite production.

Sillimanite deposits occur in Montana, Idaho, and elsewhere, but very little is known of the character and grade of the ore as a refractory raw material.

PRICES

As reported by industry in December 1954, quotations on kyanite were as follows: Per short tons, f. o. b., point of shipment, Virginia and South Carolina, 35-mesh, c. l., in bulk, $35, in bags, $38; 200-mesh, in bags, c. l., $46. Quotations on imported kyanite (55- to 59-percent grade) in bags were $75 to $85 per short ton c. i. f. Atlantic ports. One company in the eastern United States quoted sintered synthetic mullite f. o. b. Philadelphia as follows: Rough shaped dobbies, in bulk, $140 per short ton; ground, in bags, minus-4- or minus-7-mesh, $160 per short ton.

FOREIGN TRADE

Data on imports and exports of kyanite and related minerals are shown in table 1. India continued as the principal supplier in 1953, with 63 percent of the total compared with 53 percent in 1952; Union of South Africa supplied 24 percent in 1953 compared with 16 percent in 1952; and British South Africa supplied 13 percent in 1953 and none in 1952. Total imports for 1953 decreased 27 percent compared with 1952. As in 1952, competition from synthetic mullite produced in the United States was partly responsible for the decline. Contributing to the decline was the continued uncertainty of obtaining supplies of high-grade massive kyanite from India. However, high-grade massive Indian kyanite and comparable quality ore from other sources probably will continue to be imported so long as prices and quality are favorable and adequate supplies are available.

PRODUCTION, CONSUMPTION, AND USES

For many years only two companies have been producing kyanite in the United States: Commercialores, Inc., 39 Cortlandt St., New York, N. Y., from deposits at Henry Knob, near Clover, S. C.; and Kyanite Mining Corp., Cullen, Va., from a property on Baker Mountain near Farmville, Prince Edward County, Va. The Bureau of Mines withholds production figures to prevent disclosure of individual operations.

TABLE 1.—*Kyanite imported for consumption and kyanite and allied minerals exported from the United States, 1945–48 (average), and 1949–53*

(Compiled by Mae B. Price and Elsie D. Page ,Division of Foreign Activities, Bureau of Mines, from records of U. S. Department of Commerce)

Imports			Exports		
Year and origin	Short tons	Value	Year and destination	Short tons	Value
1945–48 (average)	13,930	$180,553	1945–48 (average)	[1] 345	$20,113
1949	12,119	324,856	1949	1,039	46,725
1950	17,417	587,819	1950	941	35,750
1951	19,570	812,484	1951	990	43,762
1952			1952		
Australia	84	1,999	Canada	575	22,348
British East Africa	2,595	101,173	France	60	3,443
Canada	57	4,598	Mexico	494	18,706
India	4,835	217,908			
Union of South Africa	1,486	64,879	Total 1952	1,129	44,497
Total 1952	9,057	390,557	1953		
1953			Canada	586	24,036
India	4,155	184,293	Mexico	446	17,365
Southern British Africa	858	22,477	Total 1953	1,032	41,401
Union of South Africa	1,607	80,919			
Total 1953	6,620	287,689			

[1] Revised figure.

No production of andalusite, dumortierite, sillimanite, or topaz has been reported in recent years.

Domestic consumption of kyanite, from foreign and domestic sources, and synthetic mullite during 1950–53 was about 30,000, 38,000, 40,000, and 45,000 short tons, respectively, exclusive of material purchased for the National Stockpile.

Mullite refractories represent only a small percentage of the total tonnage of refractories used in the United States, but they occupy an important position in that field because of their relatively high softening points, low coefficient of expansion, and resistance to loads at high temperatures, thermal shock, and corrosive action of certain fluxing agents. Although mullite refractories are relatively expensive, industry has found it profitable to use them for certain superduty refractories applications.

Mullite refractories are used in the form of brick and shapes or in the form of cements, mortars, plastics, and ramming mixtures.

In some instances relatively fine-grained domestic mullite is blended with the coarse-grained mullite obtained from imported kyanite or synthetic mullite in the production of refractory brick and shapes. Domestic kyanite is satisfactory for use in refractory cement and for other uses that do not require a coarse-grained material, and such uses account for the major part of the United States consumption of domestic kyanite.

For a number of years, about 90 percent of all mullite refractories have been used to line furnaces operated by the metallurgical and glass industries. About 50 percent of the mullite refractories were used by the metallurgical industry and 40 percent by the glass industry. The remaining 10 percent has been used for miscellaneous applications, chiefly in the ceramic industry.

In the metallurgical industry the principal use of mullite refractories is in electric furnaces, largely the induction type, for melting brass, bronze, copper-nickel alloys, certain steels, and ferrous alloys. Other metallurgical applications are in zinc-smelting and gold-refining furnaces.

In the glass industry mullite refractories are used mainly in constructing continuous tanks, especially in the superstructure, and in plungers, rings, and tubes for feeding molten glass to the forming machines.

In the ceramic industry small quantities of mullite refractories are used for manufacturing kiln furniture (for placing ceramic ware in kilns); in saggers (open-topped refractory boxes for protecting ware during firing); and in kiln construction. Small quantities of kyanite without calcination are used as a source of alumina in glass and as an ingredient of electrical and chemical porcelain and pyrometer tubes.

Purchase specifications for crude kyanite or related minerals, or prepared mullite grain, include limits of chemical composition, a minimum pyrometric-cone equivalent, and a specified grain-size distribution. In addition, most purchasers of new sources of supply require hot and cold load tests and spall tests on brick made by commercial processes.

OUTLOOK

Mullite-type refractories have physical-chemical properties essential for certain types of metallurgical, glass, and other furnaces. As industrial growth continues, the demand for these special refractories will increase.

In addition to the expanding requirements for present uses, the general trend is toward higher and still higher temperatures in metallurgical furnaces and in boiler installations. This creates a demand for new refractories to withstand higher operating conditions. Mullite-type or mullite-containing refractories are desired for such installations because of their physical-chemical properties and relative costs compared with other refractories that might be used.

Discovery of adequate domestic sources of massive-type, mullite-forming minerals appears doubtful. Development of synthetic mullite materials appears to offer the best source for future demands.

Geographically, the present demand for kyanite-mullite refractories is centered in the metallurgical and glass industries in the eastern United States. Westward and southwestward movement of these industries will affect both marketing and manufacturing demands of kyanite-mullite crude materials and the refractories made therefrom.

Most of the known kyanite (including sillimanite and topaz) deposits with commercial possibilities are in the East and Southeast, and the bulk of the bauxite materials that might be used as raw materials for synthetic mullite occurs in the Southeastern and South Central States. The reported or known deposits of bauxitic materials in the west are too high in iron oxide (Fe_2O_3) and/or titania (TiO_2) for manufacturing synthetic mullite to meet 1954 standards. Therefore, the raw materials, the synthetic mullite, or the fabricated products (refractories) individually or collectively would require shipment southwestward or westward for these new markets. However, research may evolve methods whereby the large reserves of relatively low silica, high iron oxide-titania bauxitic materials might be used for mullite synthesis. In addition, these investigations may evolve new standards or specifications for synthetic mullites.

In case of complete loss of access to foreign sources, it has been estimated that the United States could achieve self-sufficiency in mullite production within about 1 year by increasing manufacturing facilities for synthetic mullite. Any plant equipped to manufacture high-alumina refractories or similar products can be converted to manufacture synthetic mullite and mullite refractories.

Low-iron siliceous bauxite and domestic kyanite concentrates are of increasing interest to manufacturers of high-alumina refractories as substitutes for declining reserves of diaspore and also for selected Caribbean bauxite. However, up to 1954 known minable reserves of domestic, low-iron siliceous bauxite were insufficient to encourage expansion of their use beyond that required for mullite and synthetic mullite production.

PROBLEMS

There are five major problems in the kyanite-mullite refractories-manufacturing industry, namely: (1) Lack of adequate kyanite-mullite specification; (2) lack of enough low-cost electrical energy in suitable locations for future expansion of fused synthetic mullite production; (3) lack of detailed information on the quantity and quality of potential commercial deposits of mullite-forming materials; (4) need to develop additional uses for domestic kyanite concentrates; and (5) need for improved beneficiation methods for recovering kyanite from Florida beach sand tailings and other potential sources.

In 1954 specifications of kyanite or mullite were based primarily upon availability of materials rather than upon either scientific data or operating experience. This lack of definite specifications hampered evaluation of the suitability of mullite-forming raw materials.

Fused synthetic mullite is made in electric-arc furnaces and requires relatively cheap power, thus favoring plant locations in areas where low-cost electricity can be supplied.

Domestic kyanite production depends closely upon the refractory cement and mortar requirements of the metallurgical and glass industries—a comparatively small market, quantitywise. More diversified uses would assure more uniform demand and would increase requirements, permitting lower cost, more efficient operations.

Although there are large reserves of kyanite-bearing materials, few data are available as to the amenability to concentration of some of the more important deposits, as, for example, the tailings from Florida monazite-ilmenite concentration. Improved beneficiation techniques for the kyanite minerals would greatly expand the commercial reserves of fine-grained kyanite.

The increasing use of synthetic mullite to replace mullite from imported kyanite and the growing shortage of high-alumina refractory materials in certain areas has focused attention on the need for additional data on domestic deposits of mullite-forming raw materials.

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LEAD

By

O. M. Bishop[1] and R. L. Mentch[1]

LEAD, the softest and the heaviest of the base metals, is vital to modern civilization, particularly in transportation, communications, and chemical manufacture.

Transportation annually requires over a billion pounds of lead for storage batteries, bearing metals, and tetraethyl gasoline; and equally important, but smaller quantities are used in communications, typemetal, telephone and telegraph-cable coverings, solder, and the glass of electronic tubes. The construction industry also is a large user for pipe, sheet, and calking ingots.

Summary

The use of lead predates the Christian era by 2,000 to 4,000 years. The ancients used the metal in coinage, weights, piping, and corrosion-resistant containers. It was also employed in ceramic glazes and glass. Man still utilizes lead for most of these same products. In addition, a host of newer uses has been developed, including those in storage batteries, bearings, cable coverings, ammunition, type metal, pigments, and chemicals.

The United States is the world's leading source of lead ores and the largest producer and consumer of the metal. During the 29-year period 1925–53 the mines of the United States produced 12,822,000 tons of recoverable lead or for the same period, 26 percent of the total world mine output; and United States industries consumed about 36 percent of the world supply of new lead. In recent years world mine production has increased substantially while United States output has declined; in 1950–53 United States mine output averaged 388,000 tons annually or only 20 percent of the world total, but its consumption totaled 41 percent of the new lead.

World War II marked the beginning of United States dependence on imports for a large portion of its needs. For many years the United States had been self-sufficient in lead; but, beginning in 1940 and continuing thereafter, greatly increased domestic requirements necessitated the importation of large tonnages of foreign metal and ores and concentrates to augment home supplies. The return of the United States to a position of self-sufficiency is highly improbable. Australia, Mexico, and Canada normally are the second, third, and fourth largest producers of lead, and historically Mexico and Canada have been the chief sources of United States imports. Following the United States, other major producers of lead on a mine basis, in order of output in 1953, were Australia, Mexico, U. S. S. R., Canada, Peru, Yugoslavia, and French Morocco. Together with the United States these seven countries produced three-fourths of the total world output in 1953.

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[1] Commodity-industry analyst, Bureau of Mines.

Total developed world reserves of lead ore are estimated to contain approximately 40.5 million tons of the metal; in all probability, the lead content of inferred ore is somewhat larger. Of the total developed reserves 10 million tons is in North America, 2.5 million in South America, 10 million in Europe, 3.5 million in Africa, 2 million in Asia, and 12.5 million in Australia. The largest and most important known reserves are in Australia, Canada, the United States, Mexico, Peru, Argentina, Bolivia, Germany, Spain, Italy, Sweden, U. S. S. R., Yugoslavia, French Morocco, Algeria, and Burma.

The various segments of the lead industry in the United States—mining, smelting and refining, secondary-recovery operations, fabricating, and marketing—are limited in varying degrees to relatively few companies. The 5 largest producers, on a mine basis, normally supply over half of the total United States output; almost all the primary refining capacity is controlled by 4 companies; and in marketing 2 companies are preeminent. However, over 200 companies are engaged in secondary-recovery operations, which, in the 12 years 1942–53, produced an average of 428,000 tons of recovered lead annually.

World production of primary and secondary lead in recent years (1948–54) has been well in excess of world requirements; consequently, large stocks have accumulated. In 1950 and 1951 consumer demand and the stockpiling program not only stimulated further production but absorbed much of the excess output. When in 1952 stockpiling and world consumption were both cut back, prices began to fall. Much of the excess production was shipped to the United States to be sold in competition with domestic output. Prices fell rapidly, many mines closed, and by 1954 domestic mine production was at the lowest rate in 20 years. In the second half of 1954 resumption of United States stockpiling and increased world consumption tended to alleviate the market situation somewhat, and there were indications that domestic production was reviving. The solution to the world problem of oversupply would appear to be increased consumption with market research guiding any future production-expansion programs. Since consumption is stimulated by low lead prices, every effort must be made to improve extraction techniques and to lower costs per unit of output.

BACKGROUND

HISTORY OF LEAD INDUSTRY (*19*) [2]

Lead is among the metals longest known to man. Many of the earliest known writings refer to lead. Its uses in pipe, sheet metal, weights, and solder and in glass and ceramic glazes were known in various degrees by the ancient Egyptians, Phoenicians, Greeks, and Romans. The Phoenicians worked lead mines in Cyprus, Sardinia, and Spain, and the famed lead-silver mines of Laurium, Greece, supplied the ancient Greeks with much of their revenue. The Romans mined lead in what is now Rumania, France, England, Spain, and Italy. The great quantities of silver that enriched Rome for 6 centuries were derived from lead smelting-refining operations in Spain, Sardinia, and Britain.

In the United States lead was mined and smelted in Virginia as early as 1621, and the discovery of lead in the Upper Mississippi Valley was reported in 1690. Various efforts were made by the French to establish lead-mining operations in what is now the Southeast Missouri lead district during the first part of the 18th century, but it was not until 1763 that the area began producing on a continuous basis. The mines of the Upper Mississippi Valley were important sources of lead during much of the 19th century but have since declined, owing to exhaustion of the surface concentrations. In 1867 new discoveries in Southeast Missouri at depths of over 100 feet led to the further development of one of the most productive lead regions in the world. The year 1953 marked the 46th consecutive year in which Southeastern Missouri was the leading lead-producing mining district in the United States.

Early lead production in the United States was small; the average yearly output from 1801 through 1810 was only 1,000 tons. Production increased progressively during the early part of the 19th century. From 1831 to 1840 the United States produced about 10,000 tons of lead a year—approximately 10 percent of the total world output. The Missouri and Upper Mississippi Valley areas were the principal producing regions during that time. Lead production from 1840 through 1870 was essentially at the same rate as in the decade 1831–40; but world production had increased so that during 1861–70 United States production was only 4 percent of the world total.

Completion of the first transcontinental railway in 1869 gave great impetus to the lead-mining industry. An era of prospecting followed that led to discovery of lead deposits at Eureka, Nev.; Bingham Canyon, Park City, and Big and Little Cottonwood Canyons, Utah; Leadville, Colo.; Cerro Gordo, Calif.; Bonne Terre and Joplin, Mo.; and the Coeur d'Alene, Idaho. Production increased sharply in the 1870's; the average yearly output from 1871 through 1880 was 60,000 tons, approximately 16 percent of the world output.

Mine and smelter production increased rapidly during the 1880's and 1890's. The average yearly output from 1881 through 1890 was about 140,000 tons—nearly 30 percent of the world total. The United States became the world's largest lead producer during the 80's, and for the decade 1891–1900 production was annually over 200,000 tons, or more than 25 percent of the total world output.

Lead production increased remarkably from 1900 through 1929. In 1900 the mine production of recoverable lead totaled 270,000 tons, and by 1925 it was 684,000 tons—over one-third of the total world output. The United States attained its greatest mine output during 1925–29, when the average yearly production was 662,000 tons.

The development of selective flotation has contributed greatly to the mining industry since 1925 by making it possible to selectively concentrate the lead occurring in many complex western ores. Smelting and refining capacity was increased during the first 25 years of the 20th century, and recovery of lead from scrap has increasingly become a more important element of lead supply, so that in recent years secondary production has been over a third of the United States total lead supply.

The United States continued to be the world's leading lead-mining nation throughout the 1925–53 period; but owing to increased requirements, beginning in 1940, imports were necessary to fulfill domestic needs.

SIZE, ORGANIZATION, AND GEOGRAPHIC DISTRIBUTION OF INDUSTRY

Lead is one of the most important nonferrous metals produced and used in substantial tonnages by modern industrialized nations. From 1925 through 1953 United States mine production of lead ranged from 684,000 tons to 273,000 and for the 29 years averaged 442,000 tons or

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

about 26 percent of the total world output. In addition to the smelting-refining industry based on the reduction of ores and concentrates, the smelting of scrap lead is an important factor in the lead supply of the United States; since 1946 such production has averaged 472,000 tons annually, exceeding mine production. United States consumption (primary and secondary) from 1941 through 1953 has averaged more than 1,100,000 tons annually or considerably more than half of total world consumption.

A comparison of the lead industry of the United States with that of the world is given in table 1. The table includes data on the consumption of primary lead.

Economic lead-mineral deposits are widely distributed on all continents of the world; consequently lead mining and smelting operations are performed in many countries. In 1953, 21 countries produced over 10,000 short tons of lead on a mine basis, and 34 other countries produced smaller quantities. The United States was by far the largest producer, followed by Australia, Mexico, U. S. S. R., Canada, and Peru. Together these countries produced nearly 1,400,000 tons of lead or two-thirds of the world total.

In 1953 there were 94 smelters and refineries in 30 countries. The smelter production in the United States, Mexico, U. S. S. R., Australia, Canada, and West Germany comprised 70 percent of the total output.

World mine and smelter production, by countries, is given in tables 2 and 3 and smelting and refining facilities in the United States and foreign countries in tables 4 and 5.

TABLE 1.—*Comparison of lead industry of the United States with world totals 1925–53* [1]

[Thousand short tons]

Year	Mine production			Smelter production			Primary consumption [2]		
	Total	United States	Remainder of world	Total	United States [3]	Remainder of world	Total	United States	Remainder of world
1925	1, 683	684	999	1, 717	702	1, 015	1, 627	688	939
1926	1, 765	684	1, 081	1, 805	729	1, 076	1, 701	755	946
1927	1, 849	665	1, 184	1, 898	700	1, 198	1, 749	695	1, 054
1928	1, 809	627	1, 182	1, 858	653	1, 205	1, 786	702	1, 084
1929	1, 890	648	1, 242	1, 968	702	1, 266	1, 820	727	1, 093
1930	1, 819	558	1, 261	1, 869	608	1, 261	1, 700	575	1, 125
1931	1, 461	405	1, 056	1, 530	412	1, 118	1, 412	399	1, 013
1932	1, 315	293	1, 022	1, 279	271	1, 008	1, 192	275	917
1933	1, 305	273	1, 032	1, 273	257	1, 016	1, 301	313	988
1934	1, 446	287	1, 159	1, 464	310	1, 154	1, 485	333	1, 152
1935	1, 523	331	1, 192	1, 524	324	1, 200	1, 582	389	1, 193
1936	1, 656	373	1, 283	1, 629	399	1, 230	1, 739	490	1, 249
1937	1, 863	465	1, 398	1, 852	467	1, 385	1, 922	551	1, 371
1938	1, 942	370	1, 572	1, 873	365	1, 508	1, 808	400	1, 408
1939	1, 902	414	1, 488	1, 909	446	1, 463	(4)	(4)	(4)
1940	1, 936	457	1, 479	1, 936	517	1, 419	(4)	(4)	(4)
1941	1, 784	461	1, 323	1, 875	545	1, 330	(4)	(4)	(4)
1942	1, 793	496	1, 297	1, 835	549	1, 286	(4)	(4)	(4)
1943	1, 584	453	1, 131	1, 644	469	1, 175	(4)	(4)	(4)
1944	1, 281	417	864	1, 424	465	959	(4)	(4)	(4)
1945	1, 257	391	866	1, 233	443	790	(4)	(4)	(4)
1946	1, 268	335	933	1, 165	338	827	1, 352	542	810
1947	1, 498	384	1, 114	1, 455	441	1, 014	1, 605	744	861
1948	1, 570	390	1, 180	1, 504	400	1, 104	1, 578	745	833
1949	1, 692	410	1, 282	1, 648	476	1, 172	1, 478	579	899
1950	1, 840	431	1, 409	1, 808	505	1, 303	1, 922	885	1, 037
1951	1, 863	388	1, 475	1, 763	415	1, 348	1, 818	678	1, 140
1952	2, 028	390	1, 638	1, 940	472	1, 468	1, 757	782	975
1953	2, 094	342	1, 752	1, 973	468	1, 505	1, 931	784	1, 147
Average	1, 680	442	1, 238	1, 678	478	1, 200	1, 648	592	1, 056

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[1] Sources: Materials Survey—Lead, Bureau of Mines and Geological Survey, 1951; Yearbooks of the American Bureau of Metal Statistics; and various Bureau of Mines data. It should be borne in mind that world statistics contain numerous estimates, some of which are based on incomplete information.

[2] It is aimed, so far as possible, to have the accounting by countries on the basis of new lead, hence United States totals will not correspond with domestic consumption figures given elsewhere in this report.

[3] Refined lead production, excluding that produced from foreign base bullion.

[4] Comprehensive data not available.

TABLE 2.—*World mine production of lead, by countries, 1944–48 (average) and 1949–53* [1]

[Metric tons]

(Compiled by Pauline Roberts)

Country	1944–48 (average)	1949	1950	1951	1952	1953
North America:						
Canada	170,611	144,945	150,317	143,544	153,170	177,618
Cuba	13	47	12	------	------	------
Guatemala	438	3,154	3,000	3,300	4,200	7,066
Honduras	29	449	279	454	538	[2] 989
Mexico	189,438	220,763	238,078	225,468	246,027	221,549
Salvador [3]	40	530	530	470	100	------
United States [4]	347,970	371,860	390,838	352,135	353,947	310,139
South America:						
Argentina	19,925	16,000	[3] 20,000	[3] 20,000	18,000	15,400
Bolivia (exports) [4]	12,782	26,311	31,176	30,558	28,291	23,788
Brazil	[3] 495	2,000	4,000	3,500	2,800	3,000
Chile	1,978	2,859	3,318	7,801	[3] 4,000	[3] 3,200
Ecuador	279	380	229	30	114	114
Peru	50,807	65,357	62,118	82,350	98,069	122,900
Europe:						
Austria	2,433	4,297	4,440	4,522	5,503	5,150
Bulgaria [3]	4,500	10,000	10,000	10,000	10,000	(5)
Czechoslovakia	[3] 2,000	(5)	(5)	(5)	(5)	(5)
Finland	146	130	142	216	216	217
France	6,498	10,004	12,430	10,074	11,815	11,530
Germany:						
East [3]	[6] 1,375	2,500	2,500	2,600	2,600	(5)
West	[6] 16,930	40,944	44,830	50,377	51,597	62,901
Greece [7]	775	1,200	5,800	3,800	6,000	5,700
Hungary	60	(5)	300	(5)	(5)	(5)
Ireland	------	152	375	1,207	1,118	(5)
Italy	14,900	35,800	40,100	40,200	40,100	40,500
Norway	113	301	234	414	413	500
Poland [8]	12,677	17,850	18,000	18,000	20,000	21,300
Portugal	384	746	1,311	1,621	1,921	1,700
Rumania [3] [8]	2,673	4,000	4,000	4,000	5,000	6,000
Spain	31,366	29,685	39,266	40,442	42,384	54,204
Sweden	20,395	23,900	22,673	19,693	20,593	23,714
U. S. S. R.[3] [8]	54,200	90,000	111,600	128,400	154,200	183,200
United Kingdom	3,017	2,505	3,336	4,925	5,774	4,357
Yugoslavia	42,434	72,144	86,039	78,750	78,968	85,152
Asia:						
Burma [3]	10	300	1,000	2,000	3,000	8,000
China [3]	300	1,000	1,200	1,500	2,000	(5)
Hong Kong	------	------	------	179	300	300
Iran	------	------	2,000	1,100	17,500	16,300
Japan	8,165	9,132	10,896	12,876	17,484	18,516
Korea:						
Korea, Republic of	[3] 5,882	87	40	------	142	149
North Korea [3]		5,000	3,000	(5)	(5)	(5)
Philippines	17	550	879	571	2,300	2,434
Thailand (Siam)	------	183	691	1,321	1,048	3,600
Turkey	815	200	260	600	[3] 1,000	[3] 1,400
Africa:						
Algeria	935	1,121	1,393	2,838	4,220	7,950
Belgian Congo	658	180	------	------	------	------
Egypt	12	12	1	144	15	99
French Equatorial Africa	2,788	731	1,814	2,504	3,551	4,424
French Morocco	16,819	37,155	48,759	68,134	83,608	78,860
Nigeria	73	29	12	4	27	35
Northern Rhodesia [8]	8,057	14,169	13,905	14,194	12,802	11,694
Southern Rhodesia	60	83	------	------	------	62
South-West Africa	9,340	38,400	33,680	39,230	[4] 52,842	[4] 59,227
Spanish Morocco	150	159	178	370	732	[3] 600
Tanganyika	2	------	652	1,561	2,833	[3] 3,500
Tunisia	9,353	17,767	18,860	21,250	23,270	23,700
Uganda (exports)	4	39	44	9	8	16
Union of South Africa	151	166	600	900	575	500
Australia	193,383	216,918	222,694	228,407	236,780	249,099
Total (estimate)	1,286,000	1,550,000	1,670,000	1,690,000	1,840,000	1,900,000

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[1] Compiled by the Bureau of Mines Division of Foreign Activities for the Lead chapter of Minerals Yearbook, 1953; incorporates a number of revisions of data published in previous lead chapters.

[2] Imports into United States.

[3] Estimate.

[4] Tonnage recoverable from ore.

[5] Data not available; estimate included in total.

[6] Average for 1945–48.

[7] Includes lead content of zinc-lead concentrates.

[8] Smelter production.

TABLE 3.—*World smelter production of lead by countries where smelted, 1944–48 (average) and 1949–53*[1] [2]

[Metric tons]

(Compiled by Pauline Roberts and Berenice B. Mitchell)

Country	1944–48 (average)	1949	1950	1951	1952	1953
North America:						
Canada	144,004	132,608	154,551	147,609	166,367	150,915
Guatemala	121	68	271	60	316	658
Mexico	184,397	212,004	230,831	219,107	237,443	214,972
United States (refined)[3]	378,734	431,692	458,171	376,142	428,597	424,309
South America:						
Argentina	21,005	18,037	18,960	24,000	20,000	13,000
Brazil	296	1,172	4,200	[4] 3,000	1,946	2,450
Peru	36,496	36,017	31,693	44,247	48,622	59,160
Europe:						
Austria [5]	5,758	9,841	10,910	11,147	10,316	11,836
Belgium [5]	28,869	79,304	62,094	72,821	79,506	76,351
Czechoslovakia	[6] 3,419	(7)	(7)	(7)	(7)	(7)
France	20,841	54,450	61,236	47,974	51,538	54,917
Germany:						
East	[4] 58,660	[4] 10,000	[4] 12,000	16,800	[4] 18,000	[4] 18,000
West		54,551	66,619	76,063	92,682	106,775
Greece	908	2,389	2,125	3,890	2,460	[3] 2,500
Hungary	[8] 674	(7)	300	(7)	(7)	(7)
Italy	12,752	26,346	37,469	36,480	34,931	37,994
Poland	12,677	17,850	18,000	18,000	20,000	21,300
Portugal	263	304	591	724	1,065	(7)
Rumania [4]	2,673	4,000	4,000	4,000	5,000	6,000
Spain	30,988	33,021	40,568	44,711	46,543	51,249
Sweden	9,947	10,757	16,681	9,307	11,390	16,410
U. S. S. R. [4]	54,200	90,000	111,600	128,400	154,200	183,200
United Kingdom [4]	2,800	2,134	3,048	4,158	3,986	4,357
Yugoslavia	28,500	56,760	57,204	60,068	67,180	70,796
Asia:						
Burma	1,514	230	11	4,966	2,675	8,746
China	524	[5] 2,062	[4] [5] 4,000	[4] [5] 5,000	[4] [5] 6,000	[4] [5] 9,000
India	182	603	636	873	1,150	1,728
Indochina	10	----------	----------	----------	----------	----------
Japan	12,826	7,716	9,984	10,740	15,156	17,724
Korea:						
Korea, Republic of	[9] 183	100	(7)	(7)	126	50
North Korea	[9] 4,333	[4] 5,000	[4] 3,000	----------	----------	[4] 2,000
Africa:						
French Morocco	564	7,073	12,097	22,322	30,088	27,433
Northern Rhodesia	8,057	14,169	13,905	14,194	12,802	11,694
South-West Africa	29	----------	----------	----------	----------	----------
Tunisia	9,541	19,429	23,536	22,906	25,506	27,280
Australia [10]	155,234	152,464	163,102	168,418	159,153	175,236
Total (estimate)	1,235,000	1,495,000	1,640,000	1,600,000	1,760,000	1,790,000

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[1] Data derived in part from Monthly Bulletin of the United Nations, Statistical Summary of the Mineral Industry (Colonial Geological Surveys, London), and the Yearbook of the American Bureau of Metal Statistics.

[2] Compiled by the Bureau of Mines Division of Foreign Activities for the Lead chapter of Minerals Yearbook, 1953; incorporates a number of revisions of data published in previous lead chapters.

[3] Figures cover lead refined from domestic and foreign ores; refined lead produced from foreign base bullion not included.

[4] Estimate.

[5] Includes scrap.

[6] Average for 1945–48.

[7] Data not available; estimate included in total.

[8] Data represent Trianon Hungary after October 1944.

[9] Average for 1946–48.

[10] Excluding lead content of bullion exported, figures for which are as follows: 1944–48 average, 14,609 metric tons; 1949, 33,145; 1950, 37,615; 1951; 32,384; 1952, 38,314; 1953, 34,840.

TABLE 4.—*United States primary lead smelting and refining capacity, 1953* [1]

SILVER-LEAD SMELTERS

Location	Company	Annual capacity (short tons charge)
East Helena, Mont	American Smelting & Refining Co	336, 000
El Paso, Tex	do	300, 000
Leadville, Colo	do	180, 000
Selby, Calif	do	192, 000
Bradley, Idaho	Bunker Hill & Sullivan Mining & Conc. Co	300, 000
Tooele, Utah	International Smelting & Refining Co	300, 000
Midvale, Utah	U. S. Smelting, Refining & Mining Co	250, 000
Total		1, 858, 000

SILVER-LEAD REFINERIES

Location	Company	Annual capacity, refined lead (short tons)
Selby, Calif	American Smelting & Refining Co	72, 000
Barber, N. J	do	144, 000
Omaha, Nebr	do	200, 000
Bradley, Idaho	Bunker Hill & Sullivan Mining & Conc. Co	100, 000
East Chicago, Ind	U. S. S. Lead Refinery, Inc	40, 000
Midvale, Utah	U. S. Smelting, Refining & Mining Co	72, 000
Total		628, 000

SMELTERS AND REFINERIES OF MISSOURI LEAD [2]

Location	Company	Annual capacity, pig lead (short tons)
Herculaneum, Mo. [2]	St. Joseph Lead Co	120, 000
Federal, Ill	American Smelting & Refining Co	128, 000
Galena, Kans	Eagle-Picher Mining & Smelting Co	10, 000
Total		258, 000

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[1] Source: Yearbook of the American Bureau of Metal Statistics, 1953.

[2] The smelting and refining of Missouri lead ore is done in the same plant. The smelting is of a high grade of galena concentrate, which is generally nonargentiferous, although a little desilverization is done by St. Joseph Lead Co.

TABLE 5.—*Foreign lead smelting and refining capacity, 1953* [1]

Location	Company	Type [2]	Estimated annual capacity for refined lead (short tons)
Argentina:			
Mercedes	Elaboracion General del Plomo, S. A.	S&R	7, 900
Puerto Vilelas (Chaco)	National Lead Co., S. A	do	19, 800
Australia:			
Port Pirie, Southern Australia	Broken Hill Assoc. Smelters Proprietary, Ltd.	do	246, 400
Mount Isa, Queensland	Mount Isa Mines, Ltd	S	55, 100
Austria: Gailitz, Carinthia	Bleiberger Bergwerks-Union	S&R	11, 000
Belgium:			
Beersee	Cie. Met. de la Campine	S	6, 600
Hoboken	Soc. Gen. Met. de Hoboken	S&R	66, 100
Overpelt	Cie des Metaux d'Overpelt-Lommel et de Corphalie.	do	19, 800
Baelen-Wezel	Soc. Anon. des Mines et Fonderies de Zinc de la Vieille Montagne.	do	48, 200
Brazil:			
Apiai, Sao Paulo	Instituto de Pesquisas Tecnologicas	do	3, 900
Panelas, Parana	Plumbum, S. A	do	11, 000
Burma: Namtu, Burma	Burma Corp. (1951), Ltd	do	74, 000
Canada: Trail, B. C	Consolidated Min. & Smg. Co. of Canada	do	220, 000
Czechoslovakia:			
Banska Stiavnica	State-owned	do	1, 600
Pribram	do	do	4, 400
France:			
Montagne	Cie Royale Asturienne des Mines	do	4, 100
Couëron	Soc. Anon. des Mines et Fonderies de Pontgibaud.	do	27, 500
L'Estaque	Soc. Min. et Met. de Peñarroya	R	19, 800
Noyelles Godault	do	S&R	79, 400
Saint-Denis (Seine)	Ste. Francais des Metaux et Alliages Blanco.	do	22, 000
Federal Republic of Germany:			
Mechernich	Gewerkshcaft Mechernicher Werke	do	14, 300
Hamburg	Norddeutsche Affinerie	do	33, 100
Clausthal & Lautenthal	Preussag-Harzer Berg-und Hüttenwerke	do	26, 400
Braubach	Blei- und Silberhütte Braubach	do	44, 100
Binsfeldhammer	Stolberger Zink A.-G. f. Bergbau u. Zinkhüttenbetrieb.	do	55, 100
Oker (Harz)	Unterharzer Berg- und Hüttenwerke G. m. b. H.	do	26, 400
Friedrich-August-Hütte, Post Nordenham.	Metallwerke Unterweser A.-G	do	26, 400
East Germany:			
Hettstedt	Mansfeldscher Kupferschieferbergbau A. G	do	3, 300
Freiberg (Saxony)	Vereiningung Volkseigener Betriebe-Buntmetall.	do	11, 000
Greece: Laurium	Cie. Francaise des Mines du Laurium	do	6, 600
Guatemala: Villa Linda-Huehuetenango	Compania Minera de Huehuetenango, S. A.	S	
India: Tundoo, Katrasgarh	The Metal Corp. of India, Ltd	S&R	6, 600
Italy:			
Monteponi, Sardinia	Soc. di Monteponi	do	13, 200
La Spezzia	Soc. Min. & Met. di Pertusola	do	26, 400
S. Gavino Monreale, Sardinia	Montevecchio Soc. Italiana del Piombo e dello Zinco.	do	39, 700
Japan:			
Kamioka	Mitsui Mining & Smelting Co., Ltd	do	7, 900
Takehara	do	do	7, 100
Hosokura	Mitsubishi Metal Mining Co. Ltd	do	11, 900
Saganoseki	Nihon Mining Co. Ltd	do	12, 700
Aizu	Nihon Soda, K. K	R	2, 000
Chirigishima and Annaka	Toho Aen K. K	S&R	2, 900

See footnotes at end of table.

TABLE 5.—*Foreign lead smelting and refining capacity, 1953* [1]—Continued

Location	Company	Type [2]	Estimated annual capacity for refined lead (short tons)
Mexico:			
Monterrey, N. L	American Metal Co. Ltd	R	108, 000
Do	American Smelting & Refining Co	do	216, 000
Torreon, Coah	American Metal Co. Ltd	S	
Chihuahua	American Smelting & Refining Co	do	
San Luis Potosi	do	do	
Concepcion del Oro, Zacs	Mazapil Copper Co., Ltd	do	
Mazatlan	Met. Occidental, S. A. (Mosa)	do	
Morocco: Oued-el-Heimer	Fonderies Peñarroya-Zellidja	S&R	38, 600
Netherlands: Arnhem	N. V. Hollandsche Metallurgische Bedrijven	do	6, 600
Peru: Oroya	Cerro de Pasco Corp	do	77, 200
Poland:			
Trzebinia	Giesche Sp. Akcyjna	do	11, 000
Mala Dabrowka	Giesche Sp. Akcyjna (Huta Walther Croneck).	do	16, 500
Strzybnica	Polska Huta (Tarnowitz)	do	33, 100
Brzozowice-Kamien	Zaklady Hohenlohego	R	3, 100
Federation of Rhodesia and Nyasaland:			
Broken Hill, Northern Rhodesia	Rhodesia Broken Hill Dev. Co	S	19, 800
Rumania:			
Firija de Jos, Satu Mare	State Mines, Baia-Mare Soc. "Minaur"	S&R	5, 500
Baia-Mare, Lucaci	Phönix, Fabrica de Acid Sulfuric si Prod. Chim.	do	3, 600
Spain:			
Linares (Jaén)	Cia. "La Cruz"	do	40, 200
Malaga	Cia. Min. Met. Los Guindos	do	22, 000
Peñarroya	Soc. Min. et Met. de Peñarroya	do	66, 100
Santa Lucia, Cartagena (Murcia)	do	do	33, 100
Bellmunt de Ciurana (Tarragona)	Minas del Priorato	do	6, 600
Renteria	Cie. Royale Asturienne des Mines	do	8, 800
Linares (Jaén)	Cia. Sopwith	do	27, 500
Alsúa (Vizcaya)	Industrias Reunidas Minero-Metalurgicas	do	13, 200
Sweden:			
Rönnskär	Bolidens Gruv A. B	do	22, 000
Fliseryd	Svenska Ackumulator AB Jungner	do	5, 500
Tunisia:			
Souk-el-Khémis	Soc. Anon. Francaise de Djebel Hallouf	do	2, 600
Mégrine	Soc. Min. et Met. de Peñarroya	do	27, 600
Bizerte	Mines & Fonderies Tunisiennes	do	3, 900
United Kingdom:			
Northfleet	Britannia Lead Co., Ltd	R	82, 700
Rotherhithe & Darley Dale	H. J. Enthoven & Sons, Ltd	do	44, 100
London, Chester & Newcastle-on-Tyne.	Associated Lead Manufacturers, Ltd	S&R	82, 700
U. S. S. R.:			
Chimkent (Kazakhstan)	State-owned	do	110, 000
Leninogorsk (Kazakhstan)	do	do	110, 000
Tetiukhe (Siberia)	do	do	5, 500
Dzhaudzhukau (Caucasus)	do	do	16, 500
Yugoslavia:			
Mezica	Rudnici i Topionice Olova i Cinka (Trepca)	do	16, 500
Zvecan	do	do	66, 100

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[1] Source: Yearbook of the American Bureau of Metal Statistics, 1953, and Minerais et Metaux Société Anonyme, 1953.

[2] S denotes smelter; R refinery.

The United States primary lead industry is composed of approximately 300 companies engaged in mining, smelting and refining, and marketing lead.

MINING

Six companies—St. Joseph Lead Co.,United States Smelting, Refining & Mining Co., American Smelting & Refining Co., Bunker Hill & Sullivan Mining & Concentrating Co., the Eagle-Picher Co., and the Anaconda Copper Mining Co., annually produce about two-thirds of the total domestic mine output of lead. Some of these same companies also have substantial lead-mining interests in foreign countries. Many medium-size and small mining companies supply the remaining third of domestic mine production. In 1952 almost 97 percent of the mine output was recovered at 670 lead and lead-zinc mines, the remaining 3 percent being a by-product of copper, gold-, and silver-mining operations. In that year the 39 largest mines, each producing 4,000 or more tons of recoverable lead and zinc, supplied 74 percent of the lead mined. Only 8 percent was from the 584 lead and lead-zinc mines that produced less than 1,000 tons of combined lead-zinc.

In 1953 production was derived from an estimated 575 mines and prospects. The 25 leading mines yielded 78 percent of the domestic output, the 10 leading mines 61 percent, and the 4 largest mines 43 percent of the total production.

Nearly 60 percent of the lead mine production normally comes from the Western States, chiefly Idaho, Utah, Colorado, Montana, Washington, Arizona, and California. The most important districts or regions are the Coeur d'Alene, Idaho; West Mountain (Bingham), Utah; Summit Valley (Butte), Mont.; Metaline, Wash.; Coso (Darwin), Calif.; and Upper San Miguel, Colo.

The remaining mine production comes from the West Central States and Eastern States. The Southeastern Missouri lead district has been the leading producer of lead in the United States for 46 consesutive years. In 1953 output from this district totaled 126,000 tons or nearly double that from the second largest district. The Tri-State (Joplin) region of Oklahoma, Kansas, and Missouri is one of the larger lead-producing districts, but its importance has declined considerably over the 1925–53 period owing to substantial exhaustion of reserves. Virginia, New York, Wisconsin, Illinois, and Kentucky produce a small fraction of the United States output. The 43 leading districts are listed in table 6, in order of their production in 1953, and mine output of recoverable lead, by States, in recent years, is given in table 7.

All the ores, with some minor exceptions, are beneficiated at mills near the mines to produce concentrates for shipment to the smelters.

SMELTING

Smelting and refining operations require large investments and are economic only when a commensurately large supply of raw material is assured over a number of years. Mining companies not having facilities for treating ores and concentrates sell their product to companies that include smelting and refining in their operations or are primarily engaged in the smelting and refining business. The total primary-lead-refining capacity of the United States at the end of 1953 was approximately 886,000 tons of pig lead. The silver-bearing ores of the Western States are treated at silver-lead smelters and refineries and the low-silver content ores of southeastern Missouri, the Tri-State, and Upper Mississippi Valley region are processed at other plants. Silver-lead refineries supply 628,000 tons of the total annual capacity of refined lead, and refineries of the other primary lead producers have an annual capacity of 258,000 tons of pig lead.

Actual production of refined primary lead in 1953 totaled 468,000 tons, representing an operating rate of 53 percent of capacity.

Table 4 lists the capacities and locations of United States primary smelters and refineries.

SECONDARY RECOVERY

The recovery of lead from lead scrap and other lead-bearing scrap material constitutes an integral and important part of the United States lead industry. About 85 percent of all secondary lead in the United States is recovered as refined lead or lead in alloys in the more than 275 secondary lead smelters in the various States; the remaining 15 percent is recovered at primary smelters and refineries as refined lead or one of several lead alloys, or as lead-bearing brass or bronze in brass mills or the plants of ingot makers.

An exact total of secondary lead smelting and refining capacity is unavailable; however, in view of scrap recovery during recent years, secondary-lead smelter capacity must approximate 550,000 tons annually.

In 1953 more than 200 companies were engaged in the processing of scrap lead. The National Lead Co., with its Magnus Metals Division, is by far the largest producer. The following principal companies produced nearly 80 percent of the secondary lead output in 1953: A. M. A. Corp., American Smelting & Refining Co. (including Federated Metals Division), Bers & Co., Inc., Bowers Battery & Spark Plug Co., Electric Storage Battery Co., Goldsmith Bros. Smelting & Refining Co., Imperial Type Metal Co., Metro Smelting Co., Nassau Smelting & Refining Co. (Western Electric Co.), National Lead Co. (including

Magnus Metals Division), North American Smelting Co., Northwest Lead Co., Pennsylvania Smelting & Refining Co., Price Battery Corp., Schuylkill Products Co., Thos. Seitzinger's Sons, Southern Lead Co., U. S. S. Lead Refinery, Inc., and Hyman Viener & Sons.

The secondary smelters are principally in areas of heavy industrial concentration, usually in the vicinity of big cities, such as New York, Philadelphia, Pittsburgh, Buffalo, Baltimore, Cleveland, Detroit, Chicago, St. Louis, Los Angeles, and San Francisco.

MARKETING

There are two principal types of marketing operations in the lead industry. The first involves the purchase of ores and concentrates by custom smelters. These transactions are largely individual contracts between buyer and seller and vary a great deal with local conditions. The other kind of marketing with which this section treats is the sale of refined lead and various alloys of lead.

The American Smelting & Refining Co. and St. Joseph Lead Co. are the principal sellers of primary refined lead in the United States. These two companies market roughly one-half to three-fourths of all such lead, depending upon the quantity being imported.

The Metal Statistics Yearbook for 1953 lists 18 companies—15 domestic and 3 foreign—engaged in marketing 26 brands of refined lead—21 domestic and 5 foreign. In addition to the American Smelting & Refining Co. and the St. Joseph Lead Co., other important sellers of refined lead in the United States are: The American Metal Co., Ltd.; Northwest Lead Co.; C. Tennant, Sons and Co.; Anaconda Sales Co.; United States Smelting, Refining & Mining Co.; and the Yugoslav Government.

In addition to refined lead from ores, the primary refiners normally market a small amount of refined lead obtained from scrap and about 35,000 tons of antimonial lead. The secondary smelters in turn produce some refined lead; but their products are preponderantly secondary pig metals, including antimonial lead, solder, babbitts, and type metal.

The two major markets for refined lead in the United States are New York and St. Louis, and the bulk of domestically produced lead is sold at prices normally based upon quotations in these markets. The differential between New York and St. Louis prices is about 0.2 cent a pound. Quotations are for pig or common lead.

The Engineering and Mining Journal (Metal and Mineral Markets) daily quotation of pig lead is obtained by compiling the weighted average price of sales made during the day. Weekly averages published in this journal are the arithmetical average of the week's daily prices, while the monthly average price is the arithmetical average of the weekly prices. Pig lead is often sold at the average for the month of shipment or at a predetermined price. Annual contracts are on the basis of the average price of the month of shipment, with a fixed tonnage for each month.

Several types of pig lead are marketed by domestic producers.

Corroding lead.—Normally made by submitting desilverized, chemical, or common lead to a second refining process—but if the ores are initially low in bismuth or the refining is by the Betts process, corroding grade can be produced in a single refining operation. It usually commands a premium of $2 to $3 per ton.

Chemical lead.—A term used to describe the undesilverized lead produced from Southeastern Missouri ores. It also commands a premium of $2 to $3 per ton. These ores contain too little silver for profitable desilverization and are very low in bismuth. Copper is present in small quantities.

Acid lead and copper lead.—These are made by adding small quantities of copper to fully refined and desilverized lead. The principal difference is the maximum quantity of bismuth allowed. After copper is added, these grades serve the same uses as chemical lead and also command a premium.

Desilverized lead.—This is made from the argentiferous lead ores mined in the Rocky Mountain States. Some of these ores contain bismuth. The bullion is desilverized primarily to recover the silver and gold values.

Desilverized and soft leads.—These are used mainly in manufacturing white lead, sheet, pipe, shot, and alloys; some minor uses are the preparation of lead wool, powdered lead, ribbon lead, and assay lead. They serve further for the production of litharge, red lead, orange mineral, and lead salts.

The market prices of most lead compounds and high-lead products bear a fixed relation to the price of pig lead.

The major consuming centers for pig lead in the United States are New York, Chicago, Philadelphia, Baton Rouge, Houston, and St. Louis. Other large lead-consuming areas are Detroit, Cleveland, Baltimore, Boston, Pittsburgh, San Francisco, Buffalo, Cincinnati, Seattle, Los Angeles, Atlanta, Dallas, and Charleston, W. Va.

TECHNOLOGY

ECONOMIC GEOLOGY

Lead ores are aggregates of minerals, one or more of which contains lead in economic quantity. The common lead minerals are galena (lead sulfide), cerussite (lead carbonate), and anglesite (lead sulfate). Galena is the most abundant lead mineral found in deposits that have been exploited in the United States. Galena is commonly associated with zinc, silver, gold, and iron minerals. However, in a few districts the ore bodies are characterized by very simple mineralization, with the lead mineral present to the virtual exclusion of other ore minerals. A noteworthy example is Southeastern Missouri, where the lead ores are generally composed of galena in an essentially nonmetallic gangue, with relatively little silver, zinc, copper, or other valuable metals present.

The principal lead-ore minerals are as follows:

Mineral	Composition	Specific gravity	Lead, percent
Galena	PbS	7. 4–7. 6	86. 6
Anglesite	$PbSO_4$	6. 1–6. 4	68. 3
Cerussite	$PbCO_3$	6. 5–6. 6	77. 5
Pyromorphite	$Pb_5Cl(PO_4)_3$	5. 9–7. 1	71. 4

The more important economic deposits in the United States occur either as cavity fillings or replacements (*5*, *6*). The origin of the mineralization in the cavity filling and replacement deposits is similar. The theory that the mineral-bearing solutions were derived from a deep-seated igneous mass is most commonly accepted today. Examples of the cavity-filling type of deposit are the San Juan, Colo., and the Upper Mississippi Basin districts. Replacement-type deposits are classified further as follows: Massive type, as at Leadville, Colo., and Bingham and Tintic, Utah; lodes, as at Park City, Utah, and in the Coeur d'Alenes, Idaho; disseminated, as in the Tri-State district and in Southeast Missouri; and metasomatic, as represented by the Central district, New Mexico.

PROSPECTING AND EXPLORATION

Generally speaking, prospecting is the search for ore, while exploration is the work done to get data on the size, shape, location, and quality of an ore body. Both functions were relatively simple in the 18th and 19th centuries, when such important districts as Southeast Missouri, the Tri-State, Upper Mississippi Valley, and Austinville, Va., were first brought into production. Many silver deposits of Colorado, Utah, Nevada, Idaho, and New Mexico proved to have associated lead minerals, or search in those districts led to the discovery of lead, and great reserves of ores containing zinc and lead were found in the Butte district during exploration for gold, silver, and copper. As the easily discovered ore bodies were exploited, it became necessary to employ new prospecting techniques involving careful geologic mapping of rock formations and structural features that control or may control ore deposition. Through intensive study the mining geologist or exploration engineer has learned the fundamentals of replacement, structural control, contact metamorphism, zonal distribution, structural barriers, wall-rock alteration, oxidation, and tectonics both before and after ore deposition. Utilizing geophysical (*17*, *25*) and geochemical (*12*) techniques, largely developed in petroleum exploration, geologists and engineers have worked intensively seeking anomalous conditions indicative of conditions favorable to economic mineralization. At present the Southeast Missouri lead district has been magnetically mapped by airborne methods, as has the famous Tri-State mining district. Geophysics and geochemistry have recently achieved considerable successes in finding copper-lead-zinc ores in the Little River area of New Brunswick, but most discoveries at present are made on the basis of comparative geology and the finding of analogous conditions, which when tested by drill holes or other openings demonstrate whether a discovery has been made.

MINING

Various mining methods, depending on the type of ore body, are used in extracting lead ore. Underground methods employing either open or supported stopes are employed in most production, but open-pit methods have found limited application in the Tri-State district and Washington (*13*, *21*). Underground stoping methods in use include block caving, room-and-pillar, with and without roof bolts, shrinkage stopes, cut-and-fill stopes, and timbered stopes. Lead mines amenable to open-pit methods have been restricted to relatively few areas, and at the present have minor importance. Mining methods, their advantages and disadvantages, are described in Bureau of Mines Bulletin 419 (*15*).

Improvements in mining practice in the past 40 to 50 years have been based largely on the development and extensive use of power equipment. Thirty years ago almost all lead ores were loaded by hand shoveling, whereas now all loading is done by power shovels, scrapers, or mucking machines. Transportation also has undergone great changes, as hand tramming and mule power have been replaced by motor trains operating on heavy-gage track. More recently trackless mining, which utilizes electric- or diesel-powered units, is finding wide usage in the lead and lead-zinc mines of Washington (*10*, *18*), the Tri-State, the Upper Mississippi Valley (*1*), and Southeast Missouri.

Other changes in equipment that have done much to improve mine output per manshift include better lighting at working faces, better ventilation (including air conditioning), better and more efficient pumps, and harder hitting light rock drills, using carbide bits.

MILLING

Milling, the physical process by which the lead minerals are concentrated, consists of crushing and grinding to liberate the valuable mineral particles, followed by separation of the lead mineral particles by gravity, flotation, or some combination thereof.

The particular combination of ore-dressing equipment used varies with the quantities and kinds of ore minerals present, their value, the character of the gangue mineral, and the mineral particle sizes. Simple ores, such as coarsely disseminated lead or zinc-lead minerals occurring with a low-specific-gravity gangue, are easily treated in heavy-medium cones, jigs, and tables after being crushed and rolled in closed circuit with vibrating or trommel screens and classifiers to give properly sized feed. Collective or differential flotation of the slime products or of a reground middling product completes the flowsheet. Ores of this kind are common in the mines of the Mississippi Valley and eastern United States, but in some instances the ores are concentrated wholly by flotation.

The more complex sulfide ores consist of disseminated mixtures of fine-grained lead and zinc sulfides, usually accompanied by pyrite, some copper sulfides, and some gold and silver in a quartz or quartz-calcite gangue. Such ores may be complicated further by partial oxidation of the sulfides and high-specific-gravity gangue minerals, such as barite, siderite, rhodochrosite, or fluorite. The usual procedure on such an ore is to crush and grind in closed circuit with classifying equipment to a mesh at which the ore minerals are preponderantly separated from the gangue minerals. When the ore minerals are interlocked, the usual practice is to make a bulk sulfide concentrate, followed by regrinding and selective flotation.

Low capital and operating costs have extended the field of the sink-float method to include pretreatment of certain ores, permitting upgrading of ores diluted by nonselective mining methods.

The concentration and reduction of lead ores are accompanied by loss of metal content, which leaves room for considerable improvement. Present ore-dressing practices result in recovery of 85 to 94 percent of the sulfide lead and up to about 88 percent of the oxidized lead. Sulfide losses consist largely of the extremely fine particles, and although much research has been done and improvements have been made (*16*), the results are still unsatisfactory. The Bureau of Mines is doing basic research to improve recovery of slimed lead and zinc sulfides through improved fine grinding practice and flotation. Other research is directed toward more satisfactory recovery in the flotation of oxidized lead mineral particles (*11*).

SMELTING (*20*)

Lead is recovered from its ores almost exclusively by smelting in blast furnaces or ore hearths employing carbon fuels. Ores or concentrates that contain few impurities may be reduced to metal in roasting hearths. Air is used to oxidize the sulfides and coke or coal to reduce the oxides. Some reduction takes place by the reaction between the metallic oxides and sulfides such as $2PbO+PbS=3Pb+SO_2$. Other reactions between the lead compounds present also take place to produce metallic lead. The ore hearth practice is followed at the Federal, Ill., and Galena, Kans., smelters, where Newnam roasting hearths are used.

In blast-furnace smelting the sulfur in the ores and concentrates must be removed by roasting, which is usually accomplished on a Dwight-Lloyd sintering hearth. The Dwight-Lloyd equipment not only eliminates most of the sulfur but produces a blast-furnace feed of desirable characteristics from the mechanically mixed concentrate, undersized sinter particles, and byproduct dusts and fumes collected in the smelter collecting systems. In some plants fluxes are also a constituent of the sinter. In general, the charge to the lead blast furnace consists of lead sinter, coke, fluxing material (such as silica, lime, etc.), and some lead-bearing furnace byproducts. Air is blown through the charge burning the coke to CO and CO_2 and producing a temperature of approximately 1,400° C. The carbon monoxide formed and the hot solid carbon reduce the oxidized lead compounds to bullion, which settles in the bottom or crucible of the furnace. The slags from ores containing considerable zinc are commonly re-treated in slag-fuming furnaces to recover that zinc and any remaining lead. The lead bullion produced in the blast furnace contains any precious metals present in the ore and some metallic impurities and consequently must be refined.

Metallurgical recoveries at primary lead smelters are generally about 97 to 98 percent of the lead contained in the ore and offer a relatively small margin for improvement, except through lowering the unit cost of processing. Recent modernization programs center largely on roasting, dust collection, and transport equipment within the plants.

REFINING (*20*)

Lead bullion from the Mississippi Valley ores, termed chemical lead, is pure enough for most commercial uses without further refining. Most of the lead bullion produced by the blast furnace from complex ores contains enough gold and silver to make their extraction profitable. It also contains various base-metal impurities that must be removed before the lead is marketable for end use. The entire sequence of processes for softening and desilverizing lead bullion and the recovery of byproducts is complex and subject to many variations in practice.

Softening consists of the removal of copper, tin, antimony, and arsenic and is usually accomplished in a drossing or refining kettle. The copper is removed by holding the bullion just above the melting point and skimming copper dross from the surface. Agitation, with addition of elemental sulfur, causes the remaining copper to rise to the surface as a black copper sulfide powder, which is skimmed off. After the copper drossing the temperature of the bullion is normally raised and the bath agitated to induce surface oxidation. The tin, arsenic, and antimony are oxidized, and the oxides (being insoluble in the bath) rise to the surface with some lead oxide, which is skimmed off.

The softened bullion usually is desilverized by the Parkes process of stirring metallic zinc into the bullion. Gold and silver, in that order, combine with the zinc, rise to the surface, and are skimmed off as gold and silver zinc crusts. The zinc remaining in the lead after desilverizing is commonly removed by vacuum distillation or with caustic by the Harris process. If bismuth above acceptable limits is present, the bullion must be refined by the Betts electrolytic process, as at Trail, B. C., and East Chicago, or debismuthizing may be accomplished by the newer Kroll-Betterton process applied to the bullion after desilverizing.

BYPRODUCTS AND COPRODUCTS

Approximately two-thirds of the domestic output of lead is produced on a coproduct basis with zinc from mixed lead-zinc ores. The Southeast Missouri district is the only major lead-producing region in the United States where there has been relatively little zinc mineralization. Silver is an important coproduct of lead and lead-zinc mines, particularly in Idaho, Utah, Nevada, and Colorado. Many mines in these States would not have been opened, and many of them could not operate, if it were not for the silver credit, which helps meet the mining and other production costs. In the Southern Illinois–Kentucky district, fluorspar is recovered as a coproduct with lead and zinc. In some western mines copper and lead are produced as coproducts. Cadmium is generally recovered as a byproduct from lead blast-furnace baghouse dust. Other byproducts produced by virtually all western lead mines include gold, antimony, arsenic, and bismuth. In the Central States coarse tailings from jigs and sink-float machines are sold for railway-track ballast and road building. If the tailings are limestone or dolomite they may be sold for agricultural limestone.

FABRICATION

The fabrication of lead and its alloys includes casting, rolling, and extrusion. Lead alloys are suited to casting in a great variety of molds, ranging from those made of sand to those made of wood, paper, rubber, or plaster. Certain lead alloys are readily die-cast. The reclamation value of scrap from castings is almost 100 percent of the original alloy.

Rolling reduces the pig lead to sheets ranging in thickness from foil to 1-inch. The process consists essentially of melting the charge and casting into slabs up to 8 feet wide and 4 to 6 inches thick. The slab is then hot-rolled to a thickness of 1 inch, cold-rolled to the desired thickness, and finally trimmed to the desired size.

Lead and lead alloys can be extruded in a variety of shapes and by various methods. These may be classified into three groups: (1) The processes used in making such shapes as bar, wire, and pipe; (2) those used in producing cable sheathing; and (3) impact extrusion, as employed in manufacturing collapsible tubes.

LEAD COMPOUNDS

Various lead compounds, most important of which are white lead, red lead, and litharge, are produced from pig lead. Leaded zinc oxide is generally made directly from ores.

White lead is the popular name for basic lead carbonate, one of the oldest of the paint pigments. It is produced by a variety of methods, most common of which are the Carter, Euston, Sperry, and Thompson-Stewart processes. The Old Dutch process, most widely used some years ago, is no longer employed.

The Carter or "quick" process is basically the same as the Old Dutch, except that the metallic lead is blown with steam or air to a fine powder and treated with acetic acid, air, water, and carbon dioxide gas in large, revolving wooden cylinders. Under these conditions the corrosion of metallic lead into white lead proceeds at a rapid rate, requiring only 12 to 14 days, as compared to over 100 days by the Old Dutch process.

In the Sperry electrolytic process pure white lead is produced quickly by the electrolysis of a lead acetate solution between lead anodes and iron cathodes in a concrete cell separated by a porous diaphragm.

In the Thompson-Stewart process the chemical reactions involved are essentially the same as in the Carter process; however, the reaction rate is controlled until all the metallic lead has been oxidized to lead oxide, and the latter is reacted with acetic acid, water, and carbon dioxide to produce basic lead carbonate.

Red lead is made by heating litharge (lead monoxide) in a reverberatory furnace at 900° to 950° F. At this temperature the litharge gradually absorbs oxygen from the air and is converted to red lead. About 24 hours are required to make 85-percent grade, a longer time being required for the grades of higher red-lead content.

Litharge is produced for commercial purposes by roasting pig lead in a reverberatory furnace in the presence of air, which furnishes the oxygen for the conversion to lead monoxide. The lead oxide lumps are then ground to a powder.

Leaded zinc oxide, a pigment consisting of zinc oxide and basic lead sulfate, is manufactured by mixing lead sulfide ore with roasted zinc ore and coal and heating in a furnace. Under proper conditions, a fume of zinc oxide and basic lead sulfate is obtained, which then is cooled and collected in a baghouse. Blended leaded zinc oxides are made by dry-mixing zinc oxide and basic lead sulfate in the proper proportions.

RESERVES (*9*)

Lead-ore reserves are widely distributed on all continents, with important deposits in the United States, Canada, Mexico, Greenland, Guatemala, Peru, Bolivia, Argentina, and Chile, in North and South America and in many countries of Europe, Africa, and Asia, as well as in Australia. The total world reserves of lead in the measured and indicated classification are about 40.5 million tons. Between 80 and 90 percent of these reserves are classified as recoverable. Reserves of inferred ore will probably be as much or more. Of total measured and indicated reserves, approximately 10 million occurs in North America, 2.5 million in South America, 10 million in Europe, 3.5 million in Africa, 2 million in Asia, and 12.5 million in Australia.

North America.—The United States, with Canada and Mexico, have about 25 percent of the world's developed reserves or about 10 million tons of lead. Of this quantity the United States has about 2.5 million tons, Canada about 6.5 million, and Mexico about 1 million. Inferred reserves in the United States are estimated at about 5 million tons, while those in Canada and Mexico are believed to be at least as large as developed reserves.

The United States has maintained its preeminence in mine and smelter output of lead by maintaining sources of ore. Since January 1950 numerous exploration, development, and modernization programs to lower unit cost have been undertaken. These include development and plant expansion at the Iron King mine, Arizona; exploration and plant expansion at Telluride mines, Colorado; numerous extensive exploration programs in the Couer d'Alenes; exploration, development and construction at Hayden Creek, Indian Creek, and Madison mines in Missouri; mine exploration and development program by the Eureka Corp., Ltd., Nevada; development and operation of the Van Stone open-pit mine and expansion program at the Pend Oreille mine, Washington; mine development and expansion of plant facilities at the Austinville and Ivanhoe mines, Virginia; and mine exploration, development, and plant construction in southwest Wisconsin. Modernization programs to improve costs and metal recovery have been completed or are in progress at the Selby, Kellogg, Falena, and Herculaneum smelters.

Since 1950 prospecting, exploration, and preliminary development in Canada have indicated approximately 100 million tons of new lead-bearing ores in the Pine Point area of Northwest Territories and the Bathurst area of New Brunswick. Other important but smaller exploration and development programs include new production of lead-zinc ores at the Blue Bell, H & B, Sil-Van, and Sheep Creek mines, British Columbia; at the United Montauban mines, Quebec; increased lead production at the United Keno Hill Mines, Yukon Territory; and the coming into production of the Sterling mine (Mindamar Metals Corp), Nova Scotia.

Partial development of the Mestersvig deposit in East Greenland has indicated about 15 million dollars worth of high-grade lead-zinc ore. In Mexico the Nuestra Senora and Minas de Iguala and Topia properties have begun producing lead and zinc.

South America.—Lead reserves in South America are estimated at 2.5 million tons. Peru, Argentina, and Bolivia have the largest supplies. Uncorroborated reports indicate that Chile may have important reserves of both lead and zinc. Lack of development in most South American countries suggests that larger reserves might yet be developed. In Peru the Cerro de Pasco Corp. expanded its mine and mill facilities for the production of zinc and lead. The newly erected facilities at the Chilets mine were put into production in 1952.

Elsewhere numerous exploration and expansion programs have or will result in an increase in world output. Such programs include exploration and plant expansion in Africa, Europe, and Australia.

Europe.—Reserves of measured and indicated lead ore in Europe contain about 10 million tons of lead or approximately one-fourth of the world total. The bulk of these reserves is in Germany, Spain, Italy, and Sweden in Western Europe and in U. S. S. R. (including the Asiatic as well as the European portions), Yugoslavia, and Poland.

Africa.—Lead reserves in Africa total about 3.5 million tons. The largest deposits are in French Morocco, Algeria, and Nigeria (largely undeveloped). Other smaller but important deposits are found in South-West Africa, Northern Rhodesia, Tunisia, French Equatorial Africa, Belgian Congo, and Tanganyika.

Asia.—Lead reserves in Asia are relatively small, totaling about 2 million tons. It is believed, however, that exploration and development work could double the present estimates. The famous Bawdwin mine in Burma, formerly one of the leading world producers of lead, contains the largest reserves. Japan and China are the only other countries with notable reserves.

Australia.—Developed lead deposits in Australia are the largest in the world, containing about 12.5 million tons of lead reserves or nearly one-third of the world total. The important deposits are at Broken Hill and Captain's Flat, New South Wales; Mount Isa, Queensland; and the Read-Rosebery mine, Tasmania.

SUPPLY—DISTRIBUTION

Supply and distribution data may be analyzed by several methods, depending on the purpose for which they are to be used. Total supply, as used in this chapter, consists of the recoverable lead content of domestic mine production, secondary production, metallic imports, and the recoverable lead content of imported ores and concentrates. Total distribution is composed of consumption of primary and secondary metal and metallic exports. Supply-distribution statistics for the 29-year period, 1925–53, are shown in table 8. The total supply of lead during this period was 28,668,000 tons compared with total distribution of 27,778,000 tons. This difference of 890,000 tons arises from the inaccuracies of the consumption series, changes in stocks, and Government purchases of lead for the National Stockpile which are not accounted for in the table.

The most significant development in the lead-supply position of the United States over the 1925–53 period has been the transition from self-sufficiency to dependence on foreign sources for a large part of total supply. Domestic sources—mine production and recovery from scrap—were adequate for United States consumption requirements until the beginning of World War II, when the sharply increased military and industrial demands for lead necessitated importation of large quantities of metal and lead-bearing raw materials. Secondary production of lead has also become increasingly important since World War II. Of the total lead supply for 1940–53 domestic mine production contributed 34 percent; recovery from secondary sources 35 percent; and imports 31 percent.

The total available supply of lead in the United States for the 29 years 1925–53 averaged 989,000 tons a year compared with total distribution of 958,000 tons annually. Comparable figures for the 14-year period 1940–53 were 1,192,000 and 1,095,000 tons, respectively.

PRODUCTION AND CONSUMPTION TRENDS

Lead supplies throughout the world slightly exceed total requirements at present. Even in 1951, when consumption of lead in the United States exceeded supply by a slight margin, the deficit was not due to inadequate supplies but rather to the purchase of large quantities of metal for the National Stockpile. The emergency stocking of lead beyond consumers' requirements in Great Britain further aggravated the apparent shortage. Early in 1952 the true situation—that of adequate or excess supplies—became apparent. Imports of both metal and ores and concentrates were available to the United States in greater quantity, smelter stocks increased, and the United States market price fell below the 19-cent-a-pound ceiling to average 16.47 cents in 1952, 13.48 cents in 1953, and 14.05 cents in 1954.

A review of world mine production versus consumption of primary lead (table 1, p. 4) shows that mine production has increased much more rapidly than has consumption. World mine production of lead (in 1952–53) was approximately 4,120,000 tons, but the consumption of primary lead was only 3,688,000 tons. In 1950 and 1951 consumption exceeded mine production by 40,000 tons (3,700,000 tons mine output versus 3,740,000 tons of primary lead consumption), but in 1948–49 mine output was almost 200,000 tons over consumption. As stocks in the hands of producers and consumers increased, prices declined and competition for markets became very keen. Much of the foreign production is derived from large mines and smelting plants capable of producing metal at a lower unit cost than can most domestic mines. Details of domestic mine costs and grade of ore mined in 1952 were published by the Tariff Commission (*29*), and it was noted that, although average labor costs per ton of ore are lower in the United States than in either Canada or Mexico, the metal content is much lower, and hence the cost per unit of lead (and zinc) is much higher in the United States.

Reflecting the substantially lower prices in the second half of 1952, mine production of recoverable lead in the United States dropped from an annual rate of 408,000 tons during the first half of the year to an annual rate of 373,000 tons in the latter half of the year. Production dropped further to 342,000 tons in 1953 and in 1954 totaled 317,000 tons, the lowest since 1934.

Imports of recoverable lead to the United States in 1954 totaled about 444,000 tons, a substantial decrease from 1953 total of 552,000 tons.

In 1949–53 total supply and total distribution of lead were 24 and 4 percent, respectively, higher than in 1925–29. The most notable changes occurred in the sources of supply. Domestic mine production, secondary production, and imports composed 29, 36 and 35 percent, respectively, of supply in 1949–53 compared with 61, 26, and 13 percent, respectively, in 1925–29. The pattern of distribution has changed, in that exports have been negligible in recent years, whereas in 1925–29 exports comprised about 10 percent of total distribution. It should be noted, however, that the lead comprising these exports had been imported in bond for refining and reexport.

MINE PRODUCTION

Mine-output data are compiled on the basis of lead content in ore and concentrates and are adjusted to account for average losses in smelting. Production calculated on this basis gives a closer measure of actual lead output from year to year than statistics prepared on a smelter or refinery basis and is the most accurate method for assessing the geographic distribution of production. Mine production of lead by leading districts and by States in recent years is shown in tables 6 and 7.

Mine production was the most important single source of lead supply in the United States until 1946, when it was surpassed by secondary production. The outbreak of World War II in 1939 marked the beginning of United States dependence upon large imports to supplement production from domestic mines and secondary sources to satisfy consumer demand for lead.

Mine production declined markedly during 1925–53, and the decline is attributed principally to the substantial exhaustion of reserves in several important producing districts, notably the Tri-State region. The average yearly output for selected periods reveals the extent of the decline in United States mine production: 662,000 tons in 1925–29; 377,000 tons in 1930–39; 444,000 tons in 1941–45; 380,000 tons in 1946–49; and 388,000 tons in 1950–53. Other factors, such as the depression of the 1930's, the shortage of skilled workers during World War II, and low prices in 1953, contributed to the drop in production over the 29-year period, but the major cause would appear to be the depletion of reserves in some areas.

SECONDARY PRODUCTION

Secondary lead is recovered in the United States at primary smelters and refineries as refined lead and in antimonial lead and at secondary plants that process scrap exclusively to produce refined lead, antimonial lead, and other lead alloys (copper-base, tin-base, and others).

Recovery of secondary lead averaged 474,000 tons annually or 36 percent of the total supply in 1949–53 compared with 280,000 tons a year or 26 percent of the supply in 1925–29. Table 8 shows that output of lead from secondary materials did not decline to near the extent that mine production did during the depression years of the 1930's. The constancy of the supply of scrap lead accounted for the maintenance of a relatively high level of production of secondary metal during those years. The larger return of scrap is due to the fact that a major part of the lead consumed each year goes into nondissipative uses and eventually returns to the smelter for reworking and reuse. The nature of scrap recovery contributes to the relative stability of secondary production. Unlike a mine, which closes if the price of lead makes operation unprofitable, a factory that uses lead will continue to market its scrap lead, even if the price is very low; plant scrap must be disposed of, and usually it is more profitable to deliver to a smelter than to a waste dump. Cheap labor also facilitates the collection of scrap during periods of depressed economic activity; however, when a depressed state of industrial activity delays replacement of worn or obsolete lead items, the quantity of scrap entering the market is lessened. Secondary

TABLE 6.—*Mine production of recoverable lead in the United States, 1944–48 (average) and 1949–53, by districts that produced 1,000 tons or more during any year, 1949–53*

[Short tons]

District	State	1944–48 (average)	1949	1950	1951	1952	1953
Southeast Missouri region	Missouri	141, 719	126, 269	133, 680	122, 318	122, 942	125, 273
Coeur d'Alene region	Idaho	70, 488	74, 152	94, 697	70, 570	67, 330	69, 885
West Mountain (Bingham)	Utah	24, 614	32, 600	27, 472	29, 120	34, 328	29, 311
Summit Valley (Butte)	Montana	6, 465	11, 490	15, 679	16, 630	16, 153	16, 767
Tri-State (Joplin region)	Kansas, Southwestern Missouri, Oklahoma.	25, 224	30, 883	31, 157	26, 906	27, 356	13, 273
Metaline	Washington	3, 751	4, 030	7, 445	5, 234	(1)	8, 694
Coso (Darwin)	California	5, 632	4, 928	8, 479	7, 191	(1)	8, 269
Upper San Miguel	Colorado	2, 433	5, 285	7, 780	8, 008	7, 657	7, 440
Park City region	Utah	10, 521	8, 583	7, 538	11, 719	7, 494	4, 735
Big Bug	Arizona	2, 076	3, 330	4, 357	4, 035	4, 135	4, 339
Upper Mississippi Valley	Iowa, Northern Illinois, Wisconsin.	1, 851	2, 046	1, 801	1, 923	3, 532	3, 688
Tintic	Utah	5, 325	6, 676	6, 520	5, 553	4, 279	3, 590
Pioche	Nevada	3, 927	6, 630	6, 761	4, 751	4, 632	3, 306
California (Leadville)	Colorado	4, 850	5, 080	6, 392	5, 996	5, 624	3, 072
Rush Valley & Smelter (Tooele County).	Utah	3, 587	2, 953	1, 393	2, 674	2, 595	2, 753
Warm Springs	Idaho	2, 102	2, 339	2, 648	3, 086	3, 455	2, 583
Red Cliff	Colorado	950	1, 600	2, 110	4, 274	3, 980	2, 500
Northport (Aladdin)	Washington	447	342	237	937	(1)	2, 165
Harshaw	Arizona	1, 472	1, 546	1, 931	1, 668	1, 921	2, 104
Austinville	Virginia	4, 269	3, 313	3, 254	1, 508	3, 792	2, 016
Pioneer (Rico)	Colorado	2, 383	1, 388	1, 138	2, 231	2, 230	1, 871
Kentucky, Southern Illinois.	Kentucky, Southern Illinois.	2, 648	2, 822	1, 526	2, 516	2, 790	1, 849
Creede	Colorado	362	1, 162	1, 422	1, 167	1, 513	1, 696
Bayhorse	Idaho	1, 569	1, 073	1, 679	1, 732	1, 091	1, 484
Central	New Mexico	4, 039	2, 479	2, 315	3, 133	4, 486	1, 460
St. Lawrence County	New York	1, 261	1, 317	1, 484	1, 497	1, 120	1, 435
Sneffels	Colorado	(1)	1, 064	866	1, 094	1, 044	1, 307
Animas	do	2, 437	2, 935	3, 069	3, 963	3, 464	1, 212
Eagle	Montana	638	1, 024	1, 013	(1)	733	1, 179
Ophir	Utah	479	1, 089	948	712	999	1, 157
Breckenridge	Colorado	101	308	347	246	499	1, 056
Hansonberg	New Mexico	24	131	451	753	847	1, 031
Bossburg	Washington	599	2, 011	2, 640	1, 768	(1)	168
Battle Mountain	Nevada	92	1, 290	564	351	907	1
Old Hat	Arizona	4, 835	6, 788	5, 980	4, 241	3, 913	-------
Pima (Sierritas, Papago, Twin Buttes).	do	2, 726	4, 232	2, 996	2, 834	1, 864	-------
Heddleston	Montana	2, 458	2, 335	930	1, 398	1, 251	-------
Warren (Bisbee)	Arizona	9, 692	13, 865	7, 790	1, 606	1, 828	-------
Magdalena	New Mexico	1, 790	1, 162	926	1, 004	1, 046	-------
Aravaipa	Arizona	575	1, 271	1, 498	1, 294	865	-------
Tomichi	Colorado	863	1, 221	645	761	739	-------
Ten Mile	do	1, 415	3, 671	910	6	8	-------
Resting Springs	California	(1)	(1)	(1)	(1)	(1)	-------

[1] Figure not shown to avoid disclosure of individual company operations.

production, by form of recovery and kind of scrap processed in 1952 and 1953, is shown in table 9.

IMPORTS

The United States did not become dependent on imports of foreign lead for any sizable portion of its supply until the beginning of World War II. From 1925 to 1939 the bulk of United States imports was as base bullion and ore and matte, which were entered for smelting and refining and subsequent reexport.

Beginning in 1940, when it became apparent that domestic sources of lead would not meet the greatly increased demand, and continuing to the present a large part of the total United States supply has been obtained from foreign

TABLE 7.—*Mine production of recoverable lead in the United States, 1944–48 (average) and 1949–53, by States*

[Short tons]

State	1944–48 (average)	1949	1950	1951	1952	1953
Western States and Alaska:						
Alaska	153	51	149	21	1	9
Arizona	24, 394	33, 568	26, 383	17, 394	16, 520	9, 428
California	8, 404	10, 318	15, 831	13, 967	11, 199	8, 664
Colorado	19, 123	26, 853	27, 007	30, 336	30, 066	21, 754
Idaho	75, 890	79, 299	100, 025	76, 713	73, 719	74, 610
Montana	13, 180	17, 996	19, 617	21, 302	21, 279	19, 949
Nevada	7, 399	10, 626	9, 408	7, 148	6, 790	4, 371
New Mexico	6, 772	4, 652	4, 150	5, 846	7, 021	2, 943
Oregon	5	12	17	2	1	5
South Dakota	12	4	----------	2	2	10
Texas	59	132	129	43	56	----------
Utah	45, 939	53, 072	44, 753	50, 451	50, 210	41, 522
Washington	5, 024	6, 417	10, 334	8, 002	11, 744	11, 064
Wyoming	1	----------	----------	----------	----------	----------
Total	206, 355	243, 000	257, 803	231, 227	228, 608	194, 329
West Central States:						
Arkansas	9	1	9	33	4	----------
Kansas	7, 776	9, 772	9, 487	8, 947	5, 916	3, 347
Missouri	144, 981	127, 522	134, 626	123, 702	129, 245	125, 895
Oklahoma	14, 302	19, 858	20, 724	16, 575	15, 137	9, 304
Total	167, 068	157, 153	164, 846	149, 257	150, 302	138, 546
States east of the Mississippi River:						
Illinois	2, 972	3, 824	2, 729	3, 160	4, 262	3, 391
Kentucky	165	187	66	107	60	52
New York	1, 261	1, 317	1, 484	1, 500	1, 120	1, 435
Tennessee	40	257	113	14	18	9
Virginia	4, 351	3, 313	3, 254	1, 508	3, 792	2, 016
Wisconsin	1, 361	857	532	1, 391	2, 000	2, 094
Total	10, 150	9, 755	8, 178	7, 680	11, 252	8, 997
Grand total	383, 573	409, 908	430, 827	388, 164	390, 162	341, 872

sources. Imports of all classes totaled 277,000 tons in 1940, and increased to 377,000 tons in 1941 and 489,000 tons (the wartime high) in 1942. The yearly average for 1941–45 was 358,000 tons or 31 percent of total supply. Imports declined to 160,000 tons in 1946 but increased thereafter, reaching an alltime high of 623,000 tons in 1952 and averaging 471,000 tons annually in 1949–53.

The character of lead imports changed completely during World War II, principally because imports were not entered for treatment and reexport but for domestic consumption. Whereas pigs and bars represented only a small portion of total imports up to 1940, from 1940 through 1945 imports of this kind comprised 71 percent of the total. Ore and matte, base bullion, and reclaimed (scrap, etc.) were 24, 4, and 1 percent of all imports, respectively. In 1953 imports of pigs and bars, and ore and matte comprised 70 and 29 percent, respectively, of total imports.

Mexico has been the principal source of United States lead imports throughout the 29 years 1925–53 and during 1940–53 supplied about 40 percent of total imports. Canada is the next largest source of United States imports, followed by Peru and Australia. Bolivia, South-West Africa, and Yugoslavia have also shipped sizable quantities of lead as metal or ore to the United States in recent years. Table 10 shows imports of lead, by classes and major countries, for 1940–53.

TABLE 8.—*Salient statistics of the lead industry in the United States, 1925–53* [1]

[Thousand short tons]

Year	Supply				Distribution		
	Total	Recoverable mine production	Recoverable secondary	Imports [2]	Total	Consumption [3]	Exports [4]
1925	1,031	684	227	120	1,030	915	115
1926	1,106	684	277	145	1,122	1,032	90
1927	1,100	665	276	159	1,108	971	137
1928	1,090	627	309	154	1,140	1,011	129
1929	1,073	648	311	114	1,124	1,038	86
1930	890	558	256	76	894	834	60
1931	692	405	235	52	666	634	32
1932	524	293	198	33	504	473	31
1933	504	273	224	7	567	538	29
1934	508	287	208	13	554	541	13
1935	624	331	270	23	675	659	16
1936	659	373	263	23	780	753	27
1937	779	465	275	39	855	826	29
1938	657	370	225	62	680	625	55
1939	741	414	242	85	752	667	85
1940	994	457	260	277	821	782	39
1941	1,235	461	397	377	1,086	1,050	36
1942	1,308	496	323	489	1,071	1,043	28
1943	1,111	453	342	316	1,130	1,113	17
1944	1,063	417	331	315	1,155	1,119	36
1945	1,051	391	363	297	1,067	1,051	16
1946	888	335	393	160	957	956	1
1947	1,121	384	512	225	1,174	1,172	2
1948	1,234	390	500	344	1,135	1,134	1
1949	1,216	410	412	394	961	958	3
1950	1,451	431	482	538	1,242	1,238	4
1951	1,161	388	518	255	1,187	1,185	2
1952	1,484	390	471	623	1,134	1,131	3
1953	1,373	342	487	544	1,207	1,202	5
Average	989	442	331	216	958	919	39

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[1] Sources: Bureau of Mines (with the cooperation of the Geological Survey), Materials Survey—Lead: 1951; Yearbooks of the American Bureau of Metal Statistics; and Bureau of Mines Minerals Yearbook.

[2] Factored to include 95 percent of the lead content of ores, mattes, and concentrates, and 100 percent of pigs, bars, base bullion, and scrap. Includes material entering country under bond as well as that for consumption.

[3] Includes primary and secondary metal; data are from various sources.

[4] Includes lead that entered the United States under bond.

TABLE 9.—*Secondary lead recovered in the United States, 1952–53*

[Short tons]

Recoverable lead content of scrap processed			Lead recovered from scrap processed		
Kind of scrap	1952	1953	Form of recovery	1952	1953
New scrap:			As refined metal:		
Lead-base	51,380	49,902	At primary plants	3,070	4,211
Copper-base	8,083	8,085	At other plants	137,032	122,363
Total	59,463	57,987	Total	140,102	126,574
Old scrap:			In antimonial lead:		
Battery lead plates	254,827	247,332	At primary plants	35,145	36,749
All other lead-base	130,302	152,897	At other plants	187,806	199,806
Copper-base	26,679	28,498			
Tin-base	23	23	Total	222,951	236,555
Total	411,831	428,750	In other lead alloys	93,048	92,379
			In copper-base alloys	14,479	30,826
Grand total	471,294	486,737	In tin-base alloys	714	403
			Total	108,241	123,608
			Grand total	471,294	486,737

TABLE 10.—*United States imports*[1] *of lead, by classes and countries, 1940–53*

[Thousand short tons]

Year	Pigs and bars						Ore and matte[2]					Base bullion	Reclaimed, scrap, etc.	Grand total
	Mexico	Canada	Peru	Australia	Others	Total	Canada	Peru	Australia	Others	Total	Total	Total	
1940	129	------	18	4	1	152	36	18	17	40	111	19	--------	282
1941	100	90	41	43	------	274	23	7	23	29	82	25	--------	381
1942	192	69	22	83	------	366	31	11	21	16	79	44	3	492
1943	215	------	20	9	------	244	20	3	20	27	70	5	--------	319
1944	168	------	54	1	------	223	42	11	27	14	94	-----	3	320
1945	160	19	34	14	------	227	26	15	18	11	70	-----	3	300
1946	54	23	16	8	15	116	24	5	8	7	44	-----	2	162
1947	86	59	1	11	3	160	15	10	7	19	51	1	16	228
1948	98	54	24	30	41	247	8	9	9	38	64	7	29	347
1949	126	56	35	17	41	275	10	9	9	79	107	3	15	400
1950	221	108	32	22	59	442	9	16	10	42	77	3	20	542
1951	37	57	32	14	39	179	7	17	7	37	68	2	9	258
1952	199	105	42	83	82	511	12	28	9	56	105	-----	12	628
1953	141	49	52	70	73	385	39	33	21	68	161	1	5	552

[1] Source: Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of U. S. Department of Commerce. Data include lead imported for immediate consumption plus material entering country under bond.

[2] Not factored as in table 8; hence totals will not coincide with those in that table.

TARIFF

For well over a century there has been a tariff on lead ranging from a low of 15 percent ad valorem to a high of 3 cents a pound, except for periods when such tariffs were suspended by act of Congress. The import duty set by the Tariff Act of 1930 on lead-bearing ores, flue dust, and mattes was 1½ cents per pound (lead content) and on lead bullion, pigs, bars, scrap lead, antimonial lead, type metal, babbitt metal, solder, and alloys not specifically provided for 2⅛ cents per pound. In accordance with the Mexican Trade Agreement of January 30, 1943, these rates were reduced to ¾ cent and 1$\frac{1}{16}$ cents per pound, respectively. In June 1948 these duties were suspended for 1 year by act of Congress. As the Congress took no action on a bill to extend the suspension beyond June 30, 1949, the expiration date of the original legislation, the import duty of 1$\frac{1}{16}$ cents a pound on pig lead and ¾ cent a pound on lead in ores and concentrates was reinstated automatically on July 1 and continued throughout 1950. Abrogation of the Mexican Trade Agreement, effective January 1, 1951, restored the full duties established by the Tariff Act of 1930. On June 6, 1951, the Torquay Agreement reestablished the 1$\frac{1}{16}$-cent and ¾-cent duties on pig lead and lead in ores and concentrates, respectively. Owing to shortages of lead in the United States, import duties were suspended by congressional act on February 12, 1952, until March 31, 1953, or until the end of the present emergency, whichever came first.

The act stipulated that the rates of June 6, 1951, were to be reimposed if the average market price of lead fell to 18 cents a pound for a calendar month. The Tariff Commission informed the President on June 6 that the average price of pig lead for May was below 18 cents a pound, and on June 26, 1952, the President signed the order ending suspension of duties.

The high level of imports established in 1952–53, together with the depressed state of the domestic lead- (and zinc-) mining industry, stimulated much study of various protective tariff proposals and plans whereby the domestic industry might be aided. In September 1953 a committee formed by a large proportion of the lead and zinc mining industry petitioned the Tariff Commission for an investigation under the "escape-clause" provisions of the Trade Agreements Extension Act of 1951, which permits the President, upon recommendation of the Tariff Commission, to increase duties by up to 50 percent of the rates prevailing on January 1, 1945. Congressional committees also adopted resolutions instructing the Tariff Commission to investigate imports of lead and zinc and their effects on domestic output. The report of this investigation (*29*) was factual and contained no recommendations. It showed that the existing tariff structure restricted imports but slightly and that during the past decade a substantial part of all lead imports were exempted from duty.

The report on the "escape-clause" provisions of the Trade Agreements Extension Act of 1951

(*30*) recommended that import duties on most lead and zinc materials be increased 50 percent above the rates existing on January 1, 1945. The President did not accept the recommendations of the Tariff Commission (*32*); instead he outlined an expanded stockpiling program for strengthening the lead and zinc industry as an integral part of the Nation's defense-mobilization base.

CONSUMPTION, USES, AND SUBSTITUTES

Domestic consumption of lead, including both primary and secondary, has increased substantially during 1925–53. In 1950–53 the average annual consumption was 1,189,000 tons, or 20 percent higher than the 1925–29 yearly average.

Consumption of lead was at a high level during the boom period of the 1920's, averaging 993,000 tons a year in 1925–29. In the depression of the 30's consumption declined sharply, falling to a low of 473,000 tons in 1932. From 1933 to 1937 consumption increased steadily as business conditions improved, reaching a high of 826,000 tons in 1937. The general recession of 1938 halted industrial recovery, and lead usage decreased to 625,000 tons. It was not until 1941 that consumption exceeded that of 1937. During World War II lead consumption was high, averaging 1,075,000 tons for the 5 years 1941–45. Consumption declined markedly in 1946, but in the ensuing 7 years the annual average was even higher than during the war period; in 1950 consumption totaled 1,238,000 tons, an alltime peak.

An unusual combination of physical and chemical properties has given lead a wide range of industrial uses. The most important of these properties are: Softness and extreme workability, high specific gravity, alloying properties, low initial cost and high recoverability, high boiling point and low melting point, good corrosion resistance, and impenetrability by shortwave radiation.

The principal uses of lead (*8, 19*) are for storage batteries, tetraethyl lead, cable covering, paint pigments, building construction, ammunition, and various alloys, chiefly solder, bearing metals, and type metal. War uses, though numerous, are largely the same, except for increased substitution for scarcer metals, especially to replace copper or tin in bearing metal, terneplate, solder, and plumbing supplies. Two of the larger uses—paint pigments and tetraethyl fluid—are dissipative. The greater part of the lead used in other forms is recoverable and usually returns to supply as secondary metal. Table 11 shows the quantities of lead used for various purposes in 1948–53.

Lead is widely utilized alloyed with certain metals, principally antimony and tin. The common alloys of lead are roughly classified into bearing alloys, solders, type metals, and low-melting alloys. Numerous other alloys of lead are extensively employed in industry. The most notable contain varying quantities of antimony and are used for cable sheathing and storage-battery grids. Lead also finds wide usage in machine brasses and bronze, approximately 30,000 tons being consumed annually for this purpose.

Large quantities of lead are consumed in the production of litharge, red lead, and white lead. Litharge is used principally in storage batteries in both the positive and negative plates, either alone or mixed with finely divided metallic lead. Smaller quantities are used in ceramics, chrome pigments and varnish, insecticides, oil refining, and rubber. Red lead is used chiefly in storage batteries mixed with litharge and metallic lead in the positive plates and in paints. White lead is used almost exclusively as a paint pigment.

Another pigment, basic lead sulfate, is used extensively in leaded zinc oxide and as a stabilizer for certain plastics.

An important application of lead is nuclear shielding (*19*) against certain types of dangerous radiation, chief among these being gamma rays. The first consideration in preventing penetration of rays is density. Lead has the advantage of being the densest of any commonly available material. Where space is at a premium and utmost radiation protection is paramount, lead is prescribed. Another advantage is that lead does not become contaminated and may be used continuously without becoming radioactive and emitting its own harmful rays. For this reason it is important that lead for shielding purposes be free of impurities, particularly those that may become radioactive upon exposure to high-energy radiation. The atomic reactors operated by the Atomic Energy Commission employ a combination of concrete, lead, cadmium, and space to protect operating personnel from all types of radiation, including alpha, beta, gamma, and neutron rays. The two last are particularly dangerous because of their ability to ionize matter and hence injure living tissue. Lead is particularly effective in absorbing gamma rays. Cadmium or a hydrogenous material, such as paraffin or water, is used to shield against neutrons; but, since gamma rays are emitted when neutrons are absorbed, it is necessary to stop the gamma ray with lead shielding. Containers used in handling and shipping radioactive isotopes at the Brookhaven National Laboratories alone weigh over 1,000 tons.

The greater part of lead used as metal ultimately returns to supply as secondary metal, but that used as pigments and chemicals is wholly lost. Of the chemicals containing lead, tetraethyl lead is the most important. Tetraethyl lead is a colorless, heavy liquid that forms the active ingredient of the principal "antiknock compounds" added to gasoline to improve their antiknock qualities and efficiency. Premium gasoline for automobiles contains 2 to 3 cc. of tetraethyl lead, while most other gasolines contain up to 1.5 cc. per gallon. The improved efficiency of gasoline as a result of this additive results in a huge economy in the amount of fuel used. In 1950 it was estimated that the use of tetraethyl lead conserved 4 billion gallons of gasoline. In 1953 over 162,000 tons of lead was consumed in this use, and there is every indication that the use will increase.

There are a number of substitutes for lead in many of its uses. In the storage-battery industry, the largest consumer of lead, there are some competitive products; but these have different electrical characteristics, and the raw materials for their manufacture are not available in adequate quantities to replace any sizable quantity of lead in battery usage. From the conservation standpoint, the lead consumed in batteries does not represent a complete depletion of total supply, for about 90 percent of the lead content is recovered as secondary metal.

In the cable-covering industry—the third largest user of lead—a substitute has been developed that gives lead considerable competition. This new covering is extruded polyethylene, which is replacing lead telephone-cable sheathing for inside use. There are also various combinations of polyethylene with lead and aluminum sheathing for outside and underground use. There has been some substitution of aluminum in telephone and power cable, but numerous economic factors and technical shortcomings will have to be overcome before the substitution becomes widespread. Progress is being made in meeting these problems, however, and it seems likely that the next 10 years will see continued substitution of plastics and aluminum for lead.

The development of substitutes for lead in paint pigments has proceeded rapidly during the past few years. The lead content of paints has been reduced, and in many instances lead has been eliminated from paint formulations. Titanium and zinc pigments are the chief competitors of lead in pigments.

A substantial quantity of lead in the building industry could be replaced by aluminum, synthetic resins, and other nonmetallic materials. The use of lead in foil manufacture has already been significantly reduced, mainly through replacement by aluminum foil. Plastic films can also be substituted for lead foil.

Several materials may be used as a substitute for lead in caulking. Nonmetallic fibers and plastic compounds are probably the best materials to be substituted for lead in this application, although aluminum wool may also be utilized.

The quantity of lead used in type metal could be reduced substantially by substitution of magnesium plates.

The use of aluminum and aluminum-coated steel as replacements for lead in terneplate and the development of new bearing alloys using noncritical metals and materials may be expected in the future.

EXPORTS

Exports of lead have been relatively unimportant in the distribution of United States lead supplies over the 1925–51 period. Only during 1925–29 have they averaged 111,000 tons a year; from 1930 through 1953 United States shipments to foreign countries were about 24,000 tons annually and from 1946 to 1953 slightly over 2,000 tons a year.

PRICES

Lead prices have fluctuated considerably during the 29 years 1925–53. The average price of pig lead, New York, was 7.47 cents per pound in 1925–29 and 4.91 cents in 1935–39, with an intervening low quotation of 2.65 cents in 1932. During World War II the price of lead was controlled by the Office of Price Administration; from January 1942 to June 1946 the ceiling price was 6.50 cents per pound and from June to November 8.25 cents. The total average price for all grades of refined lead (weighted average price plus the increment of subsidies for overquota production and special mine contracts paid by the Office of Metals Reserve) was about 8.50 cents per pound throughout the war years. Prices advanced markedly in the postwar period, reaching an alltime high of 21.50 cents per pound in late 1948 and averaging 15.34 cents for 1947–50.

Following the outbreak of war in Korea in June 1950 lead prices increased steadily, attaining a high of 17.00 cents in October. Ceiling prices were established for lead on January 26, 1951, at the highest price (for each seller) at which sales were made between December 19, 1950, and January 25, 1951; thus a number of ceiling prices were maintained. Most lead sales, however, were at 17.00 cents per pound, common lead, New York. This price remained in effect until October 2, 1951, when the Office of Price Stabilization permitted an increase of 2 cents per pound. Increased supplies and lower consumption brought about substantial reduc-

TABLE 11.—*Consumption of lead in the United States, 1948–53, by products* [1]

[Short tons]

	1948	1949	1950	1951	1952	1953
Metal products:						
Ammunition	49, 635	24, 111	38, 438	40, 242	36, 182	45, 147
Bearing metals	42, 594	29, 189	38, 241	35, 410	36, 545	38, 591
Brass and bronze	23, 239	14, 946	21, 461	29, 858	25, 807	26, 203
Cable covering	171, 654	144, 340	131, 989	131, 863	142, 571	146, 565
Calking lead	31, 473	34, 944	53, 450	46, 544	45, 150	48, 236
Casting metals	8, 974	12, 672	19, 295	22, 497	18, 017	12, 906
Collapsible tubes	11, 071	8, 692	13, 386	13, 657	10, 095	11, 583
Foil	3, 203	2, 503	3, 941	2, 881	2, 124	4, 410
Pipes, traps, and bends	39, 843	29, 858	41, 361	33, 095	29, 465	28, 693
Sheet lead	31, 559	27, 144	30, 778	31, 210	28, 697	30, 476
Solder	71, 025	62, 104	94, 606	82, 465	72, 664	78, 743
Terne metal	3, 278	3, 256	3, 805	2, 051	1, 812	3, 200
Type metal	26, 279	20, 695	24, 776	28, 236	27, 413	26, 729
Total metal products	513, 827	414, 454	515, 527	500, 009	476, 542	501, 482
Storage batteries:						
Antimonial lead	203, 869	175, 308	212, 464	199, 838	187, 506	191, 753
Lead oxides	150, 536	138, 410	185, 945	175, 546	163, 424	175, 822
Total batteries	354, 405	313, 718	398, 409	375, 384	350, 930	367, 575
Pigments:						
White lead	30, 970	18, 400	36, 181	25, 578	22, 943	17, 775
Red lead and litharge	80, 356	70, 832	101, 974	88, 031	76, 742	88, 649
Pigment colors	10, 832	8, 400	13, 464	12, 796	12, 839	12, 859
Other [2]	20, 230	9, 515	14, 768	13, 099	9, 775	10, 307
Total pigments	142, 388	107, 147	166, 387	139, 504	122, 299	129, 590
Chemicals:						
Tetraethyl fluid	83, 809	94, 644	113, 846	128, 407	146, 723	162, 443
Miscellaneous chemicals	10, 280	4, 191	11, 680	6, 949	3, 996	6, 976
Total chemicals	94, 089	98, 835	125, 526	135, 356	150, 719	169, 419
Miscellaneous uses:						
Annealing	6, 132	4, 935	6, 456	6, 656	5, 084	5, 280
Galvanizing	1, 995	1, 228	2, 426	2, 173	2, 002	2, 029
Lead plating	2, 274	997	1, 521	1, 444	1, 037	987
Weights and ballasts	6, 290	4, 627	6, 870	7, 913	7, 660	8, 244
Total miscellaneous	16, 691	11, 787	17, 273	18, 186	15, 783	16, 540
Other unclassified uses	12, 495	11, 733	14, 859	16, 354	14, 522	16, 998
Grand total	1, 133, 895	957, 674	1, 237, 981	1, 184, 793	1, 130, 795	1, 201, 604

[1] Based on survey of consumers by Bureau of Mines.

[2] Includes lead content of leaded zinc oxide production.

tions in lead prices in the spring and summer of 1952. On April 29 the price of lead declined to 18.00 cents, and subsequent drops brought it to 15.00 cents on May 12. The quoted price advanced to 16.00 cents on June 24, where it remained until October 7, when it again dropped to 15.00 cents. Limited buying caused further declines in the following weeks, and on October 22 the price was down to 13.50 cents—the low point of the year. In November and December the market improved, and on December 30 the quotation for lead was 14.75 cents per pound.

Price cuts in early 1953 brought the quotation to 13.50 cents on February 2 and 13.00 cents a pound on March 4. Thereafter the price fluctuated between 12.00 and 14.00 cents a pound, becoming stable at 13.50 cents on September 17, where it remained throughout the balance of the year. Early in 1954 the price dropped, reaching a low of 12.50 cents on February 18; thereafter, however, the price increased in a series of ¼-cent advances to 15 cents a pound on October 5, where it remained for the balance of the year. Table 12 shows the average yearly quotation for lead in 1925–53.

TABLE 12.—*Average yearly quoted price of common lead, New York, 1925–53*

[Cents per pound]

1925–9.02
1926–8.42
1927–6.75
1928–6.31
1929–6.83
1930–5.52
1931–4.24
1932–3.18
1933–3.87
1934–3.86
1935–4.06
1936–4.71
1937–6.01
1938–4.74
1939–5.05
1940–5.18
1941–5.79
1942–6.48
1943–6.50
1944–6.50
1945– 6.50
1946– 8.11
1947–14.67
1948–18.04
1949–15.36
1950–13.30
1951–17.49
1952–16.47
1953–13.48
1954–14.05

SELF-SUFFICIENCY AND STRATEGIC CONSIDERATIONS IN THE UNITED STATES

From 1925 through 1953 about 45 percent of the total supply was from domestic mine production, 33 percent from secondary recovery, and 22 percent from imports. During the 4 years ended in 1953 mine production was only 28 percent of the supply, and imports and secondary lead each furnished 36 percent of the supply. In addition to heavy dependence upon imports since the beginning of World War II, it should be borne in mind that the lead recovered in secondary operations was not wholly from domestic sources; it represents recovery from new lead used in earlier years, which was from both domestic and foreign sources.

Supplies considerably exceeded requirements during the early years of World War II and particularly in 1952–53 (see table 8) and thus permitted accumulation of large stocks. If stock accumulations and exports had been eliminated, actual consumption would have required about 257,000 tons of imported lead annually, rather than the 368,000 tons actually imported. Thus, while the United States is not self-sufficient in lead, it does not depend on foreign sources to the degree recent imports suggest.

A review of imports shows that total imports from Canada and Mexico in 1950–53 averaged almost 250,000 tons, which suggests that, under emergency conditions, the actual deficit might be met by imports from those 2 countries with but minor imports from other sources.

GOVERNMENT PROGRAMS

REGULATIONS

The Government regulated the lead industry during World War II and again in 1951–52 to provide adequate supplies for all essential purposes. In World War II measures were taken to increase imports greatly, expand or at least maintain domestic mine production, increase secondary recovery, decrease exports, and control consumption.

The expansion of imports was undertaken even before the United States entered the war. Import controls were instituted in December 1941, after which time lead could be imported only by the Metals Reserve Company or other Government agencies. Duties were suspended on Government purchases after May 1942. Total recoverable imports increased from 85,000 tons in 1939 to 277,000 tons in 1940 and 489,000 tons in 1942. For 1940–45 imports averaged 345,000 tons a year, (31 percent of supply from all sources) compared with average yearly imports of 46,000 tons (7 percent) in 1935–39.

Various measures were undertaken to stimulate domestic mine production while maintaining ceiling prices. The most important of these measures—the Premium Price Plan—was designed primarily to increase production by paying premium prices for production above a certain quota, which was based on 1941 output. Through payment of these premiums, marginal and submarginal ores were mined; and tailings piles—previously unprofitable to treat—were reworked. In addition, some new and idle mines were brought into operation, and extensive development was undertaken at yet other mines. Production increased from 414,000 tons in 1939 to 496,000 tons in 1942 but declined thereafter to 391,000 tons in 1945. Mine production for 1942–45 averaged 439,000 tons per year—only 12 percent above the 1935–39 average yearly output. A serious manpower shortage in the lead mines, brought about by the drafting of skilled workers and a general migration of labor away from the mines to higher paying industries, was partly responsible for declining mine production after 1942.

During the life of the Premium Price Plan (from February 1942 through June 1947) 45 percent of all domestically mined lead received premiums. Payments totaling about 64 million dollars were made on 1,000,600 tons of overceiling lead production. The weighted average price paid for all lead during the period was 8.84 cents, of which premium payments totaled 2.61 cents a pound.

Additional measures undertaken to increase mine production were the exploratory programs of the Geological Survey and the Bureau of Mines of the United States Department of the Interior. These programs indicated several deposits of value, thus in part replenishing reserves heavily depleted during the war years. Large supplies of lead were available for consumption throughout the greater part of World War II; no serious shortages developed, and as a result lead was available as a substitute for more critical metals in certain uses.

Lead was placed under export control in March 1941, more to keep supplies from going to the Axis countries than from any serious need for conservation.

In May 1941 General Metals Order 1, designed to prevent accumulation of excessive inventories, was made applicable to lead. General Preference Order M–38, issued in October of that year, placed lead under priority control. This order, with subsequent modifications and additions, controlled the allocation of lead, including scrap, for essential military and civilian uses; prohibited or restricted its utilization for certain purposes; created a reserve pool; established inventory control; and required producers and distributors to file schedules of proposed shipments with the Director of Priorities. The order was administered after January 1942 by the War Production Board, successor to the Office of Production Management, originator of the measure.

Lead was first recommended for stockpiling in May 1941, when the Office of Production Management proposed that the Metals Reserve Company acquire 50,000 to 100,000 tons. Government stocks, built up largely from imports, increased monthly from 5,000 tons in August 1941 to a peak of 266,000 tons in April 1943. It was not until the latter part of 1944 that requirements necessitated continued withdrawal of large quantities of metal from the stockpile; by July 1945 the inventory had decreased to 65,000 tons. Thus the Government stockpiling program served the purpose for which it was intended; that is, it was effective in meeting critical wartime needs. In the postwar period stocks were made available to consumers, and in 1948 the small quantity remaining was transferred to the National Stockpile.

In 1951 the Government exercised limited controls over the lead industry owing to concern over possible shortages for defense requirements. Lead was brought under inventory control on February 16 through National Production Authority Order M–38. This order was amended on May 28 to set forth certain limitations on the acceptance of rated orders, so as to provide for equitable distribution of such orders, to provide for the creation of a setaside reserve of pig lead to meet emergency situations, and to establish limitations on the use of lead and lead products. The amended order beginning June 1 restricted the use of lead to not more than 100 percent of the average monthly quantity consumed during the first 6 months of 1950, and prohibited its use for any new purposes. Order M–76, issued July 26, placed the supply of soft pig lead produced by primary refiners under allocation control. Effective November 1 imported pig lead was also placed under allocation control by Order M–76 as amended. All NPA restrictions on lead were completely revoked on May 15, 1952, as increased supplies brought about by peak scrap recovery and the renewal of large-scale importations appeared to have overcome the shortage that existed in 1951.

BUREAU OF MINES EXPLORATION PROGRAM

During World War II the Bureau of Mines, under authorization of the Congress, undertook an exploration-development program that included lead. Over 2,200 lead and zinc properties were examined, of which 148 were drilled or otherwise explored. About 2,150,000 tons of lead ore, 13,600,000 tons of lead-zinc ore, and 6,600,000 tons of zinc ore were indicated. The most significant accomplishments were ore discoveries in the Metaline district of northeastern Washington and the Upper Mississippi Valley near Galena, Ill. Subsequent private exploration greatly increased the indicated zinc-lead reserves and resulted in increased output of lead and zinc ores from the two areas.

GOVERNMENT EXPLORATION AND EXPANSION PROGRAMS UNDER DEFENSE ACT OF 1950

On December 4, 1950, the Defense Minerals Administration was established to stimulate the production of critical metals and minerals needed for national defense. The Defense Minerals Exploration Administration was subsequently created and given responsibility for mineral-exploration programs, and the Defense Materials Procurement Administration—a division of the General Services Administration—at the same time was assigned responsibility for production expansion and procurement as they applied to minerals and metals. The DMEA financed up to 50 percent of the total cost of approved lead and zinc projects and had through 1953 helped to finance 175 mineral-exploration projects totaling $7,297,000 in cost. A directive issued by the Office of Defense Mobilization banned loans on lead and zinc applications received after May 15, 1953, because excess supplies of both metals were available. Loans were resumed, however, March 23, 1954, because it was realized that additional reserves of ore would add to the security of the Nation.

The DMPA was responsible for making purchase contracts (including subsidies to high-cost producers), for granting priority ratings for production machinery and equipment, and for recommending production expansion, operating loans, and certification of certificates of necessity for assistance through tax-amortization programs. The DMPA program was greatly reduced in early 1953; and on August 14 its remaining function—that of completing contracts in negotiation and of servicing existing contracts—was transferred to the Emergency Procurement Service of General Services Administration. Through December 1953 purchase commitments involving 26,250 tons of lead at guaranteed prices had been made with a Guatemalan producer, and a 15-cent-floor price and purchase commitment for 2,460 tons of lead had been guaranteed to an Idaho company. Loans totaling $820,000 were certified for 9 domestic and 1 foreign producer, with the objective of increasing annual metal output by 5,940 tons of lead and 27,750 tons of zinc. In all, 41 tax-amortization programs were certified, which would assist in plant expansion to provide annually about 38,000 tons of additional lead and 182,000 tons of additional zinc.

OUTLOOK

Total world lead supplies, currently at an alltime high, exceed requirements, despite greatly reduced mine production in the United States. World reserves of developed and inferred ore are adequate to meet foreseeable uses for the next 30 years, and the potential reserves of areas at present undeveloped are believed even greater.

Future United States supply will be derived in yet larger part from imports and secondary sources, as consumption is expected to increase while domestic mine production remains relatively constant. Government purchases of newly mined domestic lead should stimulate domestic mine production to about 330,000 tons in 1955, but over the ensuing decade it is likely that domestic mines will average about 300,000 tons annually.

Domestic reserves are adequate to meet this or even slightly higher output. In evaluating the long-run reserve and productive capacity of the United States, it must be realized that existing estimates of reserves understate the resource base. If the price-cost ratio of lead increases, domestic production and domestic reserves will be stimulated. Under such conditions new sources of ore supply will be developed in known and new mining districts. New techniques will also be developed to make the mining of lower grade ore profitable but whether at a rate to maintain or augment present production for over 25 years or so is unknown.

Consumption of lead in the United States in 1954 will approximate 1,100,000 tons. In 1955 it is expected to be about 1,200,000 tons, because of increased demand for automobile batteries, building construction, and tetraethyl lead. Projecting requirements on the present use pattern with normal population growth and assuming a high level of industrial activity, the total lead demand in the United States should be approximately 1,275,000 tons by 1960 and 1,450,000 by 1975.

Elsewhere in the world consumption should increase considerably owing to improved standards of living as well as increased population in many areas.

World mine production of lead can be expected to exceed 2 million tons annually in the future. Should the need for greater supplies arise, output can be expanded considerably as important new discoveries of lead have been made in recent years. The total world output will probably not decrease to any marked extent unless voluntary restrictions by producers limit it.

PROBLEMS

The major problem in the lead industry is to maintain an ample supply of ore and metal produced at prices that stimulate consumption and rising standards of living throughout the world.

The distress caused by the current oversupply can best be met by action to reduce the costs of production and by greater development of markets. In the long run, the problem in supply will be one of shortage. The easily discovered deposits have been found. Those yet to be found require intensive scientific search based on a sound and increasing knowledge of ore occurrences, geophysical guidance, and sound exploration practice.

In mining, numerous research problems to effect lower costs and improved extraction present themselves. These involve improved methods of ore breaking and transport, as well as more effective and cheaper means of ground support.

In milling there is constant need for research to lower the cost of fine crushing and grinding; research must also be continually focused on slime recovery of the sulfides and overall recovery of the complex and difficult sulfide and oxide ores.

Market development offers an area of unlimited research to expand the demand for lead and its alloys and compounds.

The lead industry needs better information on which to base industry decisions. More detailed statistics should make more accurate forecasts possible and help assure that productive capacity remains at an optimum rate. Such statistics, to be of maximum value, should apprise the industry of the total situation with relation to mine production, smelter output, and world requirements much sooner than at present.

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LIGNITE AND PEAT

By

Staff, Division of Solid Fuels

THE FIRST STAGE in the metamorphosis of vegetable matter to coal is peat. As it has been for millions of years, peat is still being formed today in swampy regions where mild temperatures favor luxurious plant life. Lignite, in turn, is the first coalified product of peat and, in the United States at least, the lowest rank of combustible matter that is used for burning, gasifying, and coking—the normal large-scale outlets for coal.

Summary

Peat—the first stage in the coalification process—differs from coal in that it consists predominantly of water. Peat, containing about 90 percent water, has only about one-quarter of the peat substance as plant remains, and these vary greatly depending upon the botany of the areas where it is found. On the other hand, lignite contains only about 30 to 40 percent water. Like all material of vegetable origin, peat and lignite contain carbon, hydrogen, oxygen, nitrogen, sulfur, and other elements in small amounts.

Of the 14 billion tons of known reserves of peat in 26 States of the United States, over 10 billion occur in Minnesota, Wisconsin, and Michigan. Lignite deposits have been found only in the Northern Great Plains, the Rocky Mountains, and the Gulf and Pacific areas.

Although peat has been used as a fuel in other countries, the abundance of higher rank fuels in the United States has kept peat out of the fuel and energy fields. Instead, it is used primarily for soil improvement, as a conditioner and filler in mixed fertilizers, as mull or litter material for domestic animals, as packing material, and in chemical applications such as filtering and tanning. Montan wax can be extracted from some kinds of peat.

Lignite, obtained primarily in strip-mining operations, can be used for heat and power generation, as an absorbent of liquids from gases, as a source of industrial gas, and to make industrial carbon for decolorizing and refining sugar.

Montan wax can also be derived from lignite. Unlike bituminous coal, lignite does not yield coke upon carbonization but a char that can be used as a briquetted fuel and as activated carbon.

BACKGROUND

ORIGIN AND CLASSIFICATION OF COAL

Peat and coal are classified according to their nature and chemical composition; these in turn depend on the kind of original plant material, its partial decay in the peat swamp, and subsequent metamorphism by geological processes throughout the ages. The following description of the origin and nature of peat and coal is adapted from a Bureau of Mines publication, entitled "Facts About Coal."[1]

FORMATION OF PEAT AND COAL

Although coal is commonly thought of as a mineral, it differs from the usual minerals, such as stone and iron ore, because it is of organic origin, meaning that coal was formed from the remains of living things—trees, shrubs, herbs, and vines—that grew millions and millions of years ago during periods of mild, moist climate. During those periods there was heavy growth of trees, ferns, and other plants in swamps and bogs. Century after century the vegetation died and accumulated. Buried to a gradually increasing depth each year by new accumulations, the remains of roots, trunks, branches, and leaves changed gradually to peat, just as decaying vegetable matter is doing today in the Dismal Swamp of Virginia and North Carolina and in smaller swamps and bogs in many areas.

Peat is the first product when organic matter begins to change to coal. In a block of peat one can often see, with the naked eye, woody fragments of stems, roots, and bark. As peat is buried, it is cut off from the oxygen in air, and rapid decay of its organic matter is prevented by slowing bacterial action. The weight of more vegetation falling on the peat helps to compress and solidify it, as does the weight of water when the deposit sinks below a lake or sea, as has often happened. Sometimes mineral sediments have settled from muddy flood waters while vegetable matter was accumulating and formed "partings" or layers of shale in the coal vein. At the end of coal-forming periods swamps remain flooded for a long time; and earthy sediments are deposited in thick beds over the peat, further compressing it and starting "coalification," the coalmaking process.

Coalification is extremely slow when it depends mainly upon pressure generated by overlying rock. For this reason, many coal deposits are still of "low rank," that is, in an early stage of coalification, although they have been buried millions of years. A few deposits, however, are situated where profound movements of the earth's crust occurred during periods of mountain building. This rock movement generated much additional heat and pressure, producing "high-rank" coals, such as medium- and low-volatile bituminous coal and anthracite. Sometimes additional heat and pressure came from the movement of molten rock oozing from deep, hot regions of the earth into its outer crust.

TYPES OF PEAT

Peat is extremely varied in nature and composition due to climatic conditions and differences in the types of plants growing in peat-forming areas. The physical form of peat varies greatly, but all raw peat contains 85 to 92 percent water. Its rate of accumulation in an active swamp has been estimated at about 10 feet in 2,500 years. When dry, peat is combustible, burning with a yellow flame during the first part of its complete combustion and with a red glow similar to wood during the latter stage. Although no general agreement exists on the classification of peats according to their botanical characteristics, they may be divided, according to gross botanical composition, into the four following groups:

Sphagnum.—Peat of this type is formed mainly of several species of sphagnum (moss), usually intermixed with small quantities of various plants and shrubs, such as ferns and sedges, orchids, sundews, pitcherplants, and often cranberry. Sphagnum peat is usually loose, spongy, and often layered; it varies from light grayish brown to deep brown in color. It is nearly always strongly acid.

Hypnum and Other Mosses.—These peats are composed mostly of disintegrated plants of hypnum, often associated with other mosses and with intermingled rootlets of sedges and other flowing plants. They are formed chiefly in areas where the ground is only slightly acid, neutral, or slightly alkaline. The material is brownish or drab, light, spongy, and matted, and often laminated and porous.

Reed Sedge.—Deposits of this type are formed mainly in shallow areas around the border of water-filled depressions, of which they gradually take possession, or in boggy meadows. Important constituents of this group are plants of the sedge family, cattail (typha), various grasses,

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[1] A selected bibliography of lignite and peat appears at the end of this chapter.

and miscellaneous shore and swamp-loving plants of other families. In their normal development, peats of this type are rather fibrous and somewhat felted or matted; they consist mainly, so far as recognizable vegetable matter goes, of roots and rootlets of the plants mentioned and often contain the rootstocks or stems of the plants. They are light yellowish to reddish, rusty brown, or even blackish.

Shrub-and-tree.—The greater part of the peat of this group consists of material derived from roots, tree trunks, branches, twigs, and bark, in a tangled mass and not highly decomposed, mixed with remains of leaves, ferns, rootlets, and often in a matrix of more finely divided material.

No standards have yet been developed for classifying peat into fuel grades. The Federal Trade Commission established rules for the agricultural classification of peat in January 1950, one of which forbids representing a product as moss peat unless at least 75 percent, on a dry basis, is peat derived from mosses. Some municipal and State governments have set up (agricultural) standards for peat they purchase; and all peat procured by the Federal Government must conform to specifications defined in the Federal Stock Catalog (sec. IV, pt. 5, Q–P 166–3, November 1947).

RANKS OF COAL

The rank or degree of coalification of the higher rank coals is determined by "proximate analysis," a chemical analysis in which coal is decomposed into four constituents: (1) Water, called moisture; (2) mineral impurity, called ash, left when the coal is completely burned; (3) volatile matter, consisting of gases driven out when coal is heated to certain temperatures; and (4) fixed carbon, the cokelike residue that burns at higher temperatures after volatile matter has been driven off. For the lower rank coals, heating value and caking and weathering properties determine rank. The rank of coal increases as the amount of fixed carbon increases and the amounts of moisture and volatile matter decrease. Moisture and volatile matter were squeezed and distilled from coal during its formation by pressure and heat, thereby raising the proportion of fixed carbon. That this change is great will be seen from the following analyses, on an ash-free basis, of a typical lignite, the lowest rank of coal, and an anthracite, the highest:

	Lignite	*Anthracite*
Fixed carbon ________________ percent __	33	92
Volatile matter ________________ do ____	26	5
Moisture ______________________ do ____	41	3
Total percent ________________	100	100

Table 1 lists the ranks of coal. Lignite (including brown coal and lignite) ranges in appearance from brown to black. It yields a brown powder when ground or when rubbed over a rough, white surface. American lignite as mined contains 30 to 40 percent moisture. When exposed to air, it soon slacks or falls to pieces because of loss of moisture. Although large deposits of lignite exist in the United States, production is not yet important nationally, partly because it occurs far from major markets. It is relatively difficult to store, and its heating value is low, making unprocessed lignite uneconomical to ship very far.

Subbituminous coal is black; as mined, it looks much like the bituminous coal commonly seen in coal yards. It contains 15 to 30 percent moisture. The range of heating values of subbituminous A coal is the same as that of high-volatile C bituminous coal. Subbituminous coal is entirely noncoking. Like lignite, it weathers upon exposure to air and is subject to spontaneous combustion if not stored properly. Like lignite also, it is used principally in the areas where it is mined.

Bituminous coal is the most abundant and widespread rank of coal in the United States. It is the coal used most commonly for industrial, power, railroad, and heating purposes. Bituminous coals may be coking or noncoking. This property is not based on the rank of the coal but rather on whether it will produce a coke when processed in a coke oven. Nearly all eastern bituminous coals have coking properties, but many western bituminous coals are noncoking or free burning.

A coking or caking coal is one that softens and flows when it becomes almost hot enough to take fire. As it is heated further, some degradation occurs, and volatile matter escapes as a gas. When the coal is heated red-hot in a sealed oven where there is little or no air, a dull, gray, porous mass—coke—remains after most volatile liquids and gases have been driven off. Because coke consists largely of fixed carbon, coking is often called carbonization.

Noncoking bituminous coal may look like coking coal, and its composition may be similar, but it burns freely without pronounced swelling. Instead of a porous mass, it leaves a char or powdery residue. Noncoking and coking bituminous coals can be used interchangeably for many purposes, such as fuel for power plants, railroads, and heating and cooking. For some purposes, however, coking coals are required, the principal one being the manufacture of coke for smelting iron ore in blast furnaces. Noncoking coals are preferred for cement and tile burning, for which a high heating value and a high proportion of volatile matter are desirable.

Most bituminous coals appear black and lus-

TABLE 1.—*Classification of coals by rank* [1]

[F. C.=fixed carbon; V. M.=Volatile matter; B. t. u.=British thermal units]

Class	Group	Limits of fixed carbon or B. t. u., mineral-matter-free basis	Requisite physical properties
I. Anthracite	1. Meta-anthracite	Dry F. C., 98 percent or more (dry V. M., 2 percent or less).	Nonagglomerating.[2]
	2. Anthracite	Dry F. C., 92 percent or more and less than 98 percent (dry V. M., 8 percent or less and more than 2 percent).	
	3. Semianthracite	Dry F. C., 86 percent or more and less than 92 percent (dry V. M., 14 percent or less and more than 8 percent).	
II. Bituminous[3]	1. Low-volatile bituminous coal.	Dry F. C., 78 percent or more and less than 86 percent (dry V. M., 22 percent or less and more than 14 percent).	
	2. Medium-volatile bituminous coal.	Dry F. C., 69 percent or more and less than 78 percent (dry V. M., 31 percent or less and more than 22 percent).	
	3. High-volatile A bituminous coal.	Dry F. C., less than 69 percent (dry V. M., more than 31 percent); and moist B. t. u.,[4] 14,000,[5] or more.	
	4. High-volatile B bituminous coal.	Moist B. t. u.,[4] 13,000 or more and less than 14,000 [5]	
	5. High-volatile C bituminous coal.	Moist B. t. u.,[4] 11,000 or more and less than 13,000.[5]	Either agglomerating or non-weathering.[6]
III. Subbituminous	1. Subbituminous A coal.	do.[4]	Both weathering and nonagglomerating.
	2. Subbituminous B coal.	Moist B. t. u.,[4] 9,500 or more and less than 11,000.[5]	
	3. Subbituminous C coal.	Moist B. t. u.,[4] 8,300 or more and less than 9,500.[5]	
IV. Lignitic	1. Lignite	Moist B. t. u.,[4] less than 8,300 [5]	Consolidated.
	2. Brown coal	do	Unconsolidated.

[1] This classification does not include a few coals that have unusual physical and chemical properties and that come within the limits of fixed carbon or B. t. u. of the high-volatile bituminous and subbituminous ranks. All of these coals either contain less than 48 percent dry, mineral-matter-free fixed carbon or have more than 15,500 moist, mineral-matter-free B. t. u.

[2] If agglomerating, classify in low-volatile group of the bituminous class.

[3] It is recognized that there may be noncaking varieties in each group of the bituminous class.

[4] Moist B. t. u. refers to coal containing its natural bed moisture but not including visible water on the surface of the coal.

[5] Coals having 69 percent or more fixed carbon on the dry, mineral-matter-free basis are classified according to fixed carbon, regardless of B. t. u.

[6] There are three varieties of coal in the high-volatile C bituminous group, namely, 1, agglomerating and nonweathering; 2, agglomerating and weathering; 3, nonagglomerating and nonweathering.

trous in the pile, but close inspection shows a banded structure, with alternate layers of bright, glossy coal and dull, grayish black coal. In some coals this dull material predominates.

Anthracite, sometimes called hard coal, has a brilliant luster and uniform texture. It will not soil objects as does coal of lower ranks. Anthracite has a higher percentage of fixed carbon and a lower percentage of volatile matter than the lower rank coals. It burns slowly with a pale-blue flame, free from smoke. Most anthracite has a somewhat lower heating value than the highest grade bituminous coals, but its lack of soot and the fact that it will burn longer without attention make anthracite an ideal domestic fuel. Most of it is now used for heating and cooking.

CHEMISTRY OF COAL

Coal, like wood and peat, contains carbon, hydrogen, oxygen, nitrogen, sulfur, and other elements in small quantities. The proportions in which the major elements are present differ greatly in different kinds of coal, and the chemistry of coal is quite complicated. For practical purposes, however, coal is analyzed by proximate analysis or sometimes by a more complete test—ultimate analysis—to determine the relative amounts of chemical elements.

Moisture, sulfur, and ash are the undesirable ingredients of coal. Volatile matter and fixed carbon are the most important contributors to the energy produced when coal is burned; and producing energy in the form of heat is the purpose of most uses of coal.

When a scientist wishes to find the heating value of a sample of coal, he weighs an exact quantity of it. He burns this completely in a bomb filled with compressed oxygen and immersed in a carefully measured quantity of water. He takes the temperature of the water at the beginning of the test and again after the burning coal has warmed it. He can then calculate the heating value of the coal, which is commonly expressed in British thermal units a pound. One British thermal unit (abbreviated B. t. u.) is the amount of heat needed to raise the temperature of 1 pound of water 1° F.

FUEL RESERVES

The United States is fortunate in being richly endowed with mineral-fuel resources. It has one-third of the geologically estimated coal and lignite reserves of the world, one-fourth of the proved petroleum reserves, and perhaps one-third of the world's proved natural-gas reserves. The United States also is the greatest producer of solid, liquid, and gaseous mineral fuels. In 1953 this country produced 23 percent of the world output of coal, 50 percent of all petroleum, and probably 90 percent of the natural gas.

To satisfy a large and still increasing demand for liquid fuel, more petroleum deposits must be discovered, more petroleum imported, or, from the standpoint of self-sufficiency, synthetic liquid fuels made from natural gas, coal, or oil shale. Each of these procedures probably will be employed to some degree, depending on economic and national defense conditions.

Additional domestic reserves of petroleum are being found at a rate almost adequate to supply the increasing domestic demand, although the cost of delivery is increasing and importation of oil had grown to 13 percent of consumption by 1953. Development of oil deposits in South America, Canada, and the Middle East are making more oil available for importation to the United States under peacetime conditions and for supplying the growing demand of other countries. The first industrial plant in the United States is beginning to manufacture chemicals and oils from natural gas at Brownsville, Tex. Bureau of Mines research at Rifle, Colo., has demonstrated that liquid fuel can be made from oil shale at comparatively small increases over prices prevailing in recent years for gasoline from petroleum. The United States has large reserves of coal and lignite that can be converted to liquid fuel as prices of petroleum rise still further.

COMPARISON OF RESERVES OF MAJOR FUEL SOURCES

Ultimately, coal must be the primary source, not only of solid fuels, but of liquid and gaseous fuels as well, as will be seen from a comparison of coal reserves with those of petroleum, oil shale, and natural gas.

The Federal Geological Survey estimated the remaining coal reserves of the United States, not including Alaska, at 1,900 billion short tons as of January 1, 1953; these estimates include beds down to 14 inches thick for anthracite and bituminous coal and to 2.5 feet thick for subbituminous coal and lignite, and at depths not exceeding 3,000 feet. The estimates are largely geologically inferred and make no allowance for mining losses; moreover, they do not consider that beds less than 2 feet thick are not economically minable by present underground methods. Investigations by the Bureau of Mines and others have shown that, on the average, about 50 percent of the coal is recovered in underground mining. Losses include coal left in pillars in areas around oil and gas wells, under towns, railroads, roads, and streams, in riverbeds, in thin and impure beds, and in isolated areas not readily accessible for mining. For these reasons recoverable coal is considered to be half of the total estimated reserve.

As of December 31, 1953, committees of the American Petroleum Institute and of the American Gas Association estimated the proved reserves of crude oil at 28.94 billion barrels and natural-gas liquids at 5.44 billion barrels, making a total of 34.38 billion barrels of liquid hydrocarbons. The American Gas Association committee estimated the proved reserves of natural gas at 211 trillion cu. ft.; the Geological Survey estimated the recoverable oil from shales, averaging not less than 30 gallons oil content per ton, at 25 billion barrels, and for shale deposits having an average content down to 15 gallons of oil per ton at 500 billion barrels.

These figures cannot be directly compared, because they are not calculated on the same basis. Only proved reserves are given for liquid and gaseous fuels. These are increased each year by new discoveries and extensions of fields and decreased by the year's production. More oil and gas reserves will be proved in the United States; in this respect, these fuels differ from coal and oil shale.

The solid-fuel reserves are based largely on geological observations after limited exploratory drilling, especially in the Western States. No great accuracy is claimed for these estimates, but future exploration is not likely to add much to the solid-fuel reserves. Revisions will probably be downward as more data are obtained in the various coal fields.

Therefore, to compare reserves of fuels, one should express the figures not only in terms of energy content—British thermal units (B. t. u.)—but also indicate whether they refer to proved, measured, or indicated amounts of material recoverable by current methods. Proved reserves of oil and gas meet the last qualification. On the other hand, estimates of recoverable coal are long-term estimates, including considerable inferred and not proved reserves, as well as beds only 14 to 28 inches thick and those 2,000 to 3,000 feet below the surface and thus too expensive to mine at present. The Federal Geological Survey studied its recent reestimation of reserves in 10 States and concluded that one-fourth of the minable reserves would fairly represent the measured and indicated reserves of coal commercially recoverable by current production methods. This amount—237 billion tons—is ⅛ of the total reserves (1,900 billion tons).

The Geological Survey also has cited estimates of authorities on the total inferred reserves of petroleum and natural gas for present and future use and compared these with the minable reserves of coal (including measured, indicated, and inferred) without reference to cost of mining. These two types of estimates are given in table 2 in terms of British thermal units.

Column 1 of this table, showing 5,669 quadrillion B. t. u. for all mineral fuels, gives the measured and indicated reserves of coal now commercially minable, proved reserves of petroleum and natural gas, and estimated reserves of oil from measured and indicated oil-shale deposits averaging not less than 30 gallons of oil per ton.

TABLE 2.—*Estimated recoverable mineral-fuel reserves of the United States, as of Jan. 1, 1953*

Kind of fuel	Reserves (quadrillion B. t. u.)		Percent of total fuel reserves	
	Proved and commercially recoverable [1]	Total potential reserve	Proved and commercially recoverable	Total potential reserve
Coal	5,117	20,469	90.3	84.0
Petroleum, including natural-gas liquids	198	413	3.5	1.7
Natural gas	200	487	3.5	2.0
Bitumen from bituminous sandstone	4	8	------	------
Oil from oil shale	150	3,000	2.7	12.3
Total	5,669	24,377	100.0	100.0

[1] Measured and indicated coal reserves are considered proved.

Column 2, showing 24,377 quadrillion B. t. u., gives the minable (½ of the total), measured, indicated, and inferred reserves of coal without any deduction for thin or deep beds, the unproved reserves of oil and gas estimated on the basis of favorable geological formations, and the estimated reserves of oil from indicated, measured, and inferred oil-shale deposits averaging not less than 15 gallons of oil per ton.

The long-term estimate of coal reserves (including inferred and difficultly accessible coal) is four times the short-term estimate, the long-term oil and gas estimates are about twice the short-term estimate, and the long-term estimate of oil from oil shale is 20 times the short-term estimate, principally because leaner and much more abundant shales are included. Obviously, coal is the dominant fuel in both estimates. It contributes 90 percent of the total energy in the short-term and 84 percent in the long-term estimate. If coal had been used to satisfy the demand for liquid and gaseous fuels in 1953, over 4 times as much coal would have been used as the 488 million tons produced that year. This example illustrates the tremendous demands that will be made on the reserves of coal, oil shale, fissionable minerals, and other sources of energy as gas and petroleum deposits become depleted. The life of these fuel reserves cannot be estimated because of the many variable factors involved. No one can accurately predict demands, new discoveries, and the effect of further technological developments. Such "guesstimates" as have been made by competent scientists who have studied the question have ranged from 75 to 300 years for fossil-fuel reserves.

RESERVES OF PEAT AND COAL

PEAT

The United States contains extensive reserves of peat in at least 26 States. Table 3 shows the approximate distribution of known reserves by regions and States. Minnesota has the largest reserves, followed by Wisconsin, Florida, and Michigan, in that order. The northern region, embracing New England, States bordering the Great Lakes, and Iowa, contains 80 percent of the total deposits of the country. Three States in this region—Minnesota, Wisconsin, and Michigan—contain 75 percent of the estimated deposits. The Atlantic coastal region holds 19 percent, and the Pacific and Gulf Coast States (exclusive of Florida) have less than 1 percent of the total.

The Minnesota and Wisconsin deposits have largely resulted from the decomposition of mosses, grasses, sedges, heath shrubs, and trees, and the texture varies from fibrous in the top layer to plastic in the bottom. The peats of Iowa, Illinois, Indiana, Ohio, Pennsylvania, and New Jersey are of similar texture, although usually more fibrous. Sphagnum peat abounds in the deposits of northern Minnesota, Wisconsin, and Maine, and some is also found in other New England States. Sedges dominate plant life in the salt marshes of the New England coast. Peat of the Atlantic coast region south of New Jersey contains little sphagnum, as both coniferous and deciduous trees contributed a large proportion of the vegetable matter of this peat.

Peat in the Florida Everglades is composed largely of the remains of saw grass and sedge.

TABLE 3.—*Known original reserves of peat in the United States, estimated on an air-dried basis, by region and State* [1]

[Thousand short tons]

Region and State	*Reserves*
Northern region:	
Minnesota	6, 835, 000
Wisconsin	2, 500, 000
Michigan	1, 000, 000
Iowa	22, 000
Illinois	10, 000
Indiana	13, 000
Ohio	50, 000
Pennsylvania	1, 000
New York	480, 000
New Jersey	15, 000
New Hampshire	1, 000
Vermont	8, 000
Massachusetts	12, 000
Connecticut	2, 000
Rhode Island	1, 000
Total	11, 050, 000
Atlantic coastal region:	
Virginia and North Carolina	700, 000
Florida	2, 000, 000
Other States [2]	2, 000
Total	2, 702, 000
Other regions:	
Gulf Coast [3]	2, 000
California	72, 000
Oregon and Washington	1, 000
Total	75, 000
Total, all regions	13, 827, 000

[1] Geological Survey, Coal Resources of the United States, a Progress Report: Circ. 293, Oct. 1, 1953, p. 38.

[2] Includes Delaware, Maryland, South Carolina, and Georgia.

[3] Exclusive of Florida.

COAL

Table 4 gives the recoverable reserves of coal as of January 1, 1953. They total 950 billion tons and include bituminous 525, lignitic 232, subbituminous 186, and anthracitic coal 7 billion tons; however, the average heating values of the various ranks of coal decrease from 14,000 B. t. u. for low-volatile bituminous coal to 6,700 B. t. u. for lignite. Therefore, these coals have also been compared in terms of energy obtainable from them by converting the tonnages in column 1 to equivalent tons of bituminous coal of 13,000 B. t. u. per pound. Thus, a reserve of 232 billion tons of lignite equals 119 billion tons of 13,000 B. t. u. bituminous coal in heating value.

Tables 5 and 6 show the distribution, by rank and geographical location, of coal reserves on the basis of equivalent heating value. According to this method of evaluation, about 54 percent of the minable reserves of coal and lignite occur in the Eastern and Interior provinces and 46 percent in the Northern Great Plains, Rocky Mountain, Gulf, and Pacific provinces.

A great difference in rank exists between eastern and western coals. Virtually all of the Eastern and Interior province coals except anthracite are coking bituminous coals. Coals of the Appalachian region are medium to strongly coking and in 1947 supplied 95 percent of the coal used for making coke. On the other hand, 70 percent of the coals of the Northern Great Plains, Rocky Mountain, Gulf, and Pacific Coast Provinces are subbituminous and lignitic. These coals do not coke on heating.

TABLE 4.—*Potential recoverable reserves of various ranks of coal as of Jan. 1, 1953*

Rank of coal	Billion net tons	Equivalent billion net tons bituminous coal of 13,000 B. t. u. per pound calorific value	Percent of total coal reserves on B. t. u. basis	Heating values used in conversion, B. t. u.
	(1)	(2)	(3)	(4)
Anthracitic	7	7	1	12, 700
Bituminous	525	545	64	13, 500
Subbituminous	186	136	15	9, 500
Lignitic	232	119	20	6, 700
Total	950	807	100	------------

TABLE 5.—*Distribution of recoverable coal reserves in terms of B. t. u. value, by ranks and provinces (weight basis)*

Rank of coal	Billion net tons of equivalent 13,000-B. t. u. coal				
	Anthracitic	Bituminous	Subbituminous	Lignitic	Total
Province					
Eastern	6.3	223	--------	--------	229
Interior	.1	206	--------	--------	206
Northern Great Plains, Rocky Mountain, Gulf, and Pacific	.4	116	136	119	327
Total	7	545	136	119	807

TABLE 6.—*Distribution of recoverable coal reserves in terms of B. t. u. value, by ranks and provinces (percentage basis)*

Rank of coal	Anthracitic	Bituminous	Subbituminous	Lignitic	Total
Province					
Eastern	1	27	--------	--------	28
Interior	--------	26	--------	--------	26
Northern Great Plains, Rocky Mountain, Gulf, and Pacific	--------	14	17	15	46
Total	1	67	17	15	100

TECHNOLOGY OF LIGNITE AND PEAT INDUSTRIES

LIGNITE

MINING

Lignite mining follows, in general, the same principles and practices as bituminous-coal mining. At present, almost all lignite in the United States is produced at strip mines because plentiful deposits are available with relatively shallow cover. The operations (currently carried on in North Dakota, Texas, and California), are (1) removal of overburden by means of power shovels, draglines, or bulldozers; (2) excavation of lignite, largely by means of power shovels; and (3) loading and transportation to railroads or nearby consumers for sale as run-of-mine lignite or to tipples equipped with crushing and screening devices for sale as lignite that is graded according to size, ash content, and heating value.

The tonnage of lignite mined underground in the Dakota area is small. Most Texas production is now made by stripping and has recently expanded rapidly so that it nearly equals that of the Dakotas, which is about 3 million tons per year.

PREPARATION

Lignite currently mined in North Dakota and Texas has low sulfur and ash contents, hence its preparation entails little or no cleaning. When and if lignites containing objectionable amounts of impurities are brought into production, cleaning processes are expected to be similar to those used for anthracite and bituminous coal; however, its high moisture content (30 to 40 percent in American lignites) makes lignite less heavy per unit volume and consequently easier to separate from pyrite, marcasite, etc. (The approximate specific gravity of the different ranks of coal is: Anthracite, 1.62; bituminous coal, 1.37; and lignite, 1.25 to 1.32.)

Present methods of preparation include crushing and screening, drying, carbonizing, and pelleting or briquetting for ultimate use. Although lignite usually requires less preparation than higher rank coals, it presents a storage problem because of oxidation, spontaneous heating, and eventual ignition, particularly in piles of unsegregated sizes of lignite. This danger can be averted by crushing and compacting in place in a pile having small angles at the edges (about 17° to 20° from the horizontal).

UTILIZATION

Like other ranks of coal, lignite is suffering losses in domestic and industrial heating, where it has been replaced to some extent by more convenient and plentiful supplies of oil and gas. On the other hand, the industrial use of lignite is in its infancy. It is a cheap, plentiful fuel (either as mined or briquetted) for heating and power generation, a source of industrial carbon for decolorizing and purifying solutions (sugar refining), an absorbent of liquids from gases (gasoline from natural gas), and a source of industrial gases. Some lignite deposits yield montan wax. In the United States only small reserves in Arkansas and California yield enough such wax for commercial workability. In recent times the Ione, Calif., deposit has been worked for this purpose on a limited scale. Wax is extracted with solvents and used in polishes, rubber, insulation, inks, greases, coatings, adhesives, explosives, textiles, carbon paper, and hardened wood.

Lignite, like all low-rank coals, gasifies relatively easily and is thus a potential source of synthesis or water gas, containing hydrogen, carbon monoxide, and nitrogen and used for the production of ammonia (fertilizer), methanol and other alcohols, solvents, synthetic liquid fuels, and hydrogen. This gas is necessary for hydrogenation of coal to chemicals and liquid fuels, and of oils for margarine and cooking fat.

Unlike the bituminous coals of the eastern United States, lignite does not produce coke. Carbonization yields a char that is relatively soft and is currently used as briquetted fuel in North Dakota and as activated carbon in Texas. The char is a rapid-burning fuel.

Lignite may be dried (from 35 to 40 percent to about 15 percent water) by high-pressure steam to yield lump fuel with a greatly decreased tendency to slack or fall apart with further loss of moisture. There are now no steam-drying operations in the United States.

Industrial exploitation of lignite in the United States—most deposits are in the Dakotas and Montana—has been limited to date, except locally as a fuel, because of two principal factors, the abundance of higher rank coals close to points of usage and a lack of industry and markets in the area of deposits. However, active development is in progress in Texas.

The outstanding potential use of lignite is as upgraded solid and liquid fuels and tar. The fuel produces electric power for making aluminum in Texas, even though that State is an oil- and natural-gas-producing center. In the first phase of processing, lignite is crushed to ¼-inch pieces or smaller and fed to a drying tower where a stream of hot gas (derived from the process itself) carries it upward and causes the moisture to vaporize and disrupt the pieces of lignite in about 40 seconds. The moisture content is thus reduced from 36 to 5 percent or less, and the heat value of the remaining, dry lignite is raised from about 6,800 B. t. u. to 10,800 per pound. Dust is recovered mechanically and can be burned to generate electricity. In a second phase dried, finely divided lignite is fed to a carbonizing tower, maintained at about 900° to 950° F., where it yields oil, tar, and char. The char is separated by means of a cyclone and burned for power. The oil and tar appear, at this stage of development, to have considerable potential value, which, when applied against the total cost of generating power, may make the latter very low. The key to commercial success of this process is development of uses and consequently markets for low-temperature tar and oils.

PEAT

TECHNOLOGY

The greatest progress in technology has been made in those countries where peat necessarily is used for fuel purposes because of lack or high cost of other fuels. Carbonization of peat for making char or charcoal, tar, and liquid byproducts has been studied in many countries, particularly Sweden, Germany, and Canada. In England peat extracts have been found to contain considerable quantities of wax of high melting point—suitable substitutes for low-grade montan wax obtained from German brown coal. Russian reports indicate the possibility of fermenting peat in solution or suspension to alcohol. The Russians are also reported to have developed very efficient combustion equipment for generating electric power from peat. Several large power stations are operating on peat in Ireland and Sweden. In the United States the Minnesota Iron Range Resources and Rehabilitation Commission has worked on the problem of using peat as a fuel for beneficiating low-grade iron ore.

USES

The principal use of peat in the United States is for soil improvement. Although some varieties contain a relatively high percentage of nitrogen (3 percent or more), peat is not a good fertilizer because the nitrogen is not readily given up as plant food like the "soluble nitrogen" of artificial fertilizer. Peat contains a large percentage of organic matter (humus), which is an excellent soil conditioner when used in large enough quantities. The second largest use for peat is as a conditioner and filler in mixed fertilizers, for which well-decomposed peat, suitably dried and ground, is ordinarily employed. In mixed fertilizers, peat acts as a carrier for the primary nutrients—nitrogen, phosphoric oxide, and potash—not as an agent for supplying plant food. Fibrous peat is also used as mull or litter material for bedding stock and for stable and poultry yards. Because of its moisture-absorbing qualities, it is used generally for stock bedding. Its deodorizing capacity makes peat useful in barnyards and stables, where it delays decomposition of the nitrogenous and other organic substances deposited on it. Nurserymen, gardeners, and others use peat as a packing material for plants, shrubbery, fruit, vegetables, eggs, and fragile materials. Other uses for peat include filtering and deodorizing agents, dyestuffs, and tanning substances.

STATISTICAL DATA

PRODUCTION, SIZE, AND GEOGRAPHICAL DISTRIBUTION OF LIGNITE INDUSTRY

The production of lignite increased gradually to 3,370,000 tons in 1950 and then declined to 2,851,000 tons in 1953.

North Dakota has become the predominant lignite-producing State, having furnished 98 percent of the output in 1953. Before 1921 Texas was the principal lignite producer, with over 1 million tons annually as late as 1928, from which it declined to 18,169 tons in 1950. (Bureau of Mines statistics on bituminous coal and lignite do not include mines producing less than 1,000 tons per year.) However, Texas production should increase materially in the future to supply the new lignite-powered electric generation plant of the Texas Power & Light Co. at Rockdale.

Contrary to the generally downward trends in bituminous-coal production during the 1920's, the 1930's, and following the postwar peak of 1947, the output of lignite in North Dakota has been unique in steadily increasing over the years. Of the various factors responsible for this trend, the most important one has probably been heavily increased production by strip mining and continued improvements in methods of utilization. Although total production in North Dakota has more than doubled between 1926 and 1953 (1,370,000 to 2,802,000 tons) the number of employees has been cut in half, and the number of mines has been reduced by two-thirds. On the other hand, production per man-day has increased from 6.56 tons in 1926 to 25.70 in 1953. The more rapid acceleration that began in 1940, when output per man-day was still as low as 8.57 tons, has continued through 1953. The average value of North Dakota lignite, f. o. b. mines, has more than doubled between 1940 and 1953 (from $1.17 to $2.36 per ton).

The tremendous swing to strip mining and the differences in average value per ton at the mine and in productivity per man-day as between underground and strip mining are indicated by the following data for 1953:

	Montana	North Dakota	South Dakota	Total
Underground mines:				
Production ....net tons..	22,148	161,438	--------	183,586
Number of mines	4	6	--------	10
Average number of men working daily	16	95	--------	111
Average production per man per day ....tons..	7.67	8.13	--------	8.07
Average value per ton	$3.80	$2.40	--------	$2.57
Strip mines:				
Production ....net tons..	2,655	2,641,120	23,671	2,667,446
Number of mines	2	40	2	44
Average number of men working daily	5	486	14	505
Average production per man per day ....tons..	6.88	29.61	8.61	28.89
Average value per ton	$3.53	$2.36	$3.47	$2.37
Total all mines:				
Production ....net tons..	24,803	2,802,558	23,671	2,851,032
Number of mines	6	46	2	54
Average number of men working daily	21	581	14	616
Average production per man per day ....tons..	7.58	25.70	8.61	24.78
Average value per ton	$3.77	$2.36	$3.47	$2.38

Of particular interest are production data from Mercer, Ward, Burke, and Divide Counties, N. Dak., where output per man-day has reached averages of approximately 23, 27, 33, and 36 tons, respectively, and value f. o. b. mines in 1953 ranged from $2.29 to $2.59 per ton. Also, information from Bowman County, where output per man-day was over 43 tons in 1953 and value per ton f. o. b. mines was $1.73, is worthy of note.

The principal competitive source of energy in North Dakota is oil. No hydroelectric power is available nor is the cost of such power likely to be competitive with lignite-generated power for large-scale use. Although natural gas has not been a strong factor until recently, its potentials are considerable unless checked by increasing price trends.

The principal factors that have retarded the use of lignite are low B. t. u. values, problems of degradation and storage, and transportation costs. The combination of high freight rates (which provide a limitation to the movement eastward of North Dakota lignite) and a quality lower than that of bituminous coal moving into the upper Midwest markets from Central and Eastern coal fields has confined the general market for North Dakota lignite to a rather restricted area. Advancements in technology with respect to upgrading, increased efficiency (at the consumer level), and development of alternatives to expensive transportation will be very important to the future of the lignite industry.

PRODUCTION, SIZE, AND GEOGRAPHICAL DISTRIBUTION OF PEAT INDUSTRY

Production of peat in the United States averaged about 200,000 short tons annually in 1951–53. Compared with the average production in 1947–49, this figure represents a 54-percent gain. Production increased fastest on the west coast, particularly in Washington, where output advanced 20-fold in the past 5 years. Over half of production in 1953 consisted of peat humus, 37 percent of reed or sedge peat, and the remainder of moss peat.

Peat has been used as a fuel for centuries, notably in the Soviet Union, Germany, Sweden, Ireland, Denmark, and Holland. In the United States, however, its use as fuel has been negligible because of ample supplies of superior fuels at reasonable costs. In this country peat is used almost exclusively for agricultural and horticultural purposes.

In 1953 commercial production of peat was reported to the Bureau of Mines by 68 producers in 18 States; however, the output from these operations does not represent the total utilization of peat in the United States. Some municipalities operate peat plants for their own needs (to improve the soils of lawns and parks), and many owners of land containing peat use it for growing crops. The area of peat and muck soils under cultivation is not known but probably amounts to thousands of acres. Such use of peat is an important factor in its economic utilization.

The value of domestically produced peat, f. o. b. plants, totaled $1,617,947 in 1953. The value of an almost equal quantity of imported material was $7,333,749. Data on employment, wages and salaries, value of buildings, machinery and equipment, and other economic factors relating to the peat industry have not been published since the 1939 Census of Mineral Industries. Because of the growth of the peat industry since that time, this information is outdated; however, the results of a Census of Mineral Industries covering 1954 will be available in the near future.

The peat industry is widely scattered over the United States. Washington led all States in peat production and in the number of active operations in 1953. Its producers supplied about one-sixth of the national output. Ohio, Florida, Michigan, and New Jersey produced half of the total domestic production. Other States where significant tonnages of peat were produced are Iowa, California, Pennsylvania, Connecticut, Indiana, and Colorado.

CONSUMPTION AND PRICES

LIGNITE

The decline in lignite production and consumption in Texas was due, of course, to the huge indigenous supplies of oil and natural gas; however, now that the large plant of the Texas Power & Light Co. at Rockdale is expected to use dried lignite for power generation, consumption of Texas lignite should increase substantially.

Consumption has increased in the North Central States because of reduced cost of production (heavy increase in strip mining) combined with adaption of lignite to use in railroads, steady growth of demand for electric power in the region, and lesser availability of alternate sources of energy than in other geographic areas.

Consumption of lignite for generating electric energy amounted to 1,509,282 tons in 1953, according to the Federal Power Commission. This use took 53 percent of total production. Consumption by States was: North Dakota, 1,097,777 tons; Minnesota, 237,105 tons; and South Dakota, 174,490 tons.

PEAT

In 1953 the consumption of peat exceeded 400,000 short tons for the first time. This tonnage was nearly five times greater than in prewar years. The latest information available indicates that approximately 59 percent of the total quantity consumed in the United States was used for soil improvement, 24 percent for a conditioner and filler in mixed fertilizers, 15 percent for poultry litter and stable bedding, and 2 percent for all other purposes, such as packing material for plants, shrubbery, fruit, vegetables, and eggs.

In 1953 the average value of moss peat was $11.87 per ton, reed or sedge type $8.82, and humus $6.65. The average price per ton of all types increased slightly to $8.34. Peat sold for soil improvement is cheapest on the average, while peat for "other uses" is highest. In the latter group some peat that is marketed in small packages for household plants averages as much as $50 per ton.

FOREIGN TRADE

Imports of peat in 1953 reached an alltime high and approximated 200,000 short tons, a quantity nearly equal to domestic production. The total dollar value, however, was over four times that of the domestic product. All imported material was moss peat and, according to classifications set up by the United States Department of Commerce, was divided into two grades: (1) Poultry and stable and (2) fertilizer. The quantity of poultry and stable grade decreased slightly in 1953, whereas that of fertilizer grade increased by 35,777 tons (25 percent). The latter grade is usually of a higher quality, and the average unit value is correspondingly higher. The principal sources of imports are Germany (58 percent), Canada (more than 33 percent), Netherlands, and Ireland. Exports of American peat are virtually nil.

OUTLOOK

LIGNITE

The future of lignite will depend considerably on the extent to which oil and natural gas become available in North Dakota and adjacent areas. Relative price levels of the respective fuels, including bituminous coal, will be the principal determinant in the industrial market and to some extent in domestic consumption, although factors of convenience have a strong effect upon domestic consumption.

Whereas domestic consumption of North Dakota lignite stayed ahead of industrial use until 1946, industrial use since then has increased steadily. With higher production of thermal power and steady rise in the availability of competitive fuels, industrial markets hold the key to any appreciable advancement in the production and use of lignite.

In addition to being limited in growth by high transportation costs, low B. t. u. values, degradation in storage and transit, and competition of oil and natural gas, the lignite industry in the North Central States is restricted by the fact that the region is primarily agricultural. The present consuming area for lignite extends about 300 miles east and southeast from the lignite deposits. The eastern market extends to the area where the cost of lignite is equaled by that of eastern coals delivered at the northern Lake ports; and the southeastern market extends to where costs are equal to that of coal barged up the Mississippi River, principally from Illinois and Kentucky. North and west of the lignite deposits, availability of higher rank coals and lack of population have prevented development of markets for lignite.

Two important areas that may be within economic reach of the North Central lignite deposits after development of cheaper transportation methods are Minneapolis-St. Paul and the northern Minnesota iron-range country. In the iron ranges large amounts of electric power will be required in the near future for processing low-grade taconite ore that may soon undergo increased utilization. The present delivered cost of fuels now used in these two areas is slightly over twice the cost of lignite at the mine.

Pipeline transportation of coal and long-distance transmission of power generated at the mine are future possibilities for increased lignite production.

PEAT

Although peat will not be able to compete in the near future with oil, natural gas, or coal for fuel purposes or as a chemical raw material, the use of peat for agricultural and horticultural purposes should increase. The demand for stock-bedding or litter purposes doubtless will also increase when the superior absorbing qualities of peat become known more generally.

PROBLEMS

Certain lignite deposits in Texas contain partings that will make some type of cleaning necessary, unless selective mining methods are used. Research on this problem is needed. Transportation and storage of lignite, particularly dried lignite or char, pose real problems owing to their tendency to heat because of their high reactivity. As yet these problems have been only partly solved.

Because of its high moisture content, lignite can be very substantially upgraded in value by drying. However, drying at atmospheric pressure with hot gas or superheated steam has to date given severe size degradation. This is of little consequence for some applications but is a problem if sized domestic or stoker fuel is desired or certain types of gasification processes are contemplated. Drying with high-pressure steam or briquetting of dried lignite are partial solutions to the size-degradation problem, but available processes are relatively expensive.

Lignite can be carbonized, yielding a reactive char and potentially valuable liquid and gaseous product; however, the yields of these products vary markedly with the different lignites. Much research along these lines is required. The economics of carbonization will depend upon the demand and price obtainable for the liquids produced. Research is required on the characteristics of these liquids.

Gasification of lignite presents the possibility of an outlet for large tonnages of lignite. Research on methods of gasification specifically suited for this fuel is required.

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LIME

(Including Calcium)

By

Oliver Bowles [1] and Joseph C. Arundale [2]

LIME PRODUCTION is an ancient industry; kilns for burning limestone into lime were mentioned as early as 184 B. C. Its early uses were confined almost exclusively to plasters and mortars, but in modern times its chemical and industrial uses have overshadowed the others. It is essential to many processes.

Summary

Lime consists of calcium oxide (CaO) or the combined oxides of calcium and magnesium ($CaO{\cdot}MgO$). It is obtained by calcining a limestone or dolomite and is used extensively as an alkali chemical and in other industrial applications. Smaller quantities are employed in mortar and plaster and as a soil conditioner. Lime is essential for many of its uses and therefore meets little competition, except in the building trades. It has many chemical and industrial uses that affect the defense program. Lime is one of the raw materials utilized in manufacturing calcium and magnesium metals.

The lime industry needs a variety of technical, economic, and statistical information. The principal problems pertain to improved processes of mining and quarrying and to more efficient calcination and hydration. Kilns are undergoing changes in design and operation as a result of research. Utilization of waste is a problem facing many lime producers. Other problems involve the technology and uses of calcium and magnesium metals.

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[1] Commodity specialist, Bureau of Mines.

[2] Assistant chief, Branch of Construction and Chemical Materials, Bureau of Mines.

BACKGROUND

Lime, often confused with limestone, is calcium oxide (CaO) and does not occur as such in nature but is derived from limestone ($CaCO_3$) by a process of calcination at a moderately high temperature which drives off the carbon dioxide (CO_2). Dolomite, the double carbonate of lime and magnesia ($CaCO_3 \cdot MgCO_3$), is also used for making lime, which consists of the combined oxides CaO·MgO. Such lime is known as dolomitic or high-magnesian. For many uses the high-calcium and dolomitic limes are interchangeable, and unless otherwise specified the term "lime" as used herein refers to both varieties. At some plants the dolomite is dead-burned, that is, calcined at a high temperature with an iron additive to give a product having a high melting point that makes it useful for refractory-furnace lining. Dead-burned dolomite is included in statistics as a part of lime production.

Lime has a great variety of uses, particularly in the chemical industries. In 1952 open-market lime was produced by 160 companies in 33 States and 2 Territories. Total production exceeded 8 million tons valued at over 95 million dollars at the point of manufacture. In addition, substantial quantities of so-called "captive-tonnage lime" were produced by companies for their own use.

Ohio, Pennsylvania, and Missouri are the principal producing States and together furnish more than half of the United States total. Other large producers are Virginia, Alabama, Illinois, and Texas.

Limestone and dolomite occur widely in many States. Of the total domestic limestone produced, about 6 percent (nearly 15 million tons in 1950) is used for making open-market lime. The balance is used chiefly as crushed and broken stone, as building stone, or for cement manufacture.

MINING METHODS

Lime companies produce their own stone by operating either open-pit or underground mines. The problems of mining and quarrying are, in general, similar to those in the crushed-stone industry as discussed in the Stone chapter. Special features of quarrying or mining stone for lime manufacture are elimination of impurities, careful sizing, and selective mining to produce a lime of constant composition, or a series of special limes. Methods of quarrying and the problems connected therewith have been described in Bureau of Mines Bulletin 269, Quarry Problems in the Lime Industry (*5*).[2] Essentially the mining process consists of stripping the overburden from an area of suitable limestone, blasting the rock with heavy charges of explosives in drillholes, and loading the broken stone into trucks or cars. When hand loading is employed, the lump rock is ready for direct delivery to shaft kilns. At many plants crushers are used to break the rock and screens to size it. For shaft kilns stone of 4-inch to "1-man" size is preferred. For rotary kilns finer sizes are used.

CALCINING

The next step after quarrying and preparing the stone is to calcine it into lime; this is done principally in shaft or rotary kilns, although some other types of calciners are also used. The kiln product is quicklime (CaO), but many plants have hydrators in which the quicklime is converted into hydrated lime ($Ca(OH)_2$). About one-fourth of the lime sold is in hydrated form. Modern methods of mining, calcining, and hydration are described in some detail in an information circular of the Bureau of Mines, The Lime Industry, published in 1952 (*3*) and in the chapter on Lime in AIME Industrial Minerals and Rocks (*15*).

CALCIUM-METAL MANUFACTURE

The manufacture of calcium metal is more complex. In one process the metal is recovered by thermal reduction of lime with aluminum in a vacuum. Calcium is produced also by the electrolysis of fused calcium chloride. It also can be recovered as a byproduct of sodium-metal production. In this process calcium chloride is mixed with sodium chloride to reduce the melting point of the electrolyte (*11*).

COPRODUCTS

It is not uncommon for lime companies to produce other stone products. For instance, part of their activities may be devoted to production of agricultural limestone or crushed stone for highways and concrete aggregate.

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter; pages given apply to the reference cited.

SUBSTITUTE MATERIALS

For many uses lime has no substitutes. In the chemical industries it is known as "the king of all the bases." For important reactions like those involved in water softening and sugar, paper, and alkali manufacture it is essential. For liming land, however, uncalcined ground limestone may be substituted for lime, although the reactions in the soil are slower. In the building trades, gypsum plaster and portland cement may be substituted for lime to some extent; however, lime is an essential constituent of various widely used mortars and plasters. Calcium metal for some purposes is vulnerable to substitution by sodium, magnesium, and aluminum. The relatively high price of calcium encourages such substitution. The price could be reduced substantially if large-scale production were maintained, but to accomplish this the market must be expanded.

REUSE

In certain applications lime, after performing some useful function, is reconverted into its original carbonate form; in some industries—notably paper and beet-sugar manufacture and in water purification—the lime carbonate thus formed may be recalcined to lime and used again. Such reused lime is a very small fraction of total consumption.

RAW-MATERIAL RESERVES

Limestones are abundant and widely distributed in the United States. However, only limestones of high purity are usable for lime manufacture. Rockwood (*15, p. 473*) presents a table of analyses of the limestones that are used at many lime plants in the United States, and they show a consistent content of 97 to over 99 percent total carbonates. Limestones of such high quality are much more restricted in occurrence than limestones in general. Surface deposits of high-grade limestones are more or less depleted in many localities; and, although the resources are ample for continued and enlarged operations for many years, a substantial number of lime producers are obliged to resort to underground operations. The supply of the raw material (lime) for manufacturing calcium metal is sufficient for all anticipated needs, and calcium can be produced in existing ferrosilicon-process magnesium plants.

USES

Lime has a multitude of uses, which fall into three main groups. The largest group comprises the chemical and industrial applications, such as metallurgy, water purification, and the manufacture of alkali, bleach, calcium carbide, glass, paper, sugar, and various other products. The distribution of lime according to use for 2 recent years is given in table 1. The second group comprises building-material applications such as mason's lime, finishing lime, and masonry mortar. The third use is soil improvement. The quantities of lime applied to all major uses are given each year in the Lime chapter of the Minerals Yearbook (*7*). The specifications for the various uses are given in some detail in a recent report of the Bureau (*3*) and also in a recent publication of the National Lime Association (*14*).

The manufacture of metallic calcium is one of the newer uses of lime. Calcium generally is referred to as one of the alkali-earth metals. It is a ductile and malleable metal which easily can be cut, pressed, extruded, or drawn into various shapes. Freshly cut surfaces are attacked by moist air, forming a bluish gray oxide. Its specific gravity is 1.55; it is a poor conductor of electricity and has a tensile strength of 8,700 p. s. i. and a hardness slightly under that of aluminum. Calcium metal and its alloys have many important applications. The metal is used in reducing organic compounds, desulfurizing petroleum, and debismuthizing lead and in the production of hard lead alloys. Calcium-silicon alloys are used as reducing agents in ferrous and nonferrous metallurgy. Calcium hydride is an efficient hydrogen carrier.

TABLE 1.—*Lime (quick and hydrated) sold by producers in the United States, 1951–52, by uses*

Use	1951			1952		
	Short tons	Value		Short tons	Value	
		Total	Average		Total	Average
Agricultural	343, 619	$3, 712, 608	$10. 80	392, 383	$3, 816, 603	$9. 73
Building:						
Finishing lime	599, 301	8, 524, 569	14. 22	597, 065	8, 643, 902	14. 48
Mason's lime	469, 101	6, 140, 350	13. 09	469, 507	6, 236, 377	13. 28
Prepared masonry mortars	92, 586	954, 352	10. 31	99, 516	1, 057, 255	10. 62
Unspecified	73, 148	811, 257	11. 09	25, 175	294, 627	11. 70
Total	1, 234, 136	16, 430, 528	13. 31	1, 191, 263	16, 232, 161	13. 63
Chemical and other industrial:						
Alkalies (ammonium, potassium and sodium compounds)	(1)	(1)	(1)	(1)	(1)	(1)
Asphalts and other bitumens	(1)	(1)	(1)	(1)	(1)	(1)
Bleach, liquid and powder [2]	2, 504	29, 155	11. 64	4, 440	47, 246	10. 64
Brick, sand-lime and slag	23, 517	286, 980	12. 20	20, 575	240, 922	11. 71
Brick, silica (refractory)	17, 187	226, 444	13. 18	17, 616	225, 882	12. 82
Calcium carbide and cyanamide	576, 659	5, 592, 490	9. 70	558, 370	5, 333, 540	9. 55
Calcium carbonate (precipitated)	26, 988	305, 237	11. 31	20, 222	579, 971	28. 68
Coke and gas (gas purification and plant byproducts)	28, 681	317, 035	11. 05	29, 060	314, 896	10. 84
Explosives	4, 407	56, 313	12. 78	9, 569	107, 561	11. 24
Food products:						
Creameries and dairies	741	12, 137	16. 38	990	15, 012	15. 16
Gelatin	6, 444	78, 340	12. 16	6, 105	73, 863	12. 10
Stock feed	21, 109	235, 365	11. 15	21, 258	248, 993	11. 71
Other [3]	2, 432	32, 466	13. 35	1, 381	16, 330	11. 82
Glassworks	237, 479	2, 403, 987	10. 12	237, 172	2, 350, 536	9. 91
Glue	9, 598	109, 746	11. 43	7, 844	88, 618	11. 30
Grease, lubricating	7, 724	89, 691	11. 61	5, 187	58, 302	11. 24
Insecticides, fungicides, and disinfectants	79, 136	988, 996	12. 50	70, 347	879, 160	12. 50
Medicines and drugs	(1)	(1)	(1)	(1)	(1)	(1)
Metallurgy:						
Nonferrous-smelter flux	4, 063	64, 763	15. 94	1, 378	22, 859	16. 59
Steel (open-hearth and electric furnace flux)	1, 354, 883	14, 190, 178	10. 47	1, 222, 543	12, 994, 294	10. 63
Ore concentration [4]	275, 022	2, 869, 987	10. 44	305, 309	3, 376, 010	11. 06
Wire drawing	21, 495	282, 688	13. 15	24, 598	291, 550	11. 85
Other [5]	13, 601	145, 638	10. 71	10, 399	108, 714	10. 45
Paints	28, 536	352, 258	12. 34	25, 926	319, 508	12. 32
Paper mills	735, 393	8, 103, 544	11. 02	683, 628	7, 422, 034	10. 86
Petroleum refining	52, 509	621, 556	11. 84	40, 621	466, 869	11. 49
Rubber manufacture	1, 126	13, 579	12. 06	2, 028	23, 749	11. 71
Salt refining	9, 852	95, 809	9. 72	9, 677	91, 545	9. 46
Sewage and trade-wastes treatment	100, 553	1, 181, 644	11. 75	89, 338	1, 084, 262	12. 14
Soap and fat	753	9, 310	12. 36	815	9, 953	12. 21
Sugar refining	34, 941	529, 822	15. 16	35, 492	529, 293	14. 91
Tanneries	68, 239	784, 041	11. 49	64, 991	725, 065	11. 16
Varnish	(1)	(1)	(1)	(1)	(1)	(1)
Water purification	577, 953	6, 181, 745	10. 70	601, 592	6, 447, 682	10. 72
Wood distillation	5, 467	61, 563	11. 26	14, 206	126, 073	8. 87
Undistributed [6]	122, 459	1, 307, 902	10. 68	143, 999	1, 484, 346	10. 31
Unspecified	259, 846	2, 855, 753	10. 99	274, 731	2, 979, 364	10. 84
Total	4, 711, 297	50, 416, 162	10. 70	4, 561, 407	49, 084, 002	10. 76
Refractory lime (dead-burned dolomite)	1, 966, 460	26, 375, 313	13. 41	1, 928, 025	26, 098, 455	13. 54
Grand total [7]	8, 255, 512	96, 934, 611	11. 74	8, 073, 078	95, 231, 221	11. 80
Hydrated lime included in above distribution	1, 919, 783	24, 587, 110	12. 81	1, 882, 824	24, 459, 136	12. 99

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[1] Included with "Undistributed" to avoid disclosure of individual operations.

[2] Bleach used in paper mills excluded from "Bleach" and included with "Paper mills."

[3] Includes citrates, tartrates, and miscellaneous food products.

[4] Includes flotation, cyanidation, bauxite purification, and magnesium manufacture.

[5] Includes barium and vanadium processing, cupola, gold recovery, and unspecified metallurgical uses.

[6] Includes alcohol, alkalies, asphalt, medicines and drugs, oil drilling, petrochemicals (glycol), magnesium products, plastics, polishing compounds, retarder, sulfur, tobacco, varnish, and miscellaneous industrial uses.

[7] Includes lime used by producers (captive tonnage) as follows 1951: 535,179 tons, valued at $4,933,013; 1952: 485,635 tons, $4,952,140.

STATISTICS

Comprehensive statistics of the lime industry are compiled annually by the Bureau of Mines (7), and a canvass on a less detailed basis is also conducted monthly. Figures on imports and exports, both of which are small, are compiled by the Bureau of Mines from data collected by the United States Department of Commerce.

PRODUCTION AND CONSUMPTION

As stocks are small and fairly constant, production and consumption are virtually synonymous. Production (sales) of lime by principal uses for a 10-year period is indicated in table 2. The five leading consuming States in 1950, in order of importance, were Pennsylvania, Ohio, Illinois, New York, and Michigan. Both production (sold and used) and consumption by States are given each year in the Minerals Yearbook. Foreign trade in lime, which consists chiefly of movements to and from adjacent parts of Canada, is very small. The United States is self-sufficient in raw materials for lime manufacture.

Lime has strategic importance because it is essential to many vital industries. Its production is scattered widely in the United States. Stockpiling for security is unnecessary, as the United States is independent of foreign supplies. Stockpiling to supply unusual demands would be difficult because lime is perishable; it deteriorates rapidly unless kept in airtight containers. Unusual demands can be met by increasing production facilities.

TABLE 2.—*Sales of lime, in the United States by principal uses, 1943–52,*

[Short tons]

Year	Agricultural	Building	Chemical and industrial	Refractory	Total
1943	454, 133	557, 958	4, 307, 799	1, 276, 725	6, 596, 615
1944	466, 390	520, 000	4, 196, 383	1, 290, 790	6, 473, 563
1945	373, 410	549, 547	3, 810, 288	1, 187, 334	5, 920, 579
1946	384, 658	845, 604	3, 684, 455	1, 077, 983	5, 992, 700
1947	340, 363	1, 008, 211	4, 035, 202	1, 395, 203	6, 778, 979
1948	323, 300	1, 140, 518	4, 255, 403	1, 544, 755	7, 263, 976
1949	328, 528	1, 052, 097	3, 618, 969	1, 318, 708	6, 318, 302
1950	332, 687	1, 248, 989	4, 137, 297	1, 759, 443	7, 478, 416
1951	343, 619	1, 234, 136	4, 711, 297	1, 966, 460	8, 255, 512
1952	392, 383	1, 191, 263	4, 561, 407	1, 928, 025	8, 073, 078

PRICES

Prices of lime per ton (containers not included) at the point of manufacture during the past 25 years have ranged from a low of $6.28 in 1932 and 1933 to a high of $11.80 in 1952. Since 1940 prices have not advanced in consonance with the prices of commodities in general. The relatively low level of lime prices is due in part to the trend toward fewer and larger plants covered in a later section of this report. The high-cost plants have suspended production or have been absorbed and renovated by the larger and more efficient establishments.

Calcium metal is currently selling for about $2 a pound.

RESEARCH

The more progressive lime companies are conducting continuous research, chiefly on equipment design and methods of operation. This involves tests of various types of kilns, temperature and time of calcination, and fuel efficiency. The number of pounds of lime produced per pound of fuel used measures performance. A problem of current interest is the development by at least two lime companies of a process of pressure hydration of dolomitic lime. The method is used by several companies. Many improvements have also been worked out in quarry methods and equipment.

Institutional research sponsored by the National Lime Association currently includes that at Rutgers University on the use of lime in treating acid trade wastes, at Massachusetts Institute of Technology on correlation of lime-calcination conditions on the properties of lime, at the University of Texas on lime for road stabilization, at Franklin Institute, on mortar-bond strengths, and at Purdue University on breaking oil emulsions with lime. Several consulting engineers have contributed valuable data on calcination methods and equipment and on the properties of lime.

Government research has been conducted by the National Bureau of Standards and the Federal Bureau of Mines. The former Bureau has studied the use of lime in trade-waste disposal and has investigated lime plasticity. The Bureau of Mines made a comprehensive study of the effects of steam in the limekiln and published the results in a Bureau report

(*1*). In cooperation with a machinery company the Bureau ran many tests on the use of a modified sintering machine to calcine limestone (*13*). The Bureau also conducted research on limestone calcination (*8*), made fuel-efficiency studies (*12*), and outlined the properties and methods for manufacturing dead-burned dolomite (*16*).

OUTLOOK

A significant trend in the lime industry is toward fewer and larger plants. This is a long-continued rather than a recent trend. It was analyzed by the Bureau of Mines in 1939 (*4*) and later studies indicated its continuance. In 1910 there were 1,073 open-market lime plants; in 1930, 336; in 1940, 314; and in 1952 only 160. Nevertheless, over the same period production was increasing substantially.

Another long-range movement is in uses. Many years ago the building trades consumed most of the lime, and chemical uses were relatively few. For the past 25 years the trend has been toward larger and larger chemical uses, until in 1952 the chemical and industrial (including refractory lime) consumed over 70 percent of the total output.

Increasing quantities are used in road stabilization and in treating industrial waste. Development of new sources of supply is expected, particularly in areas where new lime-consuming industries are established. On the other hand, migration of lime-consuming industries or exhaustion of easily available limestone deposits may lead to a recession in lime production in certain areas.

The outlook for metallic calcium is one of moderate increase in consumption. Its relatively high tensile strength, lightness, and ease of fabrication have suggested its use in light structural alloys (*10*); however, its poor resistance to corrosion and other unfavorable considerations are serious obstacles. Calcium has other potentially larger uses—for example, in storage batteries, dehydration of organic liquids, and steel and other metallurgical applications.

PROBLEMS

Quarrying and mining problems are becoming increasingly complex. More and more operators are finding it necessary to mine underground. This involves new methods, new hazards, and modified equipment, with their attendant problems. Between 1922 and 1926 the Bureau of Mines made detailed studies of quarry methods and equipment used at lime plants. The results were published in Bureau of Mines Bulletin 269 (*5*). Since that time conditions have changed greatly. Underground methods have been introduced at a greater number of operations; improved stripping equipment has been developed; and truck haulage has replaced track haulage at many places. Comprehensive studies designed to improve the efficiency of mining methods and equipment are desirable.

In the lime-manufacturing process calcination is of primary importance, and the technique of efficient calcination has not yet been mastered. Rapid calcination at high temperatures may give different hydration properties and different plasticity of the lime produced than would result from a slower calcination at a lower temperature. The interrelation of these various factors has not yet been fully determined. The interrelation of time and temperature of calcination and the qualities of the finished product is a research problem of major importance.

The design of calciners is being changed. Five or six different types are used, and many variations in each type are possible. A determination of the relative merits of various types of calcining equipment is a field of useful research.

As nearly half of the weight of the limestone used is dissipated into the air in the form of carbon dioxide during calcination, wider recovery and utilization of this waste material are problems for the industry to solve.

High-calcium lime hydrates quickly and easily, but the resistance of dolomitic lime to hydration presents a difficult problem. Incomplete hydration of dolomitic lime causes poor service. For instance, a wall plaster containing unhydrated particles may become pitted or ruptured by a process of slow hydration after the plaster has set. Satisfactory results have been obtained by pressure hydration, but fundamental studies of the hydration process are justified.

At lime plants using the ordinary types of shaft kilns, about 25 percent of the stone quarried is in sizes too small for use. This high percentage of waste presents problems of utilization that demand active study. Limestone has so many uses that there are numerous avenues of approach to waste-utilization problems. The use of rotary kilns provides a means of utilizing the smaller sizes, but such kilns are costly. Some progress has been made in designing vertical-type calciners that will calcine small stone.

Calcium problems involve research on the physical and chemical properties of the metal and its alloys with the objective of developing new uses and processes.

Producers, consumers, and Government agencies concerned with this commodity need more comprehensive technical, statistical, and economic information to guide their activities.

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LITHIUM

By

Joseph C. Arundale[1]

LITHIUM is the lightest of the metallic elements. Although discovered nearly 150 years ago and widely distributed in the rocks and waters of the earth, it was little known and little used only a few years ago. An excellent example of what can be accomplished by research, the various minerals and compounds of lithium now are produced in large tonnages and have scores of important uses; the list is growing rapidly.

Summary

The five most important sources of lithium are the minerals spodumene, lepidolite, petalite, and amblygonite (which are recovered only from pegmatites) and the compound dilithium-sodium phosphate (which is recovered from brine). The principal known domestic reserves are in North Carolina, in the Black Hills of South Dakota, and in California brines. Inferred reserves of spodumene are large. Reserves of lithium in brines also are large. Domestic reserves of lepidolite, petalite, and amblygonite are small compared with present requirements.

Lithium minerals are recovered by froth flotation, sink-float, or hand picking. They are processed by roasting with either an alkali or an acid; the resulting soluble lithium salt is dissolved in water and precipitated or reacted with other materials to form various lithium compounds. Lithium metal is produced by electrolysis of the chloride.

Lithium is used in its mineral forms, in metal form, or as various lithium compounds. Its long list of applications includes lubricants, ceramics, glass, metallurgy, air conditioning, welding and brazing, organic chemicals, dry cells, and alkaline storage batteries.

The outlook is for new and expanded uses for lithium minerals. Producers are increasing their capacity. The problems include the need for determining requirements and evaluating market potential, reducing costs, more efficient recovery, development of new uses, and additional information on reserves.

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[1] Assistant chief, Branch of Construction and Chemical Materials, Bureau of Mines.

BACKGROUND

The following table shows the approximate chemical composition and lithia (Li_2O) content of the commercial lithium minerals:

Mineral	Approximate chemical composition	Range in commercial mineral, percent lithia (Li_2O)
Spodumene	$LiAlSi_2O_6$	4–7
Lepidolite	$K_2Li_3Al_4Si_7O_{21}$ (OH, F_3).	3–4
Amblygonite	$LiAlFPO_4$	8–9
Dilithium-sodium phosphate.	Li_2NaPO_4	19–21
Petalite	$LiAlSi_4O_{10}$	2–4

Lithium minerals have been mined in the United States for over 50 years. The first large-scale production was spodumene from the Etta mine in the Black Hills. The first commercial use of lithium compounds was in lithiated waters and lithia tablets, which were reputed to have some therapeutic value. Later lepidolite was produced in California and New Mexico for use in glassmaking. Invention of the Edison alkaline storage cell provided another market for lithium compounds. Other deposits of lithium minerals were developed in the Black Hills. Some of the more important of these were the Edison, Ingersoll, Hugo, Tin Mountain, Mateen, Peerless, Tinton, Beecher, and Dyke lode. A few years before World War II the compound dilithium-sodium phosphate was recovered as a byproduct from the brine of Searles Lake, Calif., and by the end of the war had become an important source of raw material for lithium chemicals. During this same period the Bureau of Mines and the Tennessee Mineral Products Corp. jointly developed a process for recovering spodumene from the deposits in the Kings Mountain district of North Carolina. The Solvay Process Co. produced spodumene in this district from 1943 to 1945.

In World War II military requirements, particularly for lithium hydride as a carrier of hydrogen for inflating signal balloons and lithium hydroxide as a carbon dioxide absorbent in submarines, pushed production and utilization of lithium to record highs; however, when Government contracts were canceled at the end of 1944, many lithium producers were left with excess capacity, and at that time other markets of comparable size were not available. A few mines and lithium-products plants continued to operate on a limited scale; some were put in standby condition, and others were abandoned. During the next few years the remaining producers intensified their research and market-development programs. As a result, important new uses were developed, and the market for lithium minerals and compounds in such applications as glass, ceramics, and lubricants has greatly expanded in the past few years. Consequently, the lithium industry has grown rapidly.

The important lithium minerals are recovered from pegmatites or veins closely related to pegmatites and from brine.

OCCURRENCES

The bulk of the spodumene produced in the United States in the past has come from the southern Black Hills of South Dakota in the area including the towns of Pringle, Keystone, and Custer. Spodumene, lepidolite, and amblygonite and small quantities of other lithium minerals have been mined from numerous pegmatites in this area for many years. Other and probably more important deposits occur in an area extending from a point just east of Lincolnton, N. C., southwest to a point near Grover on the State line. This area is roughly 25 miles long, with a maximum width of about 2 miles, and generally is referred to as the Kings Mountain district. Spodumene-bearing pegmatites occur in this area.

Another important occurrence is the brine of Searles Lake, Calif., which contains a small percentage of lithium salts in solution.

Other deposits of various lithium minerals are known in San Diego County, Calif. (principally amblygonite and lepidolite); the Harding mine in northern New Mexico (lepidolite and spodumene); Gunnison and Fremont Counties, Colo. (lepidolite); Fremont County, Wyo. (lepidolite, amblygonite, and petalite); and numerous other localities in the United States. Lithium is also present in low concentrations in many brines, saline waters, and muds in several areas but currently is being recovered commercially only from Searles Lake.

Lithium-bearing pegmatites also occur in other countries. Important deposits of spodumene occur in the Provinces of Quebec and Manitoba, Canada, and in the great pegmatite at Manono in Belgian Congo. Lepidolite, amblygonite, and petalite occur in the pegmatites of southern Africa. Numerous lithium-bearing pegmatites are known in Brazil and other countries.

RESERVES

Measured reserves appear to be relatively small compared with projected demand, but the numerous deposits mentioned above have not been thoroughly explored. Geologists and company officials agree that reserves may prove to be, in reality, larger than present geologic estimates indicate.

The Geological Survey has estimated the lithia content of indicated reserves in the Kings Mountain district at about 4 million units (a unit being 1 percent of 1 short ton, or 20 pounds) in ore averaging 1.7 percent lithia and about 124 million units in inferred ore averaging 1.3 percent lithia. Known deposits in the Black Hills probably contain at least 1 million units of lithia.

Searles Lake is a desert salt pan or playa consisting of a body of crystalline salts to a depth of 60 to 90 feet and known to be at least 20 square miles in area. The porosity or proportion of voids in the salt body has been estimated at 25 to 45 percent. The saturated brine from these voids contains lithium in the proportion of about 1 ton of dilithium-sodium phosphate to each 1,750 tons of brine.

The magnitude of reserves in other countries is less well known but certainly very large in aggregate.

PRODUCTION

Four firms predominate in the lithium industry; each has its own supply and processes its own raw material, and each manufactures lithium compounds. These four firms are:

American Potash & Chemical Corp., which recovers dilithium-sodium phosphate as a by-product from the brines of Searles Lake, Calif. This material is processed in its plant at Trona, on the edge of the lake. In concentrating the brine, the lithium is precipitated as dilithium-sodium phosphate. This material is recovered by treating with a fatty acid derivative, adding a light mineral oil, and floating the dilithium-sodium phosphate by aeration. The resulting froth is filtered and dried. The final product, averaging 20 to 21 percent Li_2O, is converted to lithium carbonate at Trona. Lithium halide solutions are produced at the company's Eston Division plant. A new company—American Lithium Chemicals, Inc.—has been formed by American Potash & Chemical Corp. and Bikita Minerals (Pvt.), Ltd., for constructing a new plant at San Antonio, Tex., to produce lithium chemicals from lepidolite from Southern Rhodesia. Bikita Minerals owns and operates the Rhodesian property under the management of Selection Trust, Ltd., London. Associated with Selection Trust in Bikita Minerals are American Metal Co., Ltd., and American Potash & Chemical Corp.

Lithium Corp. of America produces spodumene from the Edison, Mateen, and Beecher No. 2 mines in the Black Hills of South Dakota. The last two are open-pit operations, and the Edison is an underground operation. The spodumene is recovered as a flotation concentrate in the company mill at Hill City, S. Dak., and in a sink-float unit at Keystone. It is processed into lithium compounds and metal in a plant near Minneapolis. This firm is constructing additional facilities at Bessemer City, N. C., in the Kings Mountain district to produce lithium salts.

Maywood Chemical Works operates the Etta mine in the Black Hills by a gloryhole system. Because of the nature of this ore (large crystals) it is amenable to hand picking. The high-grade lump spodumene thus procured is processed into lithium compounds and metal at the firm's plant at Maywood, N. J.

Foote Mineral Co. produces spodumene ore in the Kings Mountain district of North Carolina, and the spodumene is recovered by flotation in a nearby mill. The spodumene is processed into lithium compounds at Exton, Pa., and Sunbright, Va.

In addition to these firms there are a few smaller and intermittent producers in the Black Hills and in New Mexico and Colorado.

In 1952 the total domestic production of lithium minerals (including dilithium-sodium phosphate) was about 15,600 short tons, with an estimated 1,088 tons of contained lithia. Since then the rate of output has increased greatly.

TABLE 1.—*Shipments of lithium ores and compounds from mines in the United States, 1935–39 (average) and 1940–52*

Year	Ore (short tons)	Value (estimated)	Li_2O (estimated; short tons)
1935–39 (average)	1, 327	$48, 280	88
1940	2, 011	80, 679	113
1941	3, 832	115, 718	209
1942	5, 405	243, 516	299
1943	8, 155	314, 660	463
1944	13, 319	552, 977	848
1945	2, 446	285, 520	274
1946	3, 065	303, 892	323
1947	2, 441	151, 113	199
1948	3, 881	210, 792	291
1949	4, 838	345, 970	475
1950	9, 306	579, 922	747
1951	12, 897	896, 000	956
1952	15, 611	1, 052, 000	1, 088

Southern Africa is the only other area in which significant quantities of lithium minerals now are being produced. In 1950 that area had a reported output of 9,318 short tons of lepidolite, 292 tons of amblygonite, and 180 tons of petalite as hand-cobbed material. Most of the lepidolite and petalite was exported to the United States; some lepidolite went to France, and the amblygonite went to Germany and United Kingdom.

TECHNOLOGY

In general, two methods are employed to recover lithium compounds from lithium minerals—one an acid treatment, the other an alkali treatment. In the first method alpha spodumene is converted to beta spodumene by calcining at critical temperatures. Sulfuric acid is added to this and the mixture heated with calcium carbonate; lithium sulfate is formed. The lithium sulfate is leached from the mass and filtered. To the aqueous solution of lithium sulfate is added soda ash to precipitate lithium carbonate. This is the basic material from which other compounds are derived. Calcium hydroxide is added to the lithium carbonate to form lithium hydroxide monohydrate. To the lithium carbonate may be added hydrochloric acid, producing a lithium chloride solution which upon evaporation is converted to the anhydrous lithium chloride.

Lithium metal is produced by electrolysis of a fused mixture of lithium chloride and another alkali salt—for example, potassium chloride. Lithium metal can be reacted with hydrogen to produce the lithium hydride. The addition of ammonia to lithium hydride produces the lithium amide.

Details of the second method being used to recover lithium compounds from spodumene have not been made public.

The Bureau of Mines has devised several methods for recovering lithium from its ores. These methods are described in detail in the reports listed in the bibliography.

The domestic industry produces a long line of lithium compounds, metal, and alloys. Lithium compounds are also produced on a small scale in Germany and United Kingdom, using largely amblygonite from South-West Africa, and in France, using lepidolite from the same area.

USES

Lithium minerals have several direct applications. Lepidolite reduces the viscosity and improves the workability of a glass melt, provides a hard surface, modifies the coefficient of expansion, and increases the strength and reduces the weight of the glass. Such glasses are used in sealed-beam headlights for automobiles, trucks, and airplanes, in glass tubes in radio, television, radar and some neon-sign tubing, and in other special glasses. Spodumene also is used directly in certain ceramic batches. Petalite is used in ceramics to control shock resistance.

The various compounds of lithium have more numerous applications.

GLASS AND CERAMICS

One of the principal uses for lithium compounds at present is in ceramics. Lithium carbonate and certain other lithium compounds have the property of forming eutectics with alkaline oxides and with silica. This property is utilized in producing glasses, glazes, and enamels with high gloss and scratch and chemical resistance. In these glasses and in ceramics, lithium carbonate or other lithium compounds replace alkali carbonates in whole or in part as a flux, which keeps the alkali content to a minimum. In enamels lithium compounds reduce the maturing temperature and increase the fluidity, permitting use of thinner steels and lower firing temperatures as well as use of refractory oxides to increase acid resistance. Lithium compositions have displaced lead oxide to some extent in many glasses and glazes.

LUBRICANTS

Another large use of lithium compounds is in lubricants. In this field some of the so-called "all-purpose greases" contain lithium stearate (or other soap) as the metallic soap component. This compound is made by reacting lithium hydroxide monohydrate with stearic acid. The resultant greases retain their lubricating properties through extremes of temperature. They also have good water resistance. Such greases are used in military artillery, aircraft, trucks, automobiles, and a wide range of industrial purposes. The quantities employed in military greases range from about 50 to 75 pounds of grease per pound of lithium hydroxide monohydrate. In industrial uses it ranges from 50 to 100 pounds of grease to 1 pound of lithium hydroxide monohydrate. Others, such as agricultural equipment, trucks, and service-station greases, range from about 45 to 80 pounds of grease to 1 pound of lithium hydroxide. The average composition of all the lithium-base greases, based on past consumption data, is about 1½ percent lithium hydroxide monohydrate.

AIR CONDITIONING

In air-conditioning and industrial drying a saturated solution of lithium chloride reduces humidity and removes objectionable odors and dust from the air. Lithium chloride is one of the most hygroscopic of all inorganic compounds. A saturated solution will dry air passed in contact with it down to a very low relative humidity. Lithium bromide is used in gas-absorption type air conditioning.

ALKALINE STORAGE BATTERIES

Another important use for lithium compounds is in the Edison-type nickel-iron storage battery. These batteries are used in electric industrial trucks, mine locomotives and shuttle cars, portable lighting outfits, power supply for various devices such as chronographs, laboratory equipment and seismographs, and as a standby power supply for elevators, engine stops, fire alarms, railway signals, and many other uses. The positive plate of the cells of these batteries consists of a nickeled-steel perforated tube containing nickel oxide and nickel flake. The negative plate is finely divided iron oxide in nickeled-steel pockets. The container is made of nickeled steel with welded joints, and the electrolyte is an aqueous solution of potassium and lithium hydroxide. Lithium hydroxide monohydrate is added as a catalytic agent and to increase the capacity and life of the cell.

DRY-CELL BATTERIES

Conventional zinc or ammonium chloride electrolyte dry cells become very inefficient as the temperature is lowered and become almost completely inactive at about —20° F. Lithium chloride or bromide in the electrolyte will lower the temperature at which these batteries are still active. At —40° F. dry-cell batteries using lithium chloride still can deliver about 10 percent of the 70° F. capacity (the standard rating temperature). This is not as efficient as is desirable but is the best that now can be done.

BRAZING AND WELDING

Lithium fluoride and chloride are used as oxide scavengers and metal cleaners in welding or brazing aluminum and magnesium.

ORGANIC CHEMISTRY

Within the past several years lithium metal has assumed importance in organic synthesis. It is used instead of magnesium in some Grignard reactions. Synthetic vitamin A is produced by processes utilizing lithium alkyls.

MISCELLANEOUS USES

Other uses for lithium and its compounds, which in aggregate account for only a small percent of the total consumption, include the preparation of dense, oxygen-free, high-conductivity copper, heat-treating metal, iron castings, powder metallurgy, cosmetics, medicinals, paints, waxes, titania pigments, and others.

The lithium industry is an excellent example of what can be done in developing new uses for mineral materials in plentiful supply. Lithium which was a little known and little used material, now is the foundation of a multimillion-dollar business. It has many essential military applications and fills a variety of important civilian requirements. Many phases of this research are being continued both by industry and Government. As a result of such research, expanding markets, and improved technology the prices of the principal lithium products are approximately the same as they were 20 years ago, despite the financial inflation during that period.

PRICES

The principal lithium compounds, lithium carbonate and lithium hydroxide, both sell for under $1 per pound. Prices are not quoted for lithium minerals, but spodumene reportedly has been sold recently at about $15 per unit of lithia. Lithium metal, 98 percent pure, is quoted at $13 to $20 a pound, depending on quantity.

TARIFF

Lithium minerals are imported duty free. Lithium metal is dutiable at 25 percent ad valorem. Lithium compounds are dutiable at 12½ percent ad valorem.

OUTLOOK

Increasing demand and a favorable outlook for new and enlarged uses for its products have encouraged the lithium industry to expand.

There are several possible new domestic sources of lithium. Many brines, salines, and associated deposits are known to contain lithium. Great Salt Lake and the Salduro Marshes are examples. The large pegmatite deposits in Manitoba and Quebec may be developed.

Increased use of lithium in some current applications is anticipated. Only a small portion of the grease being manufactured contains lithium, but greater use in the lubricating field presents a large potential market. Lithium minerals and compounds are finding increasing use in glass and ceramics, such as sanitary ware, whiteware, glazes, electrical porcelains, and porcelain enamels and in air conditioning, batteries, and pharmaceutical intermediates. In metallurgy, expanded use is possible as a degasifier, grain refiner, furnace atmosphere conditioner, and brazing and welding flux. Lithium stearate in vegetable and petroleum waxes modifies their properties and points to the possibility that materials, such as low-price paraffins, might be substituted for the higher price, scarce waxes, such as carnauba, montan, and candelilla. The list of potential uses is long and includes well-drilling muds, bleaches, luminous pigments, dental-impression material, light-metal alloys, propellants, fertilizers, food preservatives, optical instruments, and many others.

PROBLEMS

The lithium industry faces a number of problems of varying degrees of importance. Determination of requirements and evaluation of market potentialities of the various lithium minerals and compounds and other technical and statistical data are needed both by industry and Government in planning future action.

Most lithium compounds are relatively high cost materials, and this is a deterrent to their wider utilization. Reduction of these costs is an objective of research.

Recovery of lithium minerals from their various ores by known methods is inefficient compared to recoveries of other mineral materials. Pegmatites characteristically contain, in varying proportions, several of the following commercial minerals: Feldspar, quartz, mica, beryl, spodumene, amblygonite, lepidolite, petalite, pollucite, columbite, tantalite, cassiterite, and monazite. Some of these are strategic and critical, others are in abundant supply. In all pegmatite operations, efficient recovery and economic disposition of the maximum number of coproducts or byproducts should be a goal. Recovery of lithium compounds from the various minerals also is inefficient. Improved processes for recovering lithium from ores and brines are needed.

Although inferred reserves of spodumene are large, they are not unlimited, and measured reserves are relatively meager. Therefore, development of adequate reserves for a greatly expanded industry is a long-range problem. Domestic reserves of certain other lithium minerals are small in terms of current consumption and anticipated requirements. Lepidolite, amblygonite, and petalite, all of which have important and growing uses, are relatively limited in supply. The problem of lepidolite as a source of cesium and rubidium is discussed in other chapters.

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MAGNESIUM

(Including Magnesium Compounds)

By

H. B. Comstock[1]

MAGNESIUM, the world's lightest structural metal, has achieved a place as a mainstay in worldwide development of air travel and air transportation. Born of military need and derived from raw materials in almost unlimited domestic supply, magnesium won approval during two world wars as a substitute in many installations for aluminum, copper, zinc, and lead. During the decade following World War II, magnesium became established as a structural metal for civilian uses.

Summary

This chapter is in two sections: (1) Magnesium and (2) Magnesium Compounds. The abundance and wide distribution of the principal magnesium-rich minerals—dolomite, magnesite, brucite, serpentine and olivine—would make the United States self-sufficient in magnesium and its compounds, even without the limitless supply known to exist in the oceans that border the United States. Sea water contains about 0.13 percent of magnesium. If 100 million tons of magnesium a year was taken out for 1 million years, the concentration in the seas would drop to approximately 0.12 percent, assuming that no more washed into the waters in the meantime. Because domestic resources place no limit on the quantities of magnesium that can be produced, the metal is especially attractive from a security point of view. Because of the strategic value of the metal for military use, the rate of expansion during the period 1918–43 was on a scale unparalleled by any major basic metals industry in the United States. By the close of World War II this lightest of the structural metals was established on its own merits.

A plentiful supply of relatively low cost power in the United States encouraged development of the electrolytic process as the most economical method of producing magnesium; however, the silicothermic plants, built during World War II and reactivated during the Korean emergency, have been proved to be advantageous as standby plants besause of the simplicity of design and operation.

Although the greater portion of production of magnesium during 1951 and 1952 was for defense, either for the National Stockpile or for military equipment, a continually widening substitution of magnesium for other metals was indicated. In 1953 this substitution began to prove effective in building and expanding a peacetime magnesium industry. Fabrication techniques had been improved, and the increased range of industrial alloys had been broadened. Over half of the magnesium consumed in the United States during that year was for civilian requirements. By the close of 1953 the commercial highway-transportation field constituted one of the largest and fastest growing markets for magnesium.

Research was continued throughout 1953 to develop magnesium alloys with improved tensile strength and corrosion resistance. Improvements were made in 1953 in facilities for casting, extrusion, forging, spinning, drawing, rolling, and pressing. The most outstanding step in this direction was noted

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[1] Commodity-industry analyst, Bureau of Mines.

when a high-speed, high-production rolling mill was started at Madison, Ill., and installation was begun there of a 14,000-ton extrusion press. As a result of this advance in technology, the price of magnesium sheet and plate was reduced on March 15, 1953, to bring it to a price per unit volume comparable to that of aluminum.

Magnesia refractories began to be used extensively in the steel industry in 1900. The demand for these refractories has steadily increased with increased production of basic open-hearth steel. Before World War I almost all of the crude magnesite mined in the United States was used in manufacturing paper. During World War I refractories consumers sought domestic sources, and by 1917 production of crude magnesite in the United States reached 317,000 short tons. Between World Wars I and II about 70 percent of the refractory magnesia used in the United States was produced from domestic crude magnesite. During World War II, when imports were cut off, the marked increase in demand for refractory magnesia by the steel industry was met by enlarging domestic mining and calcining facilities. The capacity of domestic producers of periclase from sea-water bitterns was greatly increased during World War II, and plants were built in California and New Jersey to produce magnesia from sea water and calcined dolomite. By 1952 magnesia-production facilities had been expanded into Ohio and Virginia.

In addition to refractories produced for the steel and copper industries, magnesium compounds are required in the cement, fertilizer, textiles, and chemicals industries.

Improvements in refractories from magnesia in 1953 resulted from demands for more efficient refractories in certain metallurgical and chemical processes. These processes required extremely high temperatures to meet the engineering developments in jet propulsion, atomic energy, and guided missiles.

MAGNESIUM

BACKGROUND

HISTORY, SIZE, AND ORGANIZATION OF MAGNESIUM INDUSTRY

The manufacture of magnesium on a commercial scale was started early in the 20th century at Stassfurt, Germany. It was first produced there from magnesium chloride obtained as a byproduct in extracting potash from the large deposits of carnallite (a hydrous chloride of potassium and magnesium).

Magnesium metal in an impure state was first reported to have been obtained by Sir Humphrey Davy in 1808 by electrolysis. In 1833 Michael Faraday prepared magnesium by electrolysis of fused anhydrous magnesium chloride. Later, in 1852, Robert Bunsen developed an electrolytic cell for this purpose. It was by the use of a modification of Bunsen's electrolytic cell that the German production was begun, and by 1909 it had advanced from the pilot-plant stage to limited industrial production and use (*17*).[2]

Commercial production of magnesium in the United States began in 1915, soon after imports from Germany were cut off. Before that year Germany had been the sole supplier of the metal to the world. The General Electric Co. at Schenectady, N. Y., was the first to begin active production of magnesium in the United States (*22*, *23*).

During 1916 and 1917 eight different companies operated plants for production of magnesium in the United States, though not simultaneously. Most of these companies ceased production after World War I, and by 1920 only two companies remained: The Dow Chemical Co. and the American Magnesium Corp., which in 1919 had become a subsidiary of the Aluminum Co. of America. In July 1927 American Magnesium Corp. discontinued production of magnesium and limited its activities to fabrication. This left Dow Chemical Co. the sole producer of primary magnesium in the United States (*47*).

Production of magnesium in the United States increased virtually every year after 1921, although until 1939 these increases were small. The demand for magnesium by the Allied Powers, resulting from the outbreak of World War II, was intensified by the defense program of the United States, which began in 1940.

The United States Government placed magnesium under mandatory priority status in March 1941, and an investigation of all production processes other than the well-established electrolytic process was begun. Between 1939 and 1943, 15 plants were built in the United States, of which 13 were constructed by using Government funds. During this period the Dow plant at Freeport, Tex., was built. All of the Government-owned plants were operated under management contracts with various companies, as shown in table 1 (*29*). This table also lists the production process employed in each plant and Federal expenditures through July 31, 1945.

Production of primary magnesium in the United States reached its peak in 1943 at 183,584 tons, and the consumption peak was reached in 1944 at 132,698 tons, all for war purposes. In 1944 production dropped to 157,000 tons, which still exceeded consumption for that year; and in 1945 production fell to 32,792 tons, with consumption that year 43,987 tons. When reports of production reached a figure above 20,000 tons for the first quarter of 1944, which was far above requirements, the Government removed magnesium from priority controls and began closing the plants, some of which had never reached capacity production. By July 1944, 5 Government-owned plants had been closed outright, and the output of 4 others had been limited. By November 1945 the defense plants had been shut down and the privately owned plant at Freeport, Tex., was the only active producer. The Dow Chemical Co. had closed its Midland, Mich., plant earlier that year after deciding that magnesium could be produced more economically in its Freeport, Tex., plant.

On December 7, 1945, the Surplus Property Administration filed its report to Congress on magnesium plants and facilities. This report listed the Government-owned magnesium production and fabrication facilities, outlined the disposal problems, recommended disposal programs, and proposed that certain magnesium-producing plants be retained by the Govern-

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

TABLE 1.—*Magnesium-production plants operated during World War II*

GOVERNMENT-OWNED PLANTS

Location	Operator	Rated annual capacity (short tons)	Federal expenditures through July 31, 1945
Electrolytic process			
Henderson, Nev.	Basic Magnesium, Inc	56,000	$132,687,768
Velasco, Tex	Dow Magnesium Corp	36,000	56,578,592
Lake Charles. La	Mathieson Alkali Works, Inc	27,000	48,848,228
Ludington and Marysville, Mich	Dow Magnesium Corp	36,000	39,853,547
Austin, Tex., and Carlsbad, N. Mex	International Minerals & Chemical Corp	12,000	18,767,767
Painesville, Ohio	Diamond Magnesium Co	18,000	14,823,131
Freeport, Tex. (one-half plant)	Dow Chemical Co	9,000	8,246,869
Total		194,000	319,805,902
Silicothermic process [1]			
Spokane, Wash	Electrometallurgical Co	24,000	16,131,470
Dearborn, Mich	Ford Motor Co	20,000	10,467,978
Wingdale, N. Y	Amco Magnesium Co	5,000	7,075,165
Manteca, Calif	Permanente Metals Co	10,000	6,219,766
Canaan, Conn	New England Lime Co	5,000	4,922,366
Luckey, Ohio	Magnesium Reduction Co	5,000	4,823,517
Total		69,000	49,640,262
Grand total		263,000	369,446,164

PRIVATELY OWNED PLANTS

Location	Operator	Rated annual capacity (short tons)
Midland, Mich. (electrolytic process)	Dow Chemical Co	9,000
Freeport, Tex. (one-half plant—Electrolytic process)	do	9,000
Permanente, Calif. (Carbothermic process)	Permanente Metals Corp	12,000
Total owned plants		30,000

ment. The plants at the following locations were retained and placed in standby condition:

Velasco, Tex.
Luckey, Ohio.
Painesville, Ohio.
Spokane, Wash.
Canaan, Conn.
Wingdale, N. Y.
Manteca, Calif.

It was stated that the total production capacity of these plants, together with the Freeport plant, should insure adequate production for any future emergency (*44*).

The Government-owned plants at the locations listed below had proved to be uneconomic producers, either because of high operating costs or poor physical condition.

Ludington and Marysville, Mich.
Las Vegas and Gabbs, Nev.
Austin, Tex. and Carlsbad, N. Mex.
Dearborn, Mich.
Lake Charles, La.

They were disposed of and torn down, or the buildings were used by other industries or tax-supported agencies. This, as in the period 1927–40, left the Dow Chemical Co. again the sole producer of primary magnesium. Its Freeport, Tex., plant with capacity of 18,000 tons at the close of World War II, remained the only domestic producer of primary magnesium until reactivation of the Government-owned standby plants was begun in 1950. Annual productive capacity of the Freeport plant had reached 23,000 tons at the close of 1951 and 29,500 tons before December 31, 1953.

Six Government-owned plants held in the National Industrial Reserve were reactivated in 1950, and production started in March 1951. The Luckey, Ohio, plant was not included in this reactivation program. Beryllium was produced in this plant for the Government atomic energy program throughout the Korean emergency. The reactivated plants were operated

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[1] All of the above silicothermic plants were equipped to alloy all of their production to save, if possible, the melt loss incurred each time magnesium is heated.

for the Government on management contracts and/or leases by various industrial organizations to produce magnesium for defense. By December 1951 the reactivated plants were producing at the rate of 60 percent of their total capacity. By June 30, 1952, their rate of production had reached 87 percent of total rated capacity. Table 2 shows the quantity of magnesium produced for defense during the Korean emergency by the Government-owned plants.

TABLE 2.—*Production of magnesium in reactivated Government-owned 'plants, 1951–53*

Plant location	Date production began	Date production ceased	Production (tons)
Electrolytic process:			
Velasco, Tex	Apr. 16, 1951	(1)	91, 695
Painesville, Ohio	July 27, 1951	June 3, 1953	29, 106
Silicothermic process:			
Canaan, Conn	Mar. 27, 1951	June 30, 1953	7, 650
Manteca, Calif	June 8, 1951	July 10, 1953	16, 953
Spokane, Wash.[2]	Aug. 15, 1951	June 30, 1953	14, 383
Wingdale, N. Y	Nov. 15, 1951	May 29, 1953	4, 390
Total			164, 177

[1] Production continued throughout 1953.

[2] Shut down to one-fourth capacity on Aug. 31, 1952, owing to electric power shortage in the Northwest. Annual rated capacity reduced to 20,000 tons on Aug. 1, 1952.

By March 31, 1953, the reactivated plants had produced 122,734 tons, or more than 60 percent of the total quantity of magnesium specified in the contracts. A Government directive dated April 7, 1953, scheduled the following plans for cessation of magnesium production for defense in the Government-owned plants: The Canaan, Conn., and Spokane, Wash., plants to cease production by June 30, 1953, or sooner if possible; the Painesville, Ohio, Wingdale, N. Y., and Manteca, Calif., plants to be shut down by June 30, 1953, or sooner if possible, and placed in such standby condition that they could be reactivated to produce at 50 percent of rated capacity during the first year of reactivation; and the Velasco, Tex., plant to continue in operation by extension of the contract with Dow Chemical Co. to produce magnesium through June 30, 1954. The Velasco contract was later extended to January 31, 1955.

In June 1953 the operators of the Canaan, Conn., plant signed a lease agreement to begin immediately to produce magnesium and calcium in that plant for the atomic energy defense program.

GEOGRAPHIC DISTRIBUTION OF MAGNESIUM INDUSTRY

The major portion of the magnesium industry is east of the Mississippi River. In 1940 The Dow Chemical Co. chose Freeport, Tex., as the location for a new electrolytic-process magnesium plant where sea water was readily available as source material and the waste materials could be easily turned back to the sea; and the supply of low-cost natural gas was plentiful in Texas at that time for use in generating electric power. Production of magnesium was begun at the Freeport plant on January 21, 1941. This was the only privately owned producing primary magnesium plant in the United States at the close of 1953.

Table 2 shows the location of all of the primary magnesium plants owned by the Government at the close of 1953, except for the 5,000-ton Luckey, Ohio, plant, which was not reactivated during the Korean emergency.

In 1953 approximately 85 percent of the magnesium foundries were east of the Mississippi River, and more than 90 percent of the wrought products were produced in that area.

TECHNOLOGY: GEOLOGY, MINING, MILLING, SMELTING, ETC.

Magnesium constitutes 2.5 percent of the lithosphere and is the sixth most plentiful element. It is a chemically active metal which unites readily with many other elements, and a great expenditure of energy is necessary to set it free. The abundance of sea water and mineral deposits as raw materials for production of magnesium lends added interest to the development of a wider field of application of this lightest of structural metals. Sea water contains about 0.13 percent of magnesium.

REDUCTION PROCESSES

Two processes of reduction were employed in the United States on a commercial scale in 1953: The electrolytic and silicothermic processes.

Electrolytic Process.—In the electrolytic process, as practiced at the Freeport and Velasco, Tex., plants, sea water is mixed with a slurry of calcium hydroxide or milk of lime. The magnesium in solution in the sea water is precipitated as magnesium hydroxide. The precipitated magnesium hydroxide settles out of the sea water in large settling tanks or thickeners and is pumped from the bottom of these tanks as a thin slurry containing about 17 percent magnesium hydroxide by weight. Precipitation of the magnesium permits elimination of about 98 percent of the volume of sea water in the first step of the process.

The magnesium hydroxide slurry is filtered and neutralized with hydrochloric acid to form a 15-percent solution of magnesium chloride. The magnesium chloride solution is evaporated and dehydrated to a solution having the approximate composition of $MgCl_2 \cdot 1\frac{1}{2}\ H_2O$. Dehydration is completed in an electrolytic cell where magnesium chloride is decomposed into magnesium metal and chlorine and HCl gases. These gases are recycled to produce hydrochloric acid. The Dow cell is used in all of the electrolytic plants in the United States (*27*, *40*).

Silicothermic Process.—The following ratio of raw material was required in the Government's silicothermic plants to produce 1 lb. of magnesium crystals (99.8+ percent Mg): Raw dolomite, 13.7 lb. (which calcined to 6+ lb.) and 1.25 lb. of 75-percent-grade ferrosilicon. The mixture was briquetted, and the briquets were charged into tubular retorts. The retorts were heated and evacuated. The magnesium oxide in the calcined dolomite was reduced by the silicon, producing magnesium vapor and dicalcium silicate. The magnesium vapor was condensed into crystals on a removable condenser that projected from the furnace. By this process metallic magnesium was produced in one operation directly from the calcined dolomite (*38*). The major disadvantage in the silicothermic process of reduction was batch operation, with consequent high labor costs and production time lost while retorts were cleaned and repaired.

The magnesium crystals produced in the condensers of the silicothermic plants are melted and refined, and the molten metal is ladled into the casting machine to form ingots. Melting and refining losses average 5 to 10 percent of "crystal" weight.

Dolomite is the source of magnesium in the Government-owned silicothermic plants (see also Part II: Magnesium Compounds).

During World War II other methods of reduction were tried, but did not at that time prove to be economically feasible. They are discussed under "Research."

MAGNESIUM ALLOYS

Many magnesium-base alloys were in regular production by 1953. They have characteristics that make them distinctive among metals; they are the lightest of the structural alloys, being about two-thirds the weight of aluminum and about one-fourth that of steel. Magnesium alloys have excellent machining properties; a good surface finish and high dimensional accuracy may be obtained with heavy cuts and at high cutting speeds with low expenditure of power. Magnesium alloys are fabricated by sand casting, permanent mold casting, die casting, extrusion, rolling, drawing, spinning, or forging. They may be joined by torch welding, arc welding, or spot welding.

USES

Although magnesium had served effectively as a substitute for scarcer metals during World Wars I and II, it began in 1946 to receive consideration as a structural metal for its own qualities. During World War II the major use was for military purposes; and for about 3 years, over one-third of the wartime output of the United States went into incendiaries, which were totally expendable. As a consequence of the loss of the principal wartime outlets, consumption of primary magnesium in 1946 dropped 78 percent below that in 1945, and the use pattern was altered tremendously. Although total consumption decreased, newer and wider ranges of peacetime utilization of the metal were noted in 1946, when its use for wrought products (sheet and extrusions) rose 16 percent above 1945. This reflected the value of research completed by Government and industry during the war years upon alloying and working of the metal. The figures in table 3 reflect the trends of uses of magnesium during 1944–53 and indicate the development and use of wrought products compared with castings.

The use of magnesium for cathodic protection of iron and steel, which had consumed less than 500 tons in previous years, increased to 1,937 tons in 1950. By 1953 this use had expanded to over 5,000 tons and covered the protection of large industrial boiler-type equipment, hot-water heaters, ships and other exposed marine equipment, and entire pipelines. The use of magnesium as an alloying ingredient in aluminum alloys increased in 1953 to more than five times that used in 1948. A new use for magnesium was realized in 1951 by development of the Kroll process for producing titanium on a commercial scale (see Titanium chapter). Titanium production consumed almost 3,000 tons of magnesium in 1953. Another new use for magnesium is in the production of zirconium sponge by the Kroll process (see Zirconium–Hafnium chapter).

In 1949 magnesium was used more widely than ever before in the production of movable civilian products, such as portable tools, material-handling equipment, machinery, and vehicle parts. By 1950 the metal was being widely used for such fabricated items as paper-mill rolls, handtrucks, dockboards, and gangplanks; and magnesium die castings were well established in calculators and business machines. By 1951 magnesium was used in automobiles at the rate of more than 1,000 tons a year, and in 1953 the commercial highway-transportation field constituted one of the largest and fastest growing markets for magnesium. The use of entire magnesium truckbodies made it possible to increase individual loads by several tons. In 1953 it was estimated that 80 percent of all castings going into aircraft was magnesium.

TABLE 3.—*Domestic consumption of primary magnesium (ingot equivalent and magnesium content of magnesium-base alloys) by uses, 1944–53*

[Short tons

Product	1944	1945 [1]	1946	1947	1948	1949	1950	1951	1952	1953
Structural products:										
Castings	105, 119	27, 515	1, 299	1, 083	2, 155	3, 259	3, 905	11, 819	18, 405	17, 813
Wrought products:										
Sheet	1, 543	1, 517	1, 990	1, 053	1, 261	2, 155	3, 357	5, 761	5, 569	7, 103
Extrusions	4, 784	2, 452	2, 689	1, 619	2, 529	3, 364	3, 400	5, 241	3, 756	6, 481
Forgings	344	157	99	105	103	200	104	735	12	24
Total wrought products	6, 671	4, 126	4, 778	2, 777	3, 893	5, 719	6, 861	11, 737	9, 337	13, 608
Total structural products	111, 790	31, 641	6, 077	3, 860	6, 048	8, 978	10, 766	23, 556	27, 742	31, 421
Other products:										
Powder	9, 080	4, 769	192	9	------	------	56	482	1, 553	1, 219
Aluminum alloys	6, 868	5, 589	2, 391	1, 935	2, 171	1, 759	3, 722	5, 994	8, 598	10, 347
Other alloys	12	24	41	40	43	39	255	401	960	418
Scavenger and deoxidizer	159	228	248	427	418	404	473	1, 332	1, 229	423
Chemical	156	182	150	266	407	224	373	447	566	363
Cathodic protection	------	1, 554	774	94	385	235	1, 937	2, 364	2, 100	2, 539
Other [2]	4, 633	------	------	238	226	308	469	1, 134	1, 099	3, 510
Total other products	20, 908	12, 346	3, 796	3, 009	3, 650	2, 969	7, 285	12, 154	16, 105	18, 819
Grand total	132, 698	43, 987	9, 873	6, 869	9, 698	11, 947	18, 051	35, 710	43, 847	50, 240

[1] Figures are incomplete owing to lack of returns from a number of wartime companies whose operations terminated during the war.

[2] Includes primary metal incorporated in making secondary alloy.

The fabrication capacity of the magnesium industry was reported to have doubled during the 2-year period, 1949–50. The requirements for magnesium sheet, particularly for the defense aircraft program, encouraged establishment of a rolling mill in New Kensington, Pa., in 1950 which increased rolling-mill capacity to 200 tons a month. In 1952 a mill was started at Detroit, Mich., with a rolling capacity of 100 tons per month. By the close of 1953 a still greater increase had been effected. The first high-speed, high-production rolling mill for magnesium began operation in November 1953 at Madison, Ill., with production facilities for other wrought products having the following rated capacity:

Rolling mill	8,000 tons per year of average 0.032 inch gage.
Extrusion mill	6,000 tons per year.
Alloy plant	18,000 tons magnesium remelt ingot per year.

Hot-rolled magnesium plate 6 feet wide and in lengths up to 60 feet was produced from 2,000-pound rolling ingots on this 84-inch reversing breakdown coil mill (*1*). All previous production of magnesium sheet and plate had been on hand mills utilizing rolling ingots weighing not more than 350 pounds.

SECONDARY SOURCES AND RECOVERY

Secondary metals are defined as metals or alloys recovered from scrap and residues. The term "secondary" applies only to the source of the metal and has no relation to the type of product recovered, either as to quality, degree of purity, or physical characteristics (*37*).

"Old" scrap is defined as metal contained in articles that have been discarded because of wear, damage, or obsolescence, usually after they have served a useful purpose. Typical examples of old magnesium scrap are aircraft wreckage and lithographers' plates.

"New" scrap is generated during the manufacture of articles for ultimate consumption. Typical examples are clippings, turnings, borings, skimmings, slags, drosses, and defective articles rejected by the manufacturer or returned to him unused.

Recovery of secondary magnesium from scrap did not come into importance in the United States until after World War II began. By the end of the war nine companies were engaged in secondary recovery from scrap.

Table 4, covering 1944–53, shows the recovery of secondary magnesium and its relative importance in the industry during that period.

Secondary magnesium is used in the forms listed below. It has been found particularly suitable for sacrificial purposes.

Magnesium-base alloy ingot.
Magnesium-alloy castings.
Magnesium-alloy wrought products.
Component in aluminum alloys.
Component in zinc and other alloys.
As an ingredient in chemicals.

TABLE 4.—*Recovery of secondary magnesium, 1944-53*

[Short tons]

Year	Secondary magnesium recovered			Percent of magnesium consumed
	From new scrap	From old scrap	Total	
1944	14, 081	104	14, 185	9
1945	8, 422	825	9, 247	17
1946	3, 907	1, 210	5, 117	34
1947	4, 907	4, 596	9, 503	58
1948	3, 365	4, 188	7, 553	43
1949	3, 023	2, 939	5, 962	33
1950	2, 770	4, 970	7, 740	30
1951	5, 400	3, 900	9, 300	20
1952	4, 240	7, 237	11, 477	21
1953	5, 892	6, 038	11, 930	19

Magnesium-base alloy scrap is reconverted into ingot and fabricated articles with characteristics and properties identical to those produced from virgin materials. Secondary magnesium ingot sells for virtually the same price as primary ingot. The characteristics of magnesium alloys and practice that make their repeated usage possible and economical are:

1. The variety of composition and number of alloys is quite narrow, in that the majority of both cast and wrought alloys are combinations of magnesium, aluminum, manganese, and zinc.
2. The alloys do not ordinarily take up metallic impurities in the melting and casting operations.
3. The flux-treating or refining practice invariably used for magnesium melting is adequate to remove surface contamination, dross, and other nonmetallic as well as some metallic impurities.

Magnesium scrap should be stored indoors, preferably in a room that has a relatively even temperature, to prevent condensation of moisture during periods of high humidity. Closed metal containers are preferable, but waterproofed paper bags may be used. Metal containers should be used for collecting scrap within the plant; then, if necessary, the scrap can be transferred to paper bags for separate storage or shipping.

Under no conditions should water-wet magnesium scrap be stored, shipped, or melted, since hydrogen may be evolved that would lead to a spontaneous explosion.

In the accumulation, storage, and processing of magnesium scrap, proper care must be exercised to prevent its contamination with other metals, such as aluminum, brass, copper, and bronze. Failure to prevent this contamination will greatly lower the value of the scrap and may make it entirely worthless.

Large quantities of magnesium scrap were generated during World War II at a time when the maximum tonnage of usable metal was needed. Industry developed satisfactory methods for efficient recovery of heavy scrap but did not have enough time to study methods of recovering or disposing of such accumulations of light scrap as sawings and dust or for recovering metal from melting- and refining-pot drosses. Some of the most disastrous fires and explosions experienced in the fabricators' plants during World War II were attributed to accumulations of these dusts and drosses. Work was begun during World War II by the Bureau of Mines in examination and treatment of magnesium scrap. In March 1946 the Bureau published Report of Investigations 3860, Examination and Treatment of Industrial Magnesium Foundry Wastes. The conclusions contained in this report are the basis for the handling of dust and drosses which is now being practiced by the magnesium industry.

The recommendation set out by the Bureau—that fine magnesium-alloy grindings should not be processed but should be destroyed immediately by burning—became common practice in fabricating plants. The quantities of magnesium grindings generated at even the largest foundries do not warrant the risk of processing water-wet grindings, and the products made would not pay for the required expense in processing.

SOURCES AND ADEQUACY OF STATISTICAL INFORMATION

The Bureau of Mines gathers statistical information covering the production and consumption of magnesium in the United States directly from industry and has received excellent cooperation, both from the standpoint of prompt response and complete information as requested.

Data covering shipments of magnesium cast and wrought products are collected and published by the Bureau of the Census, United States Department of Commerce.

Export-import figures covering magnesium are gathered by the Bureau of the Census.

Information concerning production of magnesium by foreign countries is accumulated by the Bureau of Mines from statistics published by the countries, by contact with foreign countries through United States embassies, and by any information it may receive through periodicals or special publications of speeches and articles.

FOREIGN TRADE AND WORLD PRODUCTION

Table 5 shows United States foreign trade in magnesium for 1942–53. The tonnage of imported magnesium scrap was a result of trade agreements with concerns in some countries where old scrap was sold or traded for shipment to the United States for new products, rather than a particular need for the scrap metal in the United States.

World production of primary magnesium during 1944–53, shown in table 6, indicates the strong position attained in the United States.

STRATEGIC CONSIDERATIONS

Under the provisions of the Strategic and Critical Materials Stockpiling Act (Pub. 520, 79th Cong., 2d sess., as amended) and the Defense Production Act (Pub. 774, 81st Cong.), a National Stockpile of primary magnesium ingot was accumulated, all of which was produced by the Government-owned plants in

TABLE 5.—*Magnesium imported for consumption and exported from the United States* [1]

[Short tons]

Year	Exports			Imports			Total United States consumption	Percent of United States consumption
	Metal in primary form	Fabricated (Mg content)	Total	Metallic and scrap	Fabricated (Mg content)	Total		
1942	4, 045	171	4, 216	(2)	(2)	(2)	22, 788	----------
1943	35, 631	4, 799	40, 430	(2)	(2)	(2)	68, 372	----------
1944	21, 001	3, 197	24, 198	27	33	60	132, 698	. 004
1945	518	860	1, 378	54	3	57	43, 987	. 01
1946	207	71	278	241	----------	241	9, 873	2.
1947	315	40	355	201	1	202	6, 869	3.
1948	274	170	444	678	----------	678	9, 698	7.
1949	432	276	708	2, 560	----------	2, 560	11, 947	21.
1950	586	322	908	843	25	868	18, 051	5.
1951	575	186	761	3, 871	108	3, 979	35, 710	11. 11
1952	[3] 1, 066	[3] 97	[3] 1, 163	252	48	300	43, 847	. 68
1953	[3] 2, 722	[3] 227	[3] 2, 949	2, 443	8	2, 458	50, 240	4. 89

[1] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

[2] Less than 0.5 ton.

[3] Due to changes in items included in each classification, data are not strictly comparable with earlier years.

TABLE 6.—*World production of magnesium metal, by countries, 1944–48 (average) and 1949–53* [1]

[Short tons]

(Compiled by Helen L. Hunt)

Country	1944–48 (average)	1949	1950	1951	1952	1953
Canada	1, 856	(2)	1, 764	4, 409	[3] 5, 511	[3] 6, 614
China, Manchuria	143	(2)	(2)	(2)	(2)	(2)
France	722	545	450	965	1, 200	1, 098
Germany:						
East [3]	6, 600	1, 100	1, 100	1, 100	1, 100	1, 100
West	[3] 1, 235	----------	----------	----------	----------	----------
Italy	600	----------	135	645	1, 075	1, 595
Japan	885	----------	----------	----------	----------	----------
Korea	582	----------	----------	----------	----------	----------
Norway	[3] 445	----------	----------	135	[3] 1, 435	[3] 4, 955
Switzerland	[3] 375	----------	275	275	335	[3] 275
Taiwan (Formosa)	100	----------	----------	----------	----------	----------
U. S. S. R.[3]	17, 385	21, 000	22, 000	33, 000	44, 000	55, 000
United Kingdom [4]	5, 785	2, 860	3, 300	5, 500	5, 065	5, 940
United States	43, 500	11, 598	15, 726	40, 881	105, 821	93, 075
Total (estimate)	80, 600	37, 500	45, 200	89, 300	168, 600	169, 800

[1] This table incorporates a number of revisions of data published before 1953.

[2] Data not available; estimate by author included in total.

[3] Estimate.

[4] Primary metal and remelt alloys.

addition to their output of metal for defense orders during the Korean emergency. To insure the quality of all magnesium ingot sent to stockpile for outdoor storage, the following chemical requirements were set by the Government:

	Percent by weight		
	Grade A		*Grade B*
Total of aluminum, copper, iron, manganese, nickel and silicon	0.20	maximum	0.20
Copper	.02	do	.02
Iron	.05	do	.005
Nickel	.001	do	.001
Magnesium	99.80	minimum	99.80

It was believed in 1953 that the eight magnesium plants in the United States had adequate capacity for producing requirements for primary magnesium, even for total mobilization. In 1954, however, the increased consumption and potential use of magnesium resulted in the industry's estimating that the available productive capacity would not be adequate.

Industry concentration is a matter to be considered. The 2 largest electrolytic plants, representing 51 percent of the total production capacity in the United States, are on Galveston Bay; and the 2 largest silicothermic plants, representing 23 percent of the total production capacity in the United States, are very near the west coast. These four largest plants might become relatively easy targets for an invading enemy.

Each of the three electrolytic plants and the silicothermic plant at Spokane, Wash., utilize 10 kw.-hr. of electricity to produce 1 pound of magnesium, as against 4½ kw.-hr. per pound required (for production of ferrosilicon) by each of the 4 remaining silicothermic plants designed to use fuel oil or gas to heat the furnaces. During World Wars I and II electricity was one of the first resources to become scarce.

PRICES AND COSTS

Before World War I magnesium was quoted at about $1.65 a pound for the imported (German) metal. After the foreign supply was cut off, prices rose to $5 or even $6 a pound for sticks, and powdered material sold at $6.50 a pound; prices remained at about these rates through 1915.

During 1916 the price dropped, and at the beginning of 1917, 99-percent pure bar magnesium was selling for $2.50 to $3 a pound. At the end of 1917 primary magnesium in bars was selling for $1.85 to $2 a pound, according to the quantity bought.

The fluctuations in prices of commercially pure magnesium ingot from 1918 to 1953 are shown in table 7.

The Government fixed the price of primary magnesium at 22.5 cents per pound on April 28, 1942, and lowered it to 20.5 cents per pound on January 1, 1943. Primary alloy ingot ranged in prices from 23 cents to 25 cents per pound in 1943. On August 30, 1945, price controls were abolished.

Price controls were reestablished on September 30, 1950, and the price of primary magnesium was fixed at 24.5 cents per pound. The increase to 27 cents per pound on March 9, 1953, was the result of increased production costs.

TABLE 7.—*Primary-magnesium-ingot prices, 1918–53*

Year	Average price per pound	Year	Average price per pound
1918	$1.81	1936	$0.30
1919	1.83	1937	.30
1920	1.60	1938	.30
1921	1.30	1939	.28
1922	1.60	1940	.27
1923	1.25	1941	.225
1924	1.07	1942	.225
1925	.86	1943	.205
1926	.80	1944	.205
1927	.68	1945	.205
1928	.82	1946	.205
1929	.56	1947	.205
1930	.48	1948	.205
1931	.34	1949	.205
1932	.29	1950	.22
1933	.28	1951	.245
1934	.26	1952	.245
1935	.30	1953	.266

Price changes since World War II: June 1, 1950—21.5 cents; July 19, 1950—22.5 cents; Sept. 30, 1950—24.5 cents; Mar. 9, 1953—27.0 cents.

The above market quotations on primary magnesium do not reflect the average cost for production in the Government-owned plants during World War II and the Korean emergency. The average cost was reported at 26.14 cents per pound during World War II, exclusive of interest on investment and depreciation charges (*44*). The average cost for production in the reactivated plants was 42.387 cents per pound, which includes all manufacturing costs and manufacturers' fees but does not include cost of rehabilitating the plants, closing the plants, and preparing them for standby after closing. This cost reflects increases in cost of labor, materials, supplies, and transportation after World War II.

TARIFF

Under the Tariff Act of 1913 magnesium and its alloys were dutiable at 25 percent ad valorem and manufactures of magnesium at 20 percent. Under the Tariff Act of 1922 a duty of 40 cents a pound was imposed on magnesium metal (including scrap), and the rates for alloys and manufactures were changed to 40 cents a pound on the magnesium content plus 20 percent ad valorem. These rates were reenacted in the Tariff Act of 1930.

When the tariff was changed in 1922 the domestic industry had just begun to adjust itself to peacetime production. The new duty was equivalent to an ad valorem rate of about 60 percent on the foreign value of magnesium metal imported in 1923—the first full year in which the new rate was in effect. In the same year the ad valorem equivalent was about 56 percent for alloys, 50 percent for powder, and 37 percent for magnesium manufactures (*47*).

By 1948 the tariff rates had been lowered to 20 cents per pound for magnesium metal; 20 cents per pound for magnesium content in scrap; and 20 cents per pound on magnesium content plus 10 percent ad valorem for alloys. These tariffs were in effect at the close of 1953, except that for magnesium scrap; duty on metallic scrap was suspended on October 1, 1950.

RESEARCH BY INDUSTRY, INSTITUTIONS AND GOVERNMENT

RESEARCH ON PRODUCTION

Work completed by the close of 1953 indicated that the electrolytic process (described under Technology) developed by The Dow Chemical Co. was the most feasible in the United States from the standpoint of cost.

During July 1941 a study was initiated at the Bureau of Mines Boulder City Experiment Station, which placed major emphasis upon development of a process that required minimum reliance upon the sale or utilization of byproducts to meet economic competition from processes already in operation. Based on information from laboratory experiments, a 10,000-ampere electrolytic cell was completed; but it became necessary to discontinue this project in the interest of other more urgent work, and the cell was not operated but was stored at the Boulder City Station. Preliminary tests showed that many of the difficulties met would require a long period of continuous operation before they became serious enough to affect operation of the cell (*31*).

Diamond Alkali Co. adopted the Dow process at the Government Painesville, Ohio, plant, utilizing dolomite as the source of magnesium. $CaCl_2$ was used to produce $MgCl_2$ by an exchange reaction with calcined slaked dolomite, $Mg(OH)_2+Ca(OH)_2$ in the presence of CO_1 (*2*). The surplus chlorine generated by this process was used to produce calcium and sodium hypochlorites, which were in strong demand during World War II.

Mathieson Alkali Works, Inc., evolved a process similar to that employed by Diamond Alkali Co., whereby ammonium chloride was treated with slaked, calcined dolomite and CO_2. The Mathieson cell was designed to produce a concentrated Cl_2, whereas in the Dow cell the chlorine is heavily diluted with HCl and air (*53*). Mathieson Alkali claimed that its cell had advantages over the Dow cell, and the 27,000-ton plant built with Government funds at Lake Charles, La., was equipped with the Mathieson cell. After operating a little more than a year the plant was closed. It had attained only 30-percent capacity.

Dow cells were installed by International Minerals & Chemical Corp., at Austin, Tex., and Carlsbad, N. Mex. (*50*). The raw material was a brine containing $MgSO_4 \cdot K_2SO_4$ obtained from the company potash plant.

The electrolytic process adopted at the Government's Henderson, Nev., plant employed a cell similar to that developed by I. G. Farbenindustrie in Germany. The I. G. Farbenindustrie cell differs from the Dow cell in that it is refractory lined and is not equipped for external heating, and it utilizes anhydrous magnesium chloride for cell feed instead of magnesium chloride. The cells at the Henderson plant did not operate at as high energy efficiency as the Dow cell, but the plant reached and exceeded its rated annual capacity before it was closed.

Shortage of electric power during World War II forced the Government to explore the possibilities of reducing magnesium by other methods than the electrolytic cell process. By 1940 it became necessary to expedite completion of plant construction for immediate production of metal. One of the first methods to be explored was the carbothermic process developed by Dr. F. J. Hansgirg, whose original plant was in Austria (*19*). This process is briefly described as follows:

Carbothermic Process.—A mixture of equal parts of carbon and magnesium oxide is briquetted, and the briquets are heated to approximately 2,000° C. in an electric-arc furnace to produce magnesium metal as a vapor and carbon monoxide. The reaction products are cooled very rapidly to 200° C. or less to prevent reoxidation of the magnesium metal. Hydrogen was used by Hansgirg as a shock-cooling medium. Permanente Metals Corp. (one of the Kaiser interests) bought Dr. Hansgirg's United States patents covering the carbothermal repuction process (*20*) and built a 12,000-ton dlant at Permanente, Calif., installing equipment similar to that in the plant in Austria designed by Dr. Hansgirg. A continuous sublimation process was not achieved in the Permanente plant before it was shut down at the close of World War II.

In 1945 the Bureau of Mines published Reports of Investigations 3806, Studies in Redistillation of Carbothermic Magnesium, and 3823, Construction and Operation of the Dearborn Magnesium Plant. The pilot plant at Dearborn burned before all research had been completed. However, one factor that had been established was that nitrogen could be used instead of hydrogen for introduction into the furnace around the electrode in the shock-cooling process. A continuous sublimation process was achieved at the Dearborn pilot plant, but reduction, shock-cooling, and subsequent separations could be carried out satisfactorily only in specially designed equipment. Because of the explosive nature of the magnesium dust at high temperature, extreme precaution was necessary to maintain the system in an atmosphere that was nonoxidizing.

Silicothermic Reduction.—The most efficient of the thermic processes of producing magnesium employed during World War II is often referred to as the ferrosilicon or Pidgeon process (described under Technology above), named for Dr. L. M. Pidgeon, who had placed in operation in Canada in 1938, a pilot plant in which magnesium oxide was reduced with ferrosilicon (*38*).

A variation of this process employed by the Electro Metallurgical Co. at the Government-owned plant at Spokane, Wash., during World War II used an internally heated electric vacuum furnace, developed by Glen D. Bagley (*3*).

Research has continued in production techniques by the silicothermic method of reduction to increase efficiency and lower the cost of production. In the Government-owned plant at Canaan, Conn., the 20 retorts in 1 of the furnaces were equipped with refractory linings, and the test was begun in November 1951. Reports furnished before closing of the plant in 1953 stated that the test revealed that approximately 4 days per month was saved on each refractory-lined retort over the time required to clean the scale from the standard metal retorts and to reshape them by introducing compressed air. This was reported to average an increased volume of production approximating 10 percent.

Research was continued in the Bureau of Mines laboratory at Albany, Oreg., to June 30, 1954, to investigate the factors that might contribute to more efficient operation of the thermic processes for the production of magnesium.

Serpentine Process.—This process of reduction was submitted to the Government in April 1942 by Idaho-Maryland Mines Corp. of Grass Valley, Calif. It reported that a small pilot-plant operation indicated the merits of a process employing an electrolytic cell similar to the Dow cell and utilizing serpentine as a source of magnesium (*41*). This process has not been used in commercial operation.

RESEARCH ON ALLOYS

In early industrial applications, aluminum was the chief alloying agent added to magnesium to increase its strength. Later zinc was added to these alloys to arrest corrosion during service. The aluminum-zinc-magnesium alloy obtained by solution heat treating and aging, designated as AZ91C, is said to produce foundry porosity-free castings nearly as strong and hard as the best of the light alloys produced.

Important developments in magnesium alloys were made during 1949 and 1950 in the use of rare-earth metals and zirconium as alloying constituents. Zirconium was added to magnesium-zinc alloys for grain refining (*30*, *34*). Rare earths in proportions up to 3 percent are added to magnesium alloys to improve the mechanical properties (creep strength) at elevated temperatures.

During 1952 investigations were reported on improvement of thorium-magnesium alloys, showing exceptional strength up to 600° F. and a degree of usefulness at 700° F., a temperature at which rare-earth alloys were not considered for use. This offered a solution of the need for alloys with improved creep strength in service above 400° F. By 1953 development work had progressed toward commercial applications in jet engines of castings from the magnesium alloys containing 2 to 3 percent thorium (*42*).

A new magnesium alloy containing beryllium, designated as AZ91B, was placed on the market in 1953 as a die-casting material. It showed lower melt loss and increased efficiency over former magnesium alloys and proved to be competitive with aluminum (*36*).

One of the chief disadvantages of magnesium alloys has been their poor resistance to corrosion. Magnesium does not form a tight, protective, outside coating. Corrosion of magnesium is accelerated by small quantities of such impurities as iron, nickel, and copper when they exceed certain tolerance limits. The addition of manganese improves the corrosion resistance of magnesium alloys by counteracting the effect of iron.

Research was continued throughout 1953 by industry and by the Bureau of Mines, the Army, Navy and Air Force, to develop fundamental data concerning improved mechanical properties, mechanism of hardening, and aging characteristics of magnesium-lithium-base alloys, which were at that time considered the lightest and most ductile of the magnesium-base alloys. Additions of aluminum and zinc as alloying constituents appeared to offer the best combination of properties.

Progress has been made in protective surface treatment and anodizing magnesium alloys (*13*).

OUTLOOK

It is difficult to set up a tabular forecast of increases in consumption of magnesium from year to year. In 1951 the Magnesium Association forecast that a total annual consumption of 100,000 to 150,000 tons by 1960 might be considered attainable if the industry were allowed to grow normally. Based upon the progress experienced during the years 1944–53 in reducing costs of fabrication, especially in extrusion, die casting, and rolling, these estimates seemed reasonable.

Consumption reached 50,000 tons in 1953, and enough progress had been made in improving both the performance of magnesium and the techniques of its fabrication to warrant the hope that it may have a greatly extended market by 1975. Magnesium sheet can be expected to share the market formerly held entirely by aluminum sheet, as in aircraft applications, household utensils, hardware, and other consumer goods. Requirements for magnesium in highway transportation, which have advanced steadily since the close of World War II, may be expected to continue to increase.

In 1953 the magnesium industry, in developing guidance for its expansion program, adopted the following forecast for volume of consumption: 1952 (actual consumption), 44,000 tons; 1955, 110,000 tons; and 1960, 125,000 tons.

PROBLEMS

The application and use of magnesium have been limited by the lack of development of alloys and alloying techniques and treatment methods that would make this metal more suitable in applications where more critical materials have been used. The development of new alloying techniques and new alloys with improved physical properties should make it possible for magnesium to achieve a place in the metals industry reflecting its abundance in nature.

The promotion of a competitive magnesium-producing industry is a major problem. Present economic production depends upon a patented electrolytic process. The development of improved processes or equipment that would reduce production costs might encourage new private interests to enter the field of magnesium production. The thermal processes of production appear encouraging from the standpoint of abundance of raw materials and relatively small capital investments, but operation of thermic plants has thus far been too costly in the United States to place them in competition with the electrolytic plants. Members of the industry suggested, in 1953, that, in view of the importance of magnesium to the national security, the Government should seriously consider sponsoring additional private magnesium-production capacity by granting aid similar to that granted for aluminum expansions started during the Korean emergency.

MAGNESIUM COMPOUNDS

BACKGROUND

SOURCES

There are about 60 magnesium-bearing minerals, but magnesite, brucite, dolomite, sea water, sea-water bitterns, and well brines account for almost all of the magnesium-compounds production (*21*). Minor quantities of olivine and serpentine are used.

Magnesite ($MgCO_3$).—Theoretically, magnesite contains 47.8 percent MgO and 52.2 percent CO_2. Magnesite occurs in crystalline and cryptocrystalline form. Deposits of cryptocrystalline magnesite ordinarily are much smaller and less widespread in occurrence than deposits of crystalline magnesite but usually are purer. Virtually all of the cryptocrystalline variety mined in the United States has come from California.

The principal domestic deposits of crystalline magnesite occur near Chewelah, Stevens County, Wash., and Gabbs, Nye County, Nev.

Brucite ($Mg(OH)_2$).—Theoretically, brucite contains 69.1 percent MgO and 30.9 percent H_2O. The only known deposit of brucite of commercial magnitude in the United States adjoins the magnesite deposit at Gabbs, Nev. (*10, 24*).

Dolomite ($CaCO_3 \cdot MgCO_3$).—Dolomite is technically described as a double salt and often referred to as calcium carbonate. In practice the term includes high-magnesian limestones. The theoretical analysis of dolomite is 21.9 percent MgO, 30.4 percent CaO, and 47.7 percent CO_2. Domestic dolomite deposits are widespread.

Salt waters.—The average magnesium content of sea water is 0.13 percent. Sea-water bitterns—the liquor residue from extraction of sodium chloride (salt)—contains 6.0 to 8.7 percent magnesium chloride ($MgCl_2$) and 4.2 to 6.1 percent magnesium sulfate ($MgSO_4$). Michigan well brines contain about 10 percent magnesium chloride.

Olivine ($Mg_2Fe_2SiO_4$) *and serpentine* ($H_4Mg_3Si_2O_9$).—These magnesium silicates occur in huge deposits in the United States and elsewhere. They have not been utilized commercially to produce magnesia but are used in refractories to a limited extent and are potential sources of magnesia (*18*).

HISTORY

Natural magnesium carbonate was discovered in Moravia and named "talcum carbonatum" by C. F. Ludwig in 1803. Previously, in 1795, J. C. Delametherie had applied the term "magnesite" to a series of magnesium salts (carbonate, sulfate, nitrate, and chloride), and A. Brongniart had applied the same term to a group of minerals, including the carbonate and certain silicates. The term "magnesite" was restricted to the natural carbonate by D. L. G. Karsten in 1808, and gradually the definition was accepted (*45*). Even today foreign export and import statistics often list magnesium salts as magnesite; and the terms caustic-calcined and refractory (dead-burned) magnesite, magnesite brick, and chrome-magnesite brick are used by industry to refer to the products, whether they are derived from magnesite, brucite, dolomite, well brines, or sea water.

Although the refractory nature of magnesium oxide had been recognized before discovery of extensive magnesite deposits in Styria in 1884, little or no commercial application seems to have been made of this knowledge until then. Magnesia refractories began to be used extensively in the steel industry in 1900. The rapid increase in the production of basic open-hearth steel resulted in a corresponding increase in demand for magnesite.

TECHNOLOGY

Caustic-calcined magnesia is produced from magnesite by calcining it at temperatures below 1,560° C. to retain 2 to 10 percent CO_2. Crude ores are calcined at temperatures ranging from 1,560° to 1,760° C. to produce refractory magnesia. The product resulting from calcining at temperatures below 1,700° C. is divided into two classes: (1) Brick grade, containing 83 to 90 percent magnesia; and (2) maintenance grade, containing 60 to 82 percent magnesia. The product obtained by calcining a purer grade of magnesite ore, or sea-water magnesia, at a temperature above 1,700° C. is termed periclase. All refractory magnesia obtained from sea water, sea-water bitterns, or well brines is of brick grade, but the addition of such materials as iron oxide or silica to the material before calcining permits the manufacture of refractory magnesia of various specifications.

The production of magnesia from sea water and dolomite is described briefly as follows: When the dolomite is taken from the quarry it is coarsely crushed, washed, and beneficiated to maintain even quality. It is again crushed before it is screened and calcined at temperatures ranging from 1,100° to 1,200° C., which drives off the carbon dioxide content and produces dolomitic lime, a combination of calcium and magnesium oxides. The dolomitic lime is reacted with sea water to form magnesium hydroxide. About half of the magnesia comes from the sea water and half from the calcined rock. The sea-water and lime slurry is filtered to remove water. The magnesium hydroxide filter cake is then carried to rotary kilns fired to temperatures that reach as high as 1,850° C. The dried filter cake, containing approximately 97 percent MgO, is then ready to be combined with other elements such as forsterite, refractory-grade chromite, or silica to produce refractories as required by the consuming industries.

PRODUCTION

Before World War I Austria produced most of the magnesite used commercially, with small quantities reported from Greece (*51*). Deposits in California furnished less than 4 percent of United States requirements.

The output of California had grown from 600 tons in 1887 to an average of about 10,000 tons of crude magnesite per year between 1908 and 1914, but almost all of it was consumed on the Pacific coast in paper manufacturing. The cutting off of foreign supplies by World War I made it necessary for eastern magnesia-refractories consumers to seek domestic sources and to offer much higher prices than those formerly prevailing. Many new deposits of magnesite were developed, old ones were reopened, and new and larger calcining plants were constructed. In 1917 California production reached a peak of over 200,000 tons valued at almost $2,000,000 (*6*). Large deposits of crystalline magnesite were discovered in Stevens County, Wash., in 1916 and in 1917 one-third of the domestic production was taken from those deposits. In 1918 Washington produced over two-thirds of the domestic output.

Between World Wars I and II about 70 percent of the refractory magnesia used in the

United States was produced from domestic crude magnesite. The imported magnesia, comprising about 30 percent of the consumption in the United States, was used mostly in manufacturing brick and shapes, whereas most of the domestic magnesite was used in producing maintenance-grade magnesia. Before the outbreak of World War II about 75 percent of the crude magnesite produced in the United States came from Washington and most of the remainder from California. Brucite mining, on a commercial basis, was begun in 1938 in Nevada, and it was used in special basic refractories. With the beginning of World War II refractory magnesia imports from Austria again were cut off. For a time thereafter, considerable quantities of refractory magnesia came from Manchuria, but these imports ceased with the attack on Pearl Harbor.

By 1942 there was a marked increase in demand for refractory magnesia, especially from the steel industry. This led to expansion in the United States of production of refractory magnesia, as well as brick-grade material. Expansion took place both by investment of private capital and on a somewhat smaller scale at Government expense. The Northwest Magnesite Co., the only producer in Washington, greatly enlarged its mining and calcining facilities, reworked old dump piles, and built a flotation plant to make possible the production of brick-grade magnesia. Magnesite mining and calcining facilities were expanded in California. The extensive Nevada deposits were brought into production in 1941 to supply raw material for both magnesium and refractories. The capacity of the Westvaco Chlorine Products Corp. plant at Newark, Calif., for the manufacture of periclase (MgO) from sea-water bitterns was expanded; new plants were built at Moss Landing, Calif. and at Cape May, N. J., to produce magnesia from sea water and calcined dolomite; magnesia-production facilities were added to the plant at Freeport, Tex., built to obtain magnesium chloride from sea water for reduction of magnesium; and plants were built in Michigan to utilize well brines. By 1952 magnesia-production facilities had been increased even more, with plants in Ohio and West Virginia producing refractory magnesia solely from dolomite.

By the end of World War II California magnesite deposits were virtually exhausted, and since then the trend has been toward an increase in the proportion of magnesia derived from processes utilizing sea water and well brines as the raw-material source. In 1953, 87 percent of the caustic-calcined magnesia and 44 percent of the refractory magnesia was produced from these sources compared with 29 and 21 percent, respectively, in 1944. During the decade, 1932–41, before World War II, sales of domestic magnesias increased from slightly more than 18,000 short tons per year to over 230,000 tons, as shown in table 8. Average annual sales during this period were 91,673 tons compared with average annual sales of 377,399 tons per year for the 12-year period from 1942–53. The figures show that the domestic industry could expand to meet the growing demand for magnesia refractories by the iron and steel and other industries. Table 9 shows the development of the industry during 1942–53.

TABLE 8.—*Magnesia sold by producers in the United States, 1932–41, by kinds*

Year	Magnesia				Total	
	Caustic-calcined		Refractory			
	Short tons	Value	Short tons	Value	Short tons	Value
1932	3, 374	$103, 196	14, 836	$308, 327	18, 210	$411, 523
1933	8, 141	249, 115	43, 613	774, 875	51, 754	1, 023, 990
1934	7, 528	222, 415	38, 535	670, 343	46, 063	892, 758
1935	6, 049	170, 326	72, 438	1, 361, 949	78, 487	1, 532, 275
1936	7, 998	221, 410	89, 979	1, 713, 527	97, 977	1, 934, 937
1937	10, 031	311, 326	83, 204	1, 598, 336	93, 235	1, 909, 662
1938	7, 400	228, 498	38, 738	730, 978	46, 138	959, 476
1939	10, 157	310, 102	86, 077	1, 699, 723	96, 234	2, 009, 825
1940	16, 261	512, 607	140, 668	2, 802, 537	156, 929	3, 315, 144
1941	30, 225	1, 052, 077	201, 481	5, 052, 879	231, 706	6, 104, 956
Total	107, 164	3, 381, 072	809, 569	16, 713, 474	916, 733	20, 094, 546
Yearly average	10, 716	338, 107	80, 957	1, 671, 347	91, 673	2, 009, 455

TABLE 9.—*Magnesia sold or used by producers in the United States, 1942–53, by kinds and sources*

Magnesia	From magnesite, brucite, and dolomite		From well brines, raw sea water, and sea water bitterns[1]		Total	
	Short tons	Value	Short tons	Value	Short tons	Value
1942						
Caustic-calcined	(2)	(2)	(2)	(2)	41, 889	$2, 028, 126
Refractory	(2)	(2)	(2)	(2)	273, 661	7, 823, 963
Total	(2)	(2)	(2)	(2)	315, 550	9, 852, 089
1943						
Caustic-calcined	(2)	(2)	(2)	(2)	191, 792	11, 497, 505
Refractory	(2)	(2)	(2)	(2)	301, 382	9, 341, 183
Total	(2)	(2)	(2)	(2)	493, 174	20, 838, 688
1944						
Caustic-calcined	99, 063	$4, 366, 474	40, 180	$2, 115, 489	139, 243	6, 481, 963
Refractory	220, 583	5, 943, 480	57, 907	2, 482, 569	278, 490	8, 426, 049
Total	319, 646	10, 309, 954	98, 087	4, 598, 058	417, 733	14, 908, 012
1945						
Caustic-calcined	16, 098	1, 122, 119	27, 172	1, 381, 425	43, 270	2, 503, 544
Refractory	193, 693	5, 068, 987	61, 301	2, 345, 231	254, 994	7, 414, 218
Total	209, 791	6, 191, 106	88, 473	3, 726, 656	298, 264	9, 917, 762
1946						
Caustic-calcined	16, 269	1, 310, 584	28, 909	1, 543, 954	45, 178	2, 854, 538
Refractory	174, 121	4, 539, 978	70, 703	2, 691, 891	244, 824	7, 231, 869
Total	190, 390	5, 850, 562	99, 612	4, 235, 845	290, 002	10, 086, 407
1947						
Caustic-calcined	10, 850	1, 005, 920	15, 981	1, 502, 704	26, 831	2, 508, 624
Refractory	209, 581	5, 794, 636	105, 340	4, 332, 949	314, 921	10, 127, 585
Total	220, 431	6, 800, 556	121, 321	5, 835, 653	341, 752	12, 636, 209
1948						
Caustic-calcined	11, 548	996, 713	21, 661	2, 383, 815	33, 209	3, 380, 528
Refractory	214, 628	7, 954, 089	115, 441	5, 490, 498	330, 069	13, 444, 587
Total	226, 176	8, 950, 802	137, 102	7, 874, 313	363, 278	16, 825, 115
1949						
Caustic-calcined	8, 992	831, 674	23, 513	2, 277, 707	32, 505	3, 109, 381
Refractory	175, 364	6, 763, 294	75, 025	3, 714, 562	250, 389	10, 477, 856
Total	184, 356	7, 594, 968	98, 538	5, 992, 269	282, 894	13, 587, 237
1950						
Caustic-calcined	6, 418	740, 369	35, 029	3, 396, 529	41, 447	4, 136, 898
Refractory	232, 283	9, 358, 483	103, 157	5, 557, 371	335, 440	14, 915, 854
Total	238, 701	10, 098, 852	138, 186	8, 953, 900	376, 887	19, 052, 752
1951						
Caustic-calcined	7, 689	889, 624	42, 292	3, 920, 755	49, 981	4, 810, 379
Refractory	295, 243	10, 610, 788	136, 954	7, 789, 343	432, 197	18, 400, 131
Total	302, 932	11, 500, 412	179, 246	11, 710, 098	482, 178	23, 210, 510

See footnotes at end of table, p. 487.

TABLE 9.—*Magnesia sold or used by producers in the United States, 1942–53, by kinds and sources*—Con.

Magnesia	From magnesite, brucite, and dolomite		From well brines, raw sea water, and sea water bitterns [1]		Total	
	Short tons	Value	Short tons	Value	Short tons	Value
1952						
Caustic-calcined	4, 528	$761, 268	33, 527	$3, 008, 198	38, 055	$3, 769, 466
Refractory	232, 766	8, 057, 848	154, 107	9, 197, 989	386, 873	17, 255, 837
Total	237, 294	8, 819, 116	187, 634	12, 206, 187	424, 928	21, 025, 303
1953						
Caustic-calcined	5, 781	799, 625	37, 239	3, 191, 684	43, 020	3, 991, 309
Refractory	221, 869	9, 024, 974	177, 263	10, 035, 822	399, 132	19, 060, 796
Total	227, 650	9, 824, 599	214, 502	13, 227, 506	442, 152	23, 052, 105
Total (12 years)					4, 528, 792	
Yearly average					377, 399	

[1] Magnesia made from a combination of dolomite and sea water is included with that from sea water.

[2] Not available.

In addition to caustic-calcined and refractory magnesia, the following magnesium compounds are produced in the United States: Magnesium chloride; magnesium carbonate; magnesium hydroxide; technical and U. S. P. extra-light, light, and heavy magnesia; magnesium sulfate (epsom salt); and various high-purity magnesium salts. Many of the statistical data on their production must be concealed to avoid disclosure of individual company operations, but table 10 shows the production for the years 1949–53 insofar as confidentiality rules permit.

USES

Bitterns from sea water and deep water wells are the source of magnesium chloride cell feed for all of the electrolytic-magnesium-producing plants in the United States. Since World War II these bitterns have become an important source of magnesia.

Dolomite is the raw material utilized as the source of magnesium in all of the silicothermic magnesium producing plants in the United States (see Part I. Magnesium). Table 11 gives the sales and imports of dead-burned (calcined) dolomite for 1949–53, the bulk of which went to the iron and steel industry. Lump and ground caustic-calcined magnesia imported for consumption during this period is listed in table 12.

Small quantities of magnesite and brucite are used to produce epsom salt. The remainder is calcined to produce caustic-calcined magnesia and refractory magnesia. Refractory magnesia is utilized almost entirely for the manufacture of refractories as grain products or as brick and shapes. Most of these refractories are used in the steel and copper industries, although magnesia brick are utilized in the hot zone of rotary kilns whenever a high-temperature refractory is required. Brick-grade magnesia also is used with chromite for making chromite-magnesia or magnesium-chromite brick, which range in composition from about 75 percent chromite and 25 percent magnesia, to 25 percent chromite and 75 percent magnesia. These brick are serviceable in metallurgical furnaces where resistance to corrosive oxides and high resistance to spalling is required. Chromite-magnesia and magnesia-chromite brick are cheaper than magnesia brick.

In 1953 the following uses and percentages were reported for caustic-calcined magnesia:

	Percent
Oxychloride and oxysulfate cement	41
Rayon	8
Insulation	13
Fertilizer	2
Rubber, filler and catalyst	1
Welding-rod flux	1
Miscellaneous (including chemicals and paper)	34

Technical and U. S. P. magnesia uses and percentages in 1953 were:

	Percent
Rayon	45
Rubber filler and catalyst	29
Refractories	13
Chemicals	10
Medicinal	3

RESERVES

In addition to the limitless quantity of magnesium in sea water, reserves of most of the magnesium minerals are large and widely

TABLE 10.—*Specified magnesium compounds produced, sold, and used by producers in the United States, 1949–53* [1] [2]

Products [1]	Plants	Produced (short tons)	Sold [2]		Used (short tons)
			Short tons	Value	
1949					
Specified magnesias: [3]					
Extra-light and light	5	1,637	1,644	$837,751	------------
Heavy	3	933	949	395,994	(4)
Total	[5] 5	------------	2,593	1,233,745	(4)
Precipitated magnesium carbonate	10	55,925	7,273	924,299	48,641
1950					
Specified magnesias: [3]					
Extra-light and light	5	2,480	2,392	1,214,844	------------
Heavy	3	1,850	1,734	513,586	(4)
Total	[5] 5	------------	4,126	1,728,430	(4)
Precipitated magnesium carbonate	10	54,633	7,389	1,134,499	47,153
1951					
Specified magnesias: [3]					
Extra-light and light	5	2,251	2,221	1,114,037	(4)
Heavy	2	(4)	(4)	(4)	------------
Total	[5] 5	------------	(4)	(4)	(4)
Precipitated magnesium carbonate	10	60,530	8,415	1,117,999	51,987
1952					
Specified magnesias: [3]					
Extra-light and light	5	1,986	2,012	1,100,078	(2)
Heavy	2	(2)	(2)	(2)	------------
Total	[4] 5	(2)	(2)	(2)	(2)
Precipitated magnesium carbonate	7	43,267	5,380	870,003	37,882
1953					
Specified magnesias: [3]					
Extra-light and light	5	2,341	2,303	1,109,848	(2)
Heavy	4	11,434	(2)	(2)	(2)
Total	[4] 6	13,775	(2)	(2)	(2)
Precipitated magnesium carbonate	7	41,034	5,010	745,423	35,768
Magnesium hydroxide, U. S. P. and technical (basis 100 percent $Mg(OH)_2$)	4	[5] 5,975	[5] 4,334	[5] 303,893	(2)

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[1] In addition, magnesium chloride, nitrate, phosphate, acetate, and trisilicate were produced.
[2] Figures withheld to avoid disclosure of individual company operations.
[3] Basis: 100 percent MgO, U. S. P. and technical.
[4] A plant producing more than 1 grade is counted but once in arriving at total.
[5] Magnesium hydroxide produced as an intermediate compound in the manufacture of magnesia or magnesium not included.

distributed throughout the continental United States. Known reserves of high-grade dolomite are large, but in most instances they are not so located that they can be mined economically under present conditions. Shortages of dolomite of the chemical purity needed to produce refractories were felt in 1953, particularly in the eastern United States. No actual estimate of reserves is available. In 1944 reserves of magnesite in Washington and Nevada were estimated to be 8 million tons of high-grade material, with probably an additional 85 million tons of lower grade inferred ore. Reserves of brucite in Nevada have been estimated at more than 3 million tons. Although serpentine and olivine had been utilized to a limited degree for making magnesium compounds by the close of 1953, they were considered to be potential sources of magnesium and refractory magnesia, and vast reserves are available. Reserves of

Table 11.—*Refractory magnesias (dead-burned and grain magnesite and periclase) imported for consumption in the United States, by countries, 1949–53*

(U. S. Department of Commerce)

Country	1949		1950		1951		1952		1953	
	Short tons	Value	Short tons	Value	Short tons	Value	Short tons	Value	Short tons	Value
Austria			11, 839	$622, 927	11, 314	$516, 886	18, 011	$785, 657	33, 026	$1, 634, 786
Brazil					56	1, 995				
Canada	1, 369	$133, 518	2, 104	188, 690	3, 995	365, 263	2, 074	204, 518	2, 888	648, 422
Czechoslovakia	1, 102	48, 000								
Germany					1, 000	47, 628				
India							1	21		
Italy			177	6, 009	3, 808	195, 377	2, 379	92, 029		
Norway					1, 000	41, 833	1, 504	64, 112		
United Kingdom					7, 612	227, 422	500	15, 400		
Yugoslavia									3, 383	185, 191
Total	2, 471	181, 518	14, 120	817, 626	28, 785	1, 396, 404	24, 469	1, 161, 737	39, 297	2, 468, 399

Table 12.—*Lump and ground caustic-calcined magnesia imported for consumption in the United States, by countries, 1949–53*

(U. S. Department of Commerce)

Country	1949		1950		1951		1952		1953	
	Short tons	Value	Short tons	Value	Short tons	Value	Short tons	Value	Short tons	Value
Austria			6	$245	496	$19, 949	303	$10, 003	56	$1, 778
Canada	1	$63	8	467	8	467	8	516		
Germany					32	1, 267				
Greece			44	1, 720	209	7, 800				
India	1, 230	43, 514	1, 458	54, 759	1, 963	71, 792	861	33, 347	1, 163	51, 908
Netherlands	245	15, 233	546	25, 911	1, 481	69, 137	16	941	16	891
United Kingdom	8	1, 108	9	1, 247	3	382				
Yugoslavia			55	2, 400			828	28, 391	1, 474	50, 636
Total	1, 484	59, 918	2, 126	86, 749	4, 192	170, 794	2, 016	73, 198	2, 709	105, 213

olivine averaging 45 to 29 percent magnesia are estimated at 230 million tons in North Carolina and Georgia and over 50 million tons in Washington.

In 1953 reports showed large reserves of magnesite in the following countries:

Austria.—The main deposits are in the Province of Styria, and the largest workings are at Veitsch and Radenthein. At Veitsch the magnesite deposit is about 500 feet wide and 1 mile long and has been explored by an adit at a depth 700 feet below the surface (*33*). Reserves are said to be adequate to last 50 to 100 years at the 1953 rate of production. Magnesite also is produced in Corinthia, Salzburg, and Tirol Provinces.

Brazil.—During 1938–43 eight deposits were discovered in central Ceara near the towns of Oros, Alencar, and Jucis. Bodenlos stated that 5 of the deposits were over 1 kilometer long and 155 to 450 meters wide (*4*). Inferred ore was estimated to be 4 million tons per meter of depth. The ore is high-grade; Fe_2O_3 and Al_2O_3 average about 1.5 percent. In 1939 a group of high-grade magnesite deposits was discovered west of Brumado, in southwestern Bahia, Brazil, which contain minable estimated reserves of almost 3 million tons per meter of depth (*5*).

Canada.—The main source of magnesite in Canada is near Calumet, Argenteuil County, Quebec. The material is an intimate mixture of magnesite and dolomite. Production from these deposits began in 1907. The ore occurs in two main zones in irregularly elongated masses, the largest of which is about 1,200 feet long, an average of 50 feet wide, and at least 400 feet deep. A large deposit of dolomitic brucite occurs near Rutherglen, Ontario. In 1932 a belt of magnesite 30 to 50 feet wide and over 4 miles long was discovered in the Cranbrook area of British Columbia (*33*). A mixture of magnesium chloride and magnesium sulfate occurs in some alkali deposits and lakes in Saskatchewan, associated with sodium sulfate. The MgO content of the larger lakes is 3 to 5 times that of sea water, and a preliminary estimate from a survey of 6 lakes indicated reserves of more than 13,000,000 tons equivalent of MgO (*52*).

Greece.—Pure white, high-grade cryptocrystalline magnesite occurs in large veins and lenses in serpentine. Minable reserves were estimated on Euboea Island, on the Chalcidice Peninsula in Macedonia, and on Mytileni Island. Minable reserves on Euboea Island alone were estimated to exceed 1 million tons assaying 94 to 98 percent $MgCO_3$ (*51*).

India.—Deposits occur in several Indian States, but the most extensive are near Salem, Madras Presidency. The Salem deposits extend over about 2,500 acres and crop out on numerous hills 60 to 100 feet above the plain level. In 1946 reserves were estimated at over 82 million long tons (*8*).

Other Countries.—Other countries with large reserves of magnesite are reported to include Australia, Czechoslovakia, Manchuria, Russia, Venezuela, and Yugoslavia, but detailed information is not available.

TARIFF

Under the Tariff Acts of 1909 and 1913 magnesite, crude or calcined, not purified, was imported into the United States duty free. The act of 1922 placed these commodities on the dutiable list; and a Presidential proclamation, under the flexible-tariff provision, effective December 10, 1927, increased the rates on crude magnesite and on caustic-calcined magnesite but made no change in the duty on dead-burned magnesite. Under the Tariff Act of 1930 no further changes were made in the rates, but periclase was specifically included with the category of dead-burned magnesite. In April 1939 the United States imposed a countervailing duty of 25 percent upon imports of all commodities from Germany in addition to the ordinary duties; in view of the Anschluss in March 1939, this rate applied as well to imports from Austria, the principal supplier of imports of refractory magnesia.

Table 13 compares the rates for the tariff on magnesium compounds imported to the close of 1951 (*48, 49*).

TABLE 13.—*Changes in United States tariff rates on crude magnesite; dead-burned and grain magnesite and periclase; and caustic-calcined magnesite* [1]

[Cents per pound]

Item	Tariff rate in—						
	Act of 1909	Act of 1913	Act of 1922	1927 [2]	Act of 1930	1948 [3]	1951 [4]
Crude magnesite	Free	Free	5/16	15/32	15/32	15/64	-----
Dead-burned and grain magnesite and periclase.	do	do	23/40	-----	23/40	-----	23/60
Caustic-calcined magnesite	do	do	5/8	15/16	15/16	15/32	-----

[1] U. S. Tariff, par. 204.

[2] Presidential proclamation, effective Dec. 10, 1927.

[3] General Agreement on Tariffs and Trade, Geneva, effective Jan. 1, 1948.

[4] General Agreement on Tariffs and Trade, Torquay, effective Oct. 19, 1951.

Dead-burned basic refractory material, consisting chiefly of magnesia and lime, is imported under paragraph 214, U. S. Tariff, Earthy or Mineral Substances, n. s. p. f. (not specifically provided for). The rate at the close of 1953 was 15 percent ad valorem. It had become effective January 1, 1948, under the General Agreement on Tariffs and Trade, Geneva. Before 1948 the rate was 30 percent ad valorem.

OUTLOOK

Successful development of methods of extracting the magnesium-bearing compounds from sea water has assured the world of a virtually inexhaustible supply, but the process requires high-grade dolomite and limestone for reaction. Reserves of these are exhaustible. Production reports in 1953 indicated that the trend of deriving an increasingly greater proportion of magnesia from sea-water processes would continue. New applications for refractories, resulting from new engineering developments, such as jet propulsion, atomic energy, and guided missiles, are creating greater demands for better refractories. New metallurgical and chemical processes require extremely high temperatures for efficient operation. Although the demand for refractories to withstand higher temperatures continues, the maximum output per unit of cost remains the decisive factor.

Costs of transportation from Washington and Nevada deposits to large consuming centers (many of them east of the Mississippi River) have encouraged expansion of the production of magnesia from sea water in the coastal areas. However, the capital cost of sea-water magnesia-production facilities was much higher, and shortages were experienced in 1953 in local supplies of high-grade dolomite and limestone required for reaction.

Extensive deposits of high-grade magnesite in Ceara, Brazil, were being explored and evaluated in 1953 as a possible source of refractory magnesia for eastern consuming centers in the United States.

Although the abundance of raw-material resources insures that the United States will remain self-sufficient in magnesia and other magnesium compounds, the trend toward increasing imports, especially from Austria and Yugoslavia, may result in lower domestic production.

PROBLEMS

The problem of providing increasingly better magnesia refractories at lower unit cost is of major importance in expanding their utilization. Known reserves of high-grade dolomite are large, but in most instances they are not so located that they can be mined economically under present conditions. High transportation costs are hindering further expansion of domestic magnesite deposits. Before the use of olivine can be expanded beyond the present limited quantity for the manufacture of forsterite refractories, further exploration, testing, evaluation, and research must be conducted. Research in refractory development and production and utilization techniques requires physical facilities and professional skills too costly for small producers and consumers.

Of even greater concern to magnesia producers, whether their product is derived from natural ores or from sea water or well brines, are the increasing imports of caustic-calcined and dead-burned magnesia. Surplus tonnage is available in both Austria and Yugoslavia, and the ocean freight rates in effect between the Adriatic and American ports are low compared with rail transportation rates from domestic producing plants to east coast consuming centers.

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MANGANESE

By

Gilbert L. DeHuff[1]

STEEL is fundamental to an industrial economy, and manganese is essential to steel production. Unless the steelmaking process is changed from present practice, there is no substitute for manganese. Manganese is important as a nonferrous alloying element, has important chemical uses, and (in the form of a particular grade of manganese dioxide) is essential to the manufacture of dry batteries. Tonnages involved are small compared to those used by the steel industry but are not so small in terms of availability of the special grades or forms required.

Summary

Domestic production of manganese ore is only a small portion of domestic consumption and a much smaller portion of world production; the latter comes from manganese deposits of several geologic types, mined by both open-pit and underground methods, and, until recently, little beneficiated except for simple concentration processes. Smelting practice is similar to that for iron ore, both blast and electric furnaces being used for production of the manganese ferroalloys used in producing steel. In recent years manganese metal, produced electrolytically instead of by smelting, has been in increasing demand for special applications that warrant the higher cost of the high-purity metal. Iron is an important coproduct of manganese, and certain nonferrous metals are also recovered from the mining of some manganese ores.

Some substitution for manganese is possible in the production of steel, but no substitutes have been developed to the point that they can be used in quantity. Little substitution is to be expected for the battery and major chemical uses. Some conservation of ferromanganese is possible under certain conditions through substitution of spiegeleisen, silicomanganese, or other ferroalloys, but little appears to be expected by way of conservation measures for manganese itself.

Synthetic battery-grade ore is in demand for the manufacture of dry cells, but its cost is too high and the productive capacity too low for it to be considered for metallurgical purposes.

Appreciable quantities of manganese are returned to open-hearth furnaces as a constituent of the scrap fed to the furnaces, and some manganese-bearing open-hearth slags are recycled to pig-iron blast furnaces. The manganese content of the wasted portion of open-hearth slags shows considerable promise as a future source of manganese, and methods for its recovery are under investigation.

World reserves of high-grade manganese ore are estimated to be approximately 1 billion tons, of which roughly two-thirds lies in the U. S. S. R. and associated countries. The bulk of the remainder is found in Africa, India, and Brazil. The United States, the largest consumer with the possible exception of the U. S. S. R., has large deposits of low-grade refractory manganiferous

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[1] Commodity-industry analyst, Bureau of Mines.

materials but very little high-grade ore. Approximately 95 percent of United States consumption goes to the steel industry, where it is necessary to the production of steel and is an important constituent of certain alloys. Manganese dioxide, either natural or synthetic, but of particular crystal structure, is required for the manufacture of dry-cell batteries, and manganese or manganese ore is important to certain portions of the chemical and nonferrous-metal industries.

Manganese is high on the list of strategic and critical materials, with a large portion of the imports coming from distant overseas countries. This situation has been recognized and has resulted in the establishment of a National Stockpile for each of the three grades of manganese ore—metallurgical, battery, and chemical; and in research activities largely pointed toward development of the Nation's potential resources of manganiferous materials.

Domestic commercial prices for manganese ore are governed by the market prices for imported ore, which include ocean freight charges. United States duty on manganese ore is one-quarter cent a pound of contained manganese from most countries, 1 cent a pound of contained manganese from U. S. S. R. and certain associated countries; manganese ore from Cuba and the Philippines is duty free.

Because of the drop in the average grade of imported manganese ores in recent years, industry has found it necessary to lower the grade of ferromanganese produced while at the same time the steel industry is being called upon for steels of higher and closer specifications. This reflects the basic problem of the domestic manganese industry, namely, the need for domestic sources of good-grade ore procurable in quantity at economical prices.

BACKGROUND

THE MANGANESE INDUSTRY

The manganese industry consists of ore producers, traders, and ore consumers. The producers are the operators of mines and processing plants. Consumers are ferroalloy producers, iron and steel producers, chemical plants, battery manufacturers, and segments of the nonferrous-metal industry. Traders may be independent dealers (usually export-import firms) or affiliates of prominent consumers.

Domestic production of metallurgical-grade manganese ore in 1953 was less than 10 percent of consumption and came from 3 principal sources: Nodule production by Anaconda Copper Mining Co. from low-grade Montana carbonate ores; nodule production by Manganese, Inc., from low-grade Nevada oxide ores; and shipments of ore and concentrates on the Government Domestic Manganese Purchase Program, largely coming from Virginia, Arkansas, and Tennessee. Two producers of battery concentrates, in Montana and California, and one Nevada producer of synthetic battery ore made from low-grade domestic ores contributed substantially in that year to national requirements for battery-grade ore. No chemical grade ore, as such, was produced domestically, although there was some chemical use of the nodules produced by Anaconda.

Except for the Anaconda Copper Mining Co., a producer of ferromanganese, the domestic manganese-ore-producing industry is not integrated with the consuming industries. The principal ferromanganese producers, Electro Metallurgical Co., United States Steel Corp., and Bethlehem Steel Co., do have important interests in foreign manganese ore mining operations, however. Another, E. J. Lavino & Co., is an important dealer in foreign manganese ores. In addition to those companies already mentioned, some 6 or 7 others produce ferromanganese, silicomanganese, or spiegeleisen. Manganese metal, most of which is produced as electrolytic manganese, is made by two companies: Electro Manganese Corp., and Electro Metallurgical Co.

World production of manganese ore is currently estimated at some 8 to 9 million metric tons a year containing 3 to 3½ million tons of manganese metal. United States production of manganese ore in 1953 was 143,000 metric tons, and consumption was approximately 2,000,000 metric tons.

GEOLOGY

Manganese occurs in a wide variety of oxide, silicate, and carbonate minerals, and the deposits are of four geologic types: Hydrothermal, sedimentary, residual, and metamorphosed. Of these, the sedimentary and residual have been most important economically, followed by the metamorphosed. However, any one locality may have evidence of more than one type of deposition.

Hydrothermal ore bodies are formed by deposition from hot-water solutions whose source is held to be deep within the earth. Although known occurrences do not constitute major sources of good-grade ore, hydrothermal deposits probably provided the manganese for many high-grade ores known today.

Sedimentary deposits, such as those of Chiatura and Nikopol in Russia, are layered deposits formed in seas or lakes by either chemical or mechanical action or both. The chemical action consists of leaching the manganese minerals from the primary source rocks or other deposits with redeposition as oxides either in place or at some distant locality.

Residual deposits result from the leaching of gangue minerals from the primary source rocks or from other deposits, leaving behind a residual concentration of the manganese minerals.

The metamorphosed deposits result from intense alteration by heat and pressure of any of the other types of deposits, but most of the metamorphosed deposits were originally sedimentary.

DESCRIPTION OF UNITED STATES DEPOSITS

Table 1 gives a general idea of the magnitude and distribution of the principal reserves of manganiferous materials, exclusive of slags, in the United States. Brief geologic descriptions of the first four deposits follow. These contain over 95 percent of the metallic manganese as indicated in the table.

Aroostook County, Maine.—Bedded deposits enclosed in steep dipping slates; intricately folded and faulted; tested in depth to 1,550 feet by diamond drilling. Recognized manganese minerals include various silicates and manganiferous carbonates.

Cuyuna Range, Minn.—Excluding the now commercial manganiferous iron ores, which are the result of oxidation of primary iron-bearing

TABLE 1.—*The 11 largest manganese deposits of the United States* [1]

[Tonnages and grade are in most instances crude estimates, not based on extensive exploration

District	Raw material, tons	Metallic manganese, tons	Average grade percent
1. South Dakota, Chamberlain district	2, 500, 000, 000	45, 000, 000	1–2
2. Minnesota, Cuyuna range	500, 000, 000	25, 000, 000	5
3. Maine, Aroostook County	314, 000, 000	28, 000, 000	9
4. Arizona, Artillery Peak	195, 000, 000	7, 500, 000	4±
5. North and South Carolina, Gaffney-Kings Mountain	25, 000, 000	1, 000, 000	3–4?
6. Montana, Butte district	5, 000, 000	700, 000	14
7. Colorado, Leadville district	4, 000, 000	600, 000	15
8. Nevada, Three Kids district	5, 000, 000	500, 000	10
9. Nevada, Pioche district	4, 000, 000	400, 000	10
10. Montana, Philipsburg district	800, 000	180, 000	22. 5
11. Arkansas, Virginia, Tennessee, Georgia, and Alabama	(2)	(2)	----------

[1] Excluding slags.
[2] Insufficient data for estimation. Deposits are scattered and of variable character.

formations in pre-Cambrian rocks, the low-grade manganese resources consist of (1) greenish carbonate slate, an unoxidized primary source; and (2) oxidized materials, both carbonate slate and cherty iron formation, too low-grade to be marketable at present. The iron- and manganese-bearing horizons have been strongly folded and faulted.

Artillery Peak, Ariz.—Sedimentary deposits of three structural types have been recognized, but only the bedded oxide deposits of the Chapin Wash formation offer commercial promise. These are found in 2 zones, separated 750 to 1,000 feet stratigraphically. The greater portion of the reserve tonnage is found in the upper zone, in which the manganiferous material is much more continuous than in the lower.

Chamberlain, S. Dak.—A very large, horizontal, deposit of manganese-bearing shales, averaging perhaps 50 feet thick in its central portion, crops out along the banks of the Missouri River and its tributaries in south-central South Dakota for a total meander length of over 500 miles. The manganiferous zone of interest consists of alternating layers of bentonitic shale, bentonite, and manganese-iron carbonate concretions. The concretions form discontinuous layers 1 to 9 inches thick and constitute 3.78 percent of the zone; they have an average content of 15.5 percent manganese, while the enclosing shale of the zone has a non-nodular manganese content varying from 0.04 to 1.15 percent.

MINING METHODS

Manganese mines range from primitive to the most modern of operations. For the most part they are either open pits or relatively shallow underground workings, an exception being the mines at Butte, Mont., where veins and stringers of rhodochrosite ore in quartz and altered granite are mined to depth.

The Nsuta mine, Gold Coast, is a modern open pit of some 3,000 tons per day capacity, provided with diesel-electric power, railroad, and dock facilities. It is understood to be the largest single manganese mine in the world.

The longwall retreating method of mining, common to many coal mines, is understood to be in use at both Nikopol and Chiatura, where the deposits occur as flat-lying beds. In this method the area to be served by shaft or adit is first developed to its outer boundaries by haulageways. Mining then begins at these outer limits, and the bed is completely removed progressively back toward the shaft or portal, timbers supporting the roof temporarily while the ore is being removed from the working face. At Nikopol, where there are reported to be some 20 operating mines, the manganese beds lie 50 to 100 feet below the surface and are about 50 square miles in total extent, though not continuous. At Chiatura in the Caucasus Mountains, the mined bed, which averages 6 to 7 feet in thickness, outcrops on steep hillsides and is entered by adits. The ore is soft and according to reports is drilled with electric augers. The Chiatura deposit is continuous over 14 square miles and is the largest single deposit known. Mechanization is reported to be well advanced.

Although current details are lacking, underground mining is done in the Lafayette and Miguel Burnier districts of Brazil, in Chile, at Bou Arfa and Imini in French Morocco, at Charco Redonda in Cuba, and in Bombay State and the Central Provinces, India. Open-pit methods are used in most Indian operations, in South Africa, and in Belgian Congo.

ORE DRESSING

The milling of manganese ores have had minor commercial importance until recently. Of the large deposits contributing to the world supply, most require only relatively simple concentrating processes, such as log washing and primary crushing. Only in times of abnormal market conditions, when the marginal deposits become profitable, have milling and other mechanical methods of beneficiation become of importance.

Domestically, many methods of beneficiation have been tried with varying degrees of success on representative numbers of low-grade deposits. Differing characteristics of the various ores require corresponding differences in milling techniques, complicating the problem of domestic production from low-grade ores by limiting the quantities of raw ore that can be treated by any one established milling process in any geographical area.

SMELTING, ETC.

The basic similarity of manganese ore and iron ore has permitted the use of similar furnaces for smelting the ores. The major portion of the ferromanganese produced in the United States is smelted in blast furnaces. Recoveries of metal charged are not as high for manganese as for iron, roughly 85 percent of manganese in the charge being recovered in the metal with 15-percent loss distributed between slag and fume.

The flow of manganese through the production of pig iron and ferromanganese into the finished steel ingot is shown in figure 1. It will be noted that there are two distinct source

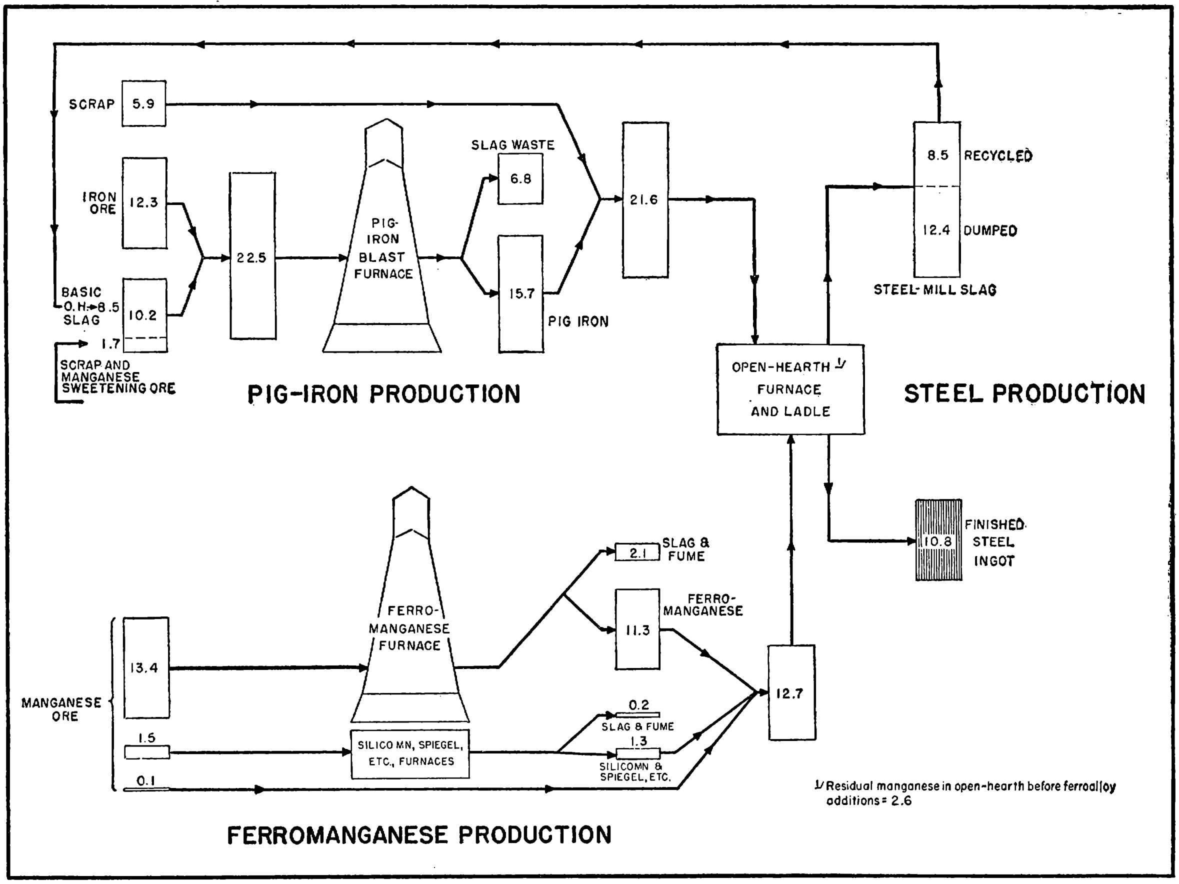

FIGURE 1.—Empirical Flowsheet Showing Approximate Consumption and Losses of Manganese in Production of Pig Iron, Ferromanganese, and Steel.

[Pounds of manganese per ingot ton of steel.]

groupings of manganese: (1) Iron ore, scrap, open-hearth slag, and a small quantity of manganese ore for sweetening, in the pig-iron flowsheet and (2) straight manganese ore in the ferromanganese flowsheet.

Ferromanganese production uses standard ferrograde manganese ore, which may be defined for this purpose as an ore having a manganese-iron ratio no lower than approximately 6 : 1 and containing a minimum of phosphorus and as little gangue as is economically practical. The sweetening ores used for pig-iron production, on the other hand, need have no set manganese-iron ratio, although a minimum of phosphorus and gangue are again desirable.

In calculating the efficiency of the pig-iron and steel-making processes in terms of manganese, it should be emphasized that these processes are designed for production of iron and steel, not manganese. Manganese is used in the process to counteract the effects of oxygen and sulfur and having accomplished this purpose is discarded. Whatever manganese remains in the steel has a good effect on the product. It may be seen from figure 1 that the total net input of manganese is 34.9 pounds per ton of ingot (43.4 pounds less 8.5 pounds recycled as slag) as against a recovery of 10.8 pounds in the finished ingot ton. Net loss is thus 24.1 pounds per ton; and recovery efficiency, in terms of manganese, is 31 percent.

The manganese in the blast-furnace slag, although it amounts to 6.8 pounds per ingot ton of steel, is not considered recoverable, as these slags assay only about 1 percent manganese.

The slag and fume losses in the ferromanganese operation, although relatively small, are thought to be recoverable.

It will be noted that the dumped open-hearth and ladle slags contain almost as much manganese per ingot ton as is used as input for the ferromanganese circuit. Considerable attention is now being given to the possibilities of returning this manganese to the ferroalloy feed.

Electric-furnace smelting has been concerned with relatively small tonnages compared to blast-furnace smelting and has tended to be more costly. These factors have favored use of the electric furnace for production of special grades of ferromanganese, silicomanganese and other manganese alloys that command higher prices; or for special situations, such as those where high transportation costs compensate for higher smelting costs.

Because of the availability of furnaces of types capable of smelting manganese ores into usable alloys, there has been little incentive to investigate alternative, more efficient, smelting techniques, particularly when such innovations may require large amounts of capital investment for equipment.

Some manganese metal, as distinguished from the manganese ferroalloys, is produced in electric furnaces; but most is produced as electrolytic manganese—an almost pure metal. The production of electrolytic manganese entails a leach of the ore followed by electrolysis and eliminates smelting. Like the electric furnace, there are limiting factors of tonnage and costs. It does have the advantage, however, that low-grade ores can be used for feed, although at higher cost.

BYPRODUCTS, COPRODUCTS, AND COMMODITY RELATIONSHIPS

The most important coproduct of manganese is the iron contained in most manganese ores. During the production of ferromanganese, in which the manganese-iron ratio is desired at 6 or 8 :1, the iron becomes part of the alloy and enters the steel along with manganese at the time alloy additions are made. Iron assumes even greater importance in the low-manganese, high-iron ores used in pig-iron blast furnaces. Its presence makes possible shipment of ores that would not be moved if the cost had to be carried by manganese alone. The Egyptian ores, containing 26 percent manganese and an equivalent quantity of iron, are an example, being credited with approximately 50 percent metal.

Some manganese ores could not be mined economically were it not for their content of lead, zinc, silver, or other nonferrous metals.

Commodity relationships are largely those of measuring the quantities of manganese needed to supply the demands for other commodities. Since such a large portion of the manganese consumed is used in steel production, the manganese-steel ratio may be used in forecasting manganese demands.

SUBSTITUTES

In the production of steel in the United States under emergency conditions the tendency is to substitute manganese for other alloy elements that may be in relatively shorter supply. All satisfactory substitutes for manganese known today are more difficult to obtain, and none is developed to the point wherein it could be used in appreciable quantities. For example, titanium can substitute for some of the functions of manganese, but its use introduces new technical problems. Moreover, the quantities of titanium necessary to replace effectively the quantities of manganese being consumed are astronomical considered from the point of availability of titanium.

For battery use there is no real substitute for manganese as long as the present type of dry cell is used. During and after World War II the mercury battery was seriously investigated, but owing to certain inherent difficulties with the cell it is believed that it will not replace the present type of dry cell.

The chemical industry has many applications for manganese, some of which are not essential in an emergency. Although changes in processing methods may at times eliminate requirements for manganese, the greater part of chemical consumption offers little chance for substitution at present.

SYNTHETICS

Synthetic manganese ores are limited in this section to the production of manganese dioxide by chemical or electrolytic means and does not include those processes involving ordinary beneficiation or certain pyrometallurgical processes capable of producing a usable type of manganese ore from very low grade materials.

Synthetic-ore production is now limited to supplying the battery and possibly chemical industries where special uses for this high-grade product are found and premium prices paid. Although several processes are known that will produce manganese dioxide of high purity, the synthetic product has not been able to compete with natural ores for metallurgical use because of cost. Known processes require fairly large capital expenditures for recovering small tonnages of high-grade product. To provide appreciable quantities of manganese for metallurgical purposes by proved methods of synthetic-ore production would require plant expansion out of proportion to the value of the manganese produced, except in an extreme emergency.

SECONDARY SOURCES AND RECOVERY

Although appreciable quantities of manganese are returned to the open-hearth furnace with the scrap, little of it is recovered in the finished ingot, as the action of the steelmaking process removes manganese from the metallic bath and discards it in the flush and tap slags. This possible source of manganese—open-hearth slag—is under intensive investigation at present and is discussed more fully later. A certain portion of the basic open-hearth slags produced are at present used to the extent of being recycled in the pig iron production cycle (fig. 1).

No practical economic methods are known for recovering manganese consumed in the battery and chemical industries. These quantities are small compared to the manganese needed annually and do not appear to justify intensive investigation. Some secondary use is made of manganese in the chemical industry, where manganese dioxide is used as an oxidizing agent and emerges from the primary process as manganese sulfate. This sulfate is recovered, where economically possible, and used in fertilizers and animal feeds to provide needed traces of manganese.

WORLD RESERVES

World reserves of high-grade manganese ore (table 2) are estimated to approximate 1 billion tons, of which roughly ⅔ are in U. S. S. R. and associated countries. Known reserves in other countries approximate 335 million tons, or ⅓ of the world total. Both geographical entities have possibilities for increasing reserves by further exploration and discovery.

At present the United States takes a large portion of the Free-World output and contributes virtually nothing to the Free-World reserve position for high-grade ores. India, Union of South Africa, and Brazil have small but growing steel industries of their own and will eventually be drawing more heavily on their domestic manganese deposits for home use. French Morocco directs much of its ore to Europe, and Gold Coast is called upon heavily for European needs.

UNITED STATES RESERVES

Manganese deposits have been reported in virtually every State of the United States. However, no deposits comparable in size and grade to those found in the major producing countries of the world have ever been discovered. The United States does have several large deposits of low-grade refractory material containing large quantities of manganese that could be recovered under conditions of improved technology, higher prices, or both.

PRODUCTION OF MANGANESE ORE

World production of manganese ore is shown in table 3. It is to be noted that the largest single item, Russian production, is an estimate subject to considerable possible adjustment.

Since 1941 domestic production of manganese ore (35 or more percent manganese content) has been above 100,000 short tons a year, reaching a peak of 248,000 tons in 1944 and a low of 105,000 tons in 1951; from which it increased to 158,000 tons in 1953.

TABLE 2.—*World reserves of manganese ore by countries* [1]

[Million tons]

Country	Estimated ore				Remarks
	High-grade		Intermediate		
	Tons	Grade	Tons	Grade	
North America:					
Cuba	1	45	1.2	±20	
Mexico	5	+45	1	±25	
United States	[2] 1	+35	11	13	
South America:					
Brazil	60	38–50			
Chile	1.2	+35			
Africa:					
Angola	.4	±48	3	(3)	Recent development not fully explored.
Belgian Congo	10	±45			
Egypt			±9	±25	Ferruginous manganese ore.
French Equatorial Africa	50	±48			Recent development not fully explored.
French Morocco	30	±45	20	±25	
Gold Coast	12	±50			
Union of South Africa	±60	32–50			Exploration continuing; potential considerable.
Europe:					
Czechoslovakia			4.5	18	Ferruginous manganese ore.
Greece	.4	30–56			Numerous small deposits.
Hungary	4.5	+35	6	8–20	
Italy	.5	34–37			
Rumania			7	14–40	
U. S. S. R.	±550	28–50	±75	17–24	
Yugoslavia	.5	32–38			
Asia:					
China	18	38–52	11	+20	
India	100	+40			
Manchuria			3	−30	
Republic of Philippines	.5	+35			Potential of numerous scattered deposits has been estimated as high as 6.
Turkey	±1	30–50			
Oceania: Australia	.5	42–52			Excludes manganiferous iron ore at Iron Monarch.
Total	±900		±200		

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[1] Data for many of the countries listed are inadequate to estimate firm reserve figures. However, data shown are believed to be in the correct order of magnitude.

[2] Numerous small deposits capable of yielding +35 percent concentrates without sintering, largely at premium prices.

[3] High iron.

USES OF MANGANESE AND MANGANESE ORE

Manganese, or manganese ore, is used for three principal purposes:

(1) In the steel industry, as a reagent in the steelmaking process and as an ingredient in special alloy steels.

(2) In the chemical industry, as an oxidizing agent in chemical processes and for production of permanganate and other manganese chemicals.

(3) In the manufacture of dry-cell batteries, as manganese dioxide, the depolarizing agent in the cell.

In addition, although large tonnages are not involved, manganese is important in the production of aluminum and magnesium and is otherwise an important nonferrous alloying element, particularly with copper for the manufacture of manganese bronzes and high-damping alloys.

The steel industry in the United States consumes about 95 percent of the country's total. When used in the production of steel (fig. 1), manganese ores, for the most part, are first converted for convenience and control to ferromanganese, silicomanganese, and spiegeleisen, alloys of manganese and iron of known composition. Manganese usage by the steel industry in large measure results from its affinity for oxygen and sulfur, its ability to

TABLE 3.—*World production of manganese ore, by countries,*[1] *1944–48 (average) and 1949–53* [2]

[Metric tons]

(Compiled by Berenice B. Mitchell and Pearl J. Thompson)

Country [1]	Mn (percent)	1944–48 (average)	1949	1950	1951	1952	1953
North America:							
Canada (shipments)	-------	41	-------	-------	-------	-------	-------
Cuba	36–50+	133, 269	62, 503	79, 209	154, 091	251, 677	[3] 353, 218
Mexico	40+	48, 566	53, 900	32, 400	79, 190	141, 696	244, 816
United States (shipments)	35+	151, 737	114, 427	121, 971	95, 260	104, 670	142, 914
South America:							
Argentina	35–38	[3] 3, 585	(4)	1, 200	2, 500	(4)	(4)
Brazil (exports)	38–50	164, 825	149, 896	148, 339	119, 900	172, 510	166, 101
Chile	40–50	22, 958	28, 870	33, 530	36, 578	[3] 48, 600	[3] 60, 000
Peru	40+	25	-------	762	946	2, 215	[3] 3, 200
Europe:							
Greece	35+	183	150	320	10, 592	22, 924	12, 533
Hungary (concentrates)[3]	35–48	25, 980	40, 000	40, 000	40, 000	40, 000	40, 000
Italy	30	17, 473	24, 302	19, 434	28, 557	40, 570	39, 156
Portugal	35+	5, 196	508	798	7, 615	11, 065	12, 546
Rumania	30–36	[3] 23, 161	66, 816	(4)	(4)	(4)	(4)
Spain	40+	25, 171	18, 651	19, 002	20, 790	28, 493	28, 968
Sweden	30+	(4)	-------	58	(4)	(4)	(4)
Switzerland	-------	1, 707	-------	-------	-------	-------	-------
U. S. S. R. (estimate)	[5] 41+	1, 602, 400	1, 500, 000	2, 000, 000	2, 500, 000	2, 500, 000	[5] 3, 500, 000
United Kingdom	-------	5, 874	-------	-------	-------	-------	-------
Yugoslavia	30+	6, 707	13, 837	13, 338	12, 868	12, 687	10, 017
Asia:							
Burma	35+	[3] 308	-------	-------	[3] 2, 000	6, 604	8, 718
China	41	[3] 15, 576	(4)	(4)	(4)	(4)	(4)
India	40+	368, 092	656, 190	897, 100	1, 304, 536	1, 485, 733	[3] 1, 752, 700
Indonesia	-------	2, 845	-------	-------	-------	7, 833	16, 986
Iran	36–46	-------	[3] [6] 4, 200	[3] [6] 9, 360	(4)	9, 327	[3] 6, 000
Japan	32–40	120, 793	100, 000	134, 066	198, 000	207, 376	183, 899
Korea, Republic of	30–48	(4)	(4)	100	2, 247	7, 416	3, 058
Malaya	30	1, 016	-------	[7] 72	[7] 195	-------	-------
Philippines	35–51	[3] 9, 788	26, 288	29, 867	22, 343	20, 627	21, 508
Portuguese India	32–50+	1, 199	16, 220	38, 220	86, 793	111, 066	150, 000
Turkey	30–50	4, 698	22, 576	32, 178	50, 517	106, 395	74, 884
Africa:							
Angola	48	1, 000	18, 600	9, 308	46, 192	55, 094	65, 864
Belgian Congo	50	9, 768	12, 247	16, 990	70, 945	127, 978	216, 664
French Morocco	35–50	91, 740	233, 825	287, 265	372, 233	426, 316	429, 517
Gold Coast (exports)[8]	48	641, 767	752, 963	722, 784	819, 018	806, 934	757, 963
Northern Rhodesia	30+	-------	-------	-------	1, 280	3, 989	7, 243
Southern Rhodesia	-------	[9] 10	166	-------	-------	1, 433	-------
South-West Africa	-------	-------	-------	993	6, 560	26, 507	36, 881
Spanish Morocco	50	[9] 13	653	36	1, 122	3, 635	3, 419
Tunisia	35–40	68	-------	-------	-------	-------	-------
Union of South Africa	40+	204, 786	655, 175	790, 937	758, 870	874, 637	827, 656
Oceania:							
Australia	-------	1, 984	13, 303	15, 108	8, 096	7, 182	19, 083
Fiji	-------	[9] 71	102	203	641	2, 042	(4)
New Caledonia	45+	-------	2, 100	5, 392	20, 135	16, 850	4, 715
New Zealand	-------	255	310	358	408	324	294
Papua	-------	127	[10] 163	22	41	(4)	43
Total (estimate)	-------	3, 800, 000	4, 600, 000	5, 600, 000	7, 000, 000	7, 800, 000	9, 300, 000

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[1] In addition to countries listed, Bulgaria, and North Korea have produced manganese ore; data of output are not available, but estimates for them are included in the totals. Czechoslovakia and Egypt report production of manganese ore, but because the manganese content averages less than 30 percent and these ores are essentially ferruginous manganese ores, the output is not included in this table. Egypt produced the following tonnages: 1944–48 (average), 12,010; 1949, 138,568; 1950, 152,169; 1951, 155,364; 1952, 209,164; and 1953, 278,806; occasionally a small tonnage contains more than 35 percent manganese.

[2] This table incorporates a number of revisions of data published in previous manganese chapters.

[3] Estimate.

[4] Data not available; estimate by author of chapter included in total.

[5] The 1953 production estimated for ore of 35 percent or more manganese content.

[6] Year ended March 20 of the year following that stated. 1952 and 1953 are calendar years.

[7] Exports.

[8] Dry weight.

[9] Average for 1 year only, as 1948 was the first year of production.

[10] Year ending June 30 of year stated.

alloy with iron, and its hardening effects. It is noteworthy that during war periods the demand for high-manganese steels increases considerably.

In the chemical industry, manganese oxide (usually the dioxide, MnO_2) is a source of oxygen. Manganese ore or metal is used in making various manganese salts.

In operation of the dry cell, hydrogen released from the electrolyte tends to form a nonconductive film around the carbon electrode. Manganese dioxide in the absorbent mix provides free oxygen, which combines with the hydrogen to produce harmless water. Both natural and synthetic MnO_2 are used for this purpose, but only certain crystal structures are active.

For uses where manganese is advantageously introduced in exceptionally pure form, such as the production of stainless steels, certain chemicals, and in the nonferrous metal industries, electrolytic manganese has become increasingly important.

CONSUMPTION

With the possible exception of the U. S. S. R., which uses large quantities of manganese ore in the manufacture of pig iron, the United States is the largest consumer of manganese ore in the world. No other country begins to approach either of these two in tonnage consumed.

Consumption of manganese ore in the United States is shown in table 4. For the 10-year

TABLE 4.—*United States consumption of manganese ore, 1925–53*

[Short tons]

(35 percent or more manganese content)

Year	Total consumption	Consumed by—			Steel production, ingots and steel for castings (thousand short tons)	Manganese ore consumed per ton of steel produced (pound)		Manganese content of ore imported for consumption (percent)	Manganese consumed per ton of steel produced (pound)	
		Metallurgical industry	Chemical industry	Battery manufacturers		Total for all uses	Metallurgical consumption only		Total for all uses	Metallurgical use only
1953	2, 195, 742	2, 127, 420	26, 775	41, 547	111, 610	39. 35	38. 12	44. 72	17. 60	17. 05
1952	1, 809, 184	1, 747, 878	21, 409	39, 897	93, 168	38. 84	37. 52	45. 43	17. 65	17. 05
1951	1, 892, 609	1, 821, 216	27, 266	44, 127	105, 200	35. 98	34. 62	46. 36	16. 68	16. 05
1950	1, 650, 429	1, 583, 834	25, 276	41, 319	96, 836	34. 08	32. 72	46. 64	15. 89	15. 26
1949	1, 360, 042	1, 309, 132	16, 441	34, 469	77, 978	34. 88	33. 58	46. 64	16. 27	15. 66
1948	1, 538, 398	1, 473, 259	17, 139	48, 000	88, 640	34. 72	33. 24	47. 66	16. 55	15. 84
1947	1, 419, 131	1, 353, 613	19, 162	46, 356	84, 894	33. 44	31. 88	48. 11	16. 09	15. 34
1946	1, 136, 687	1, 067, 803	21, 283	47, 601	66, 603	34. 14	32. 06	48. 88	16. 69	15. 67
1945	1, 485, 859	1, 405, 789	21, 227	58, 843	79, 702	37. 28	35. 28	48. 34	18. 02	17. 05
1944	1, 593, 098	1, 522, 451	20, 798	49, 849	89, 642	35. 54	33. 96	48. 13	17. 11	16. 34
1943	1, 588, 323	1, 531, 936	17, 283	39, 104	88, 837	35. 76	34. 48	48. 25	17. 25	16. 64
1942	1, 481, 672	1, 441, 110	[1] 4, 060	36, 502	86, 032	34. 44	33. 50	48. 25	16. 62	16. 16
1941	1, 310, 458	1, 272, 473	--------	37, 985	82, 839	31. 68	30. 62	48. 11	15. 22	14. 73
1940	1, 480, 866	--------	--------	--------	66, 983	44. 22	--------	48. 13	21. 28	--------
1939	735, 211	--------	--------	--------	52, 799	27. 84	--------	50. 04	13. 93	--------
1938	571, 122	--------	--------	--------	31, 752	35. 98	--------	48. 42	17. 42	--------
1937	1, 069, 043	--------	--------	--------	56, 637	37. 76	--------	48. 96	18. 49	--------
1936	950, 310	--------	--------	--------	53, 500	35. 52	--------	49. 89	17. 72	--------
1935	462, 880	--------	--------	--------	38, 184	24. 24	--------	49. 35	11. 96	--------
1934	413, 911	--------	--------	--------	29, 182	28. 36	--------	48. 58	13. 78	--------
1933	346, 048	--------	--------	--------	26, 020	26. 60	--------	--------	--------	--------
1932	124, 164	--------	--------	--------	15, 323	16. 20	--------	--------	--------	--------
1931	374, 293	--------	--------	--------	29, 059	25. 76	--------	--------	--------	--------
1930	587, 100	--------	--------	--------	45, 583	25. 76	--------	--------	--------	--------
1929	[2] 738, 080	--------	--------	--------	63, 205	[2] 26. 72	--------	--------	--------	--------
1928					57, 729					
1927					50, 327					
1926					54, 089					
1925					50, 841					

[1] September–December only.

[2] Average.

Average data for period 1944–53, inclusive, are as follows: Mn ore consumed per net ton of steel produced: (*a*) All uses, 35.96 pounds; (*b*) metallurgical only, 34.47 pounds. Elemental manganese consumed per net ton of steel produced, on basis of Mn content of ores imported for consumption: (*a*) All uses, 16.87 pounds; (*b*) metallurgical only, 16.16 pounds.

period 1944–53, 36 pounds of manganese ore averaging 46.7 percent in manganese content was consumed in the United States for all purposes per short ton of steel produced. Of this quantity, chemical and battery consumption was 1.5 pounds and metallurgical 34.5 pounds.

STATISTICAL INFORMATION

Statistics on manganese imports, consumption, and production have been steadily improved to date. Domestic processers, consumers, and producers are canvassed at regular intervals on a voluntary reporting basis, and import data are obtained by the Customs Offices. The most difficult group with which to maintain proper contact is the smaller domestic producers, consisting largely of individuals and partnerships whose operations are sporadic and dependent on market conditions that can change almost overnight insofar as these individuals are concerned.

Statistics on production from foreign mines and international trade in manganese are subject to great differences in degree of availability and accuracy; however, mineral attachés and similar United States officers supply a large volume of valuable information. Statistics on manganese consumption in foreign steel practice also are not too satisfactory due to the many different ways of reporting such data. These problems are not limited to manganese, however.

Proper correlation and interpretation of data pertaining to manganese require considerable experience and knowledge of all phases of the subject and do not lend themselves well to routine mechanical tabulation and analysis.

ESSENTIALITY, SELF-SUFFICIENCY, AND STRATEGIC NATURE

Manganese is essential to the manufacture of steel, dry cells, and certain chemicals.

In recent years, excepting 1954, the United States has produced less than 10 percent of its manganese requirements from domestic deposits and has obtained only an additional 15 to 35 percent from other Western Hemisphere sources, the balance being imported mainly from Africa and India (table 5). Domestic production in 1954 was of slightly higher proportion due to the Government's domestic manganese purchase programs, together with reduced

TABLE 5.—*Manganese ore (35 percent or more Mn) imported into the United States, 1952–53, by countries*

(Compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce)

Country	General imports [1] (short tons)				Imports for consumption [2]					
					Short tons				Value	
	Gross weight		Mn content		Gross weight		Mn content			
	1952	1953	1952	1953	1952	1953	1952	1953	1952	1953
Angola	[3] 64,463	63,863	[3] 31,501	31,141	[3] 68,297	64,395	33,655	31,343	[3] $3,030,600	$2,768,024
Australia	-----	10,320	-----	4,541	-----	10,320	-----	4,541	-----	347,500
Belgian Congo	54,144	140,478	27,000	70,198	56,321	140,478	28,032	70,198	2,496,334	5,784,670
Brazil	169,372	169,768	75,052	75,821	174,241	155,373	77,448	68,844	4,300,963	4,863,567
British Western Pacific Islands	-----	1,176	-----	588	-----	-----	-----	-----	-----	-----
Burma	-----	554	-----	324	-----	-----	-----	-----	-----	-----
Chile	[3] 28,590	61,799	[3] 12,733	28,306	21,733	32,416	9,646	13,969	757,713	1,126,076
Costa Rica	91	364	38	167	-----	455	-----	206	-----	20,302
Cuba	259,230	397,257	113,051	172,355	259,230	397,257	113,051	172,355	8,801,648	13,990,786
French Morocco	85,316	73,587	[3] 42,872	36,761	74,191	66,730	37,159	33,616	[3] 3,343,136	3,313,568
French Pacific Islands	22,459	7,963	10,303	3,699	9,628	14,360	4,529	6,459	353,533	424,019
Gold Coast	[3] 368,069	511,259	[3] 177,377	253,355	[3] 282,053	333,622	135,350	164,526	8,204,873	14,309,989
Greece	8,372	6,569	3,175	2,982	1,350	4,046	547	1,660	29,446	104,265
India	1,028,289	1,296,905	461,763	572,640	772,456	1,218,174	352,018	539,871	[3] 22,318,538	38,449,667
Indonesia	13,126	6,984	5,959	3,092	13,878	7,390	6,340	3,472	567,414	292,235
Mexico	135,718	171,462	57,214	70,936	92,251	129,862	39,258	55,227	2,675,885	3,886,241
New Zealand	545	-----	265	-----	545	-----	265	-----	24,887	-----
Peru	3,687	4,704	1,736	2,005	3,755	1,153	1,766	570	140,046	39,326
Philippines	10,587	12,955	4,677	5,859	10,587	12,955	4,677	5,859	321,594	459,187
Portugal	-----	6,593	-----	3,106	-----	-----	-----	-----	-----	-----
Portuguese Asia	55,815	88,565	23,902	36,961	45,777	79,210	19,417	33,070	1,686,791	3,027,397
Turkey	41,149	39,513	18,545	18,140	18,908	40,803	8,319	18,793	666,577	1,764,381
Union of South Africa	319,719	428,348	140,552	177,943	298,305	406,024	129,567	168,421	6,110,503	8,548,845
United Kingdom	[3] 39	-----	[3] 24	-----	[3] 39	-----	[3] 24	-----	[3] 4,410	-----
Total	[3] 2,668,780	3,500,986	[3] 1,207,739	1,570,920	[3] 2,203,545	3,115,023	1,001,068	1,393,000	[3] 65,834,891	103,520,045

[1] Comprises ore received in the United States during year; part went into consumption, and remainder entered bonded warehouses.

[2] Comprises receipts during year for consumption and ore withdrawn from bonded warehouses during year; excludes imports for manufacture in bond and export.

[3] Revised figure.

consumption. The United States will continue to be dependent on foreign manganese ores until domestic low-grade resources can be economically drawn upon. This can be achieved only by technologic developments, which will probably require the assistance of higher prices as well.

Manganese becomes extremely strategic and critical on the United States list of such materials when essentiality, bulk, capital investment required, and lack of self-sufficiency are all considered.

STRATEGIC CONSIDERATIONS

Ocean-borne manganese ore is highly vulnerable to attack due to long shipping lanes and the large tonnages involved. Exposure time of most ore-trade vessels to enemy attack averages 4 to 6 weeks, and alternative means of international transportation do not appear feasible. The movement of manganese ore by air, while not impossible when considered as the only commodity to be moved, would cause a disproportionate drain on United States resources in emergency periods.

The domestic manganese-alloy industry is closely associated with the steel-producing industry and is concentrated in the same general areas. Most manganese ore is delivered to the east coast of the United States and is processed into ferromanganese and other alloys at scattered locations within a 350-mile radius of Baltimore, Md., the major port of entry.

High transportation costs to eastern consuming areas have been a factor retarding production of manganese in the western portion of the United States.

Since certain major consumers of manganese alloys are also alloy producers, the problem of alloy transportation otherwise is not serious. Although the alloy producers are largely concentrated in the northeastern part of the country they are far enough apart to prevent total destruction by any one blow and yet close enough to permit the movement of raw materials from one plant to the other in the event of the destruction or crippling of any one facility.

DEFENSE PROGRAM

The purpose of stockpiling is to provide a supply of strategic material that would be otherwise impossible or difficult to obtain during war or an emergency. The National Defense Program provided an abnormal demand for manganese ore of all three grades (metallurgical, battery, and chemical) by superimposing the demands for material to complete the National Stockpile upon demands of industry, which in themselves were abnormally high because of the large quantities of steel required by the defense effort over and above those quantities needed in ordinary economic growth.

Two principles were followed for accumulating enough stocks to carry through the emergency period, while at the same time maintaining production.

Every effort was made to increase the flow of imports of ore from those areas of the world already producing and those containing undeveloped reserves. Exploration and development loans were made, and purchase contracts consummated, all designed to facilitate and encourage new or increased production. These efforts, aided by high world prices, resulted in record imports at some expense to grade of ore imported. Now, with reduced stockpile commitments and reduced industrial demands, the foreign situation is in danger of approaching its position at the beginning of the last emergency, except that the position of the Western Hemisphere has become more favorable as a source of ore for the United States.

Similar policies have prevailed in regard to domestic production, with the important addition of incentive prices. Prices for domestic ore were set at almost double the price for imported ores of the same quality. The ores purchased by General Services Administration are either resold to the National Stockpile at market prices or to industry as the individual case might dictate. The results have been to increase domestic production 50 percent in 2½ years and 100 percent in 3½ years. This includes the new production of manganese nodules from low-grade Nevada oxide ores by Manganese, Inc., at Henderson, Nev., the production of synthetic manganese dioxide from low-grade domestic oxide ores by Western Electrochemical Co., also at Henderson, Nev., and a revival of manganese mining in Virginia and Tennessee.

Stockpiling, while desirable and necessary, is only a means of attaining national security for a limited time. Investigation, research, and development of domestic sources constitute the route to long-term self-sufficiency. As a part of the Defense Program, considerable research has been devoted to the problem of finding practical means for recovering manganese from the country's refractory manganiferous materials, and several companies have been engaged in testing such processes in large-scale pilot plants or constructing plants for that purpose or for actual commercial production.

PRICES AND COSTS

Prices of manganese ore in the United States have always been subject to the influence of the price of imported material. In periods of high demand, when prices are raised, domestic production responds with larger tonnages. This is only true as long as manganese-ore prices advance faster than the general price index, so that marginal and submarginal operations are attractive. As soon as general prices catch up to advanced manganese prices, or manganese prices return to the general price level, then domestic production falls off sharply, if not supported, as at present, by special Government prices.

Transportation costs are an important factor in the price of manganese ore from foreign sources, as approximately one-third of the average price delivered at eastern seaboard is chargeable to ocean freight. Since this high proportion of the final cost is transportation, deliveries of manganese ore are highly sensitive to changes in shipping rates. Importers tend to hold shipments to a minimum when rates are high and ship larger tonnages when rates are low.

TAXES AND TARIFFS

Taxes imposed by foreign governments are generally not so high as to cut the flow of ore from established areas. In negotiations for development of untouched deposits, however, taxes and duties have the effect of making development questionable or difficult.

United States tariffs on manganese ores and ferromanganese are shown in table 6.

CONSERVATION

Although some conservation is possible in the use of manganese today, it is estimated that the percentage is small. It is a fairly reasonable assumption that the consumer of manganese tries to balance the cost of excess usage against the alternative cost of some other material that might be required to assure the same efficiency. In steel making, it often may be possible to use less manganese, but such a saving in manganese would be offset in longer furnace time, with a resultant higher fuel cost, labor cost, and capital-equipment charge.

The use of alternative alloys, deriving their manganese from offgrade ores, offers some possibilities for conservation of the higher grade ores.

RESEARCH

Study is being made of the production and use of manganese ferroalloys made from offgrade ores and also of the substitution of elements other than manganese.

Although several private industry groups have done research on manganese, their efforts have been largely concerned with production of a definite single material or development of a specific property or group of properties. It has remained for the Bureau of Mines to conduct large-scale research with the objective of adding considerable quantities of manganese to domestic production.

Pioneering research was conducted in 1935 and 1936 by the Bureau on the feasibility of producing electrolytic manganese commercially, and later research for its development for commercial production was undertaken by both industry and the Bureau. It was brought into regular commercial production by industry in 1939. Hundreds of examinations and investigations have been made by the Bureau of domestic deposits and of methods for preparing the ores for industrial use.

Preparation and beneficiation of domestic ores have been carried out along three lines: Mechanical concentration, such as jigging,

TABLE 6.—*United States tariff schedule—manganese*

Item	Rates of duty in effect Nov. 9, 1954		
	Effective trade agreement rate [1]	Cuba and the Philippines	1930 Tariff Act [2]
Manganese ore (including ferruginous) or concentrates, including manganiferous iron ore, containing over 10 percent Mn.	¼ cent per pound contained Mn.	Free	1 cent per pound contained Mn.
Ferromanganese, 30 percent or more Mn:			
Containing not over 1 percent carbon	15/16 cent per pound contained Mn plus 7½ percent ad valorem.	------------	1⅞ cents per pound contained Mn plus 15 percent ad valorem.
Containing over 1 percent and less than 4 percent carbon.	15/16 cent per pound contained Mn.	------------	1⅞ cents per pound contained Mn.
Containing 4 percent or more carbon	⅝ cent per pound contained Mn.	------------	Do.

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[1] Applicable to most nations.

[2] Applicable to U. S. S. R. and certain associated countries.

flotation, and magnetic separation; hydrometallurgy, in which some chemical leaching agent is used; and pyrometallurgy, employing some form of smelting technique.

Mechanical concentration has been applied successfully to many of the country's small marginal ore deposits and may be a necessary step in the application of hydrometallurgical and pyrometallurgical processes to certain large deposits of very low grade and refractory manganese-bearing materials.

Hydrometallurgical processes are numerous, and many have been tested through the pilot-plant stage. One objection to the processes under consideration has been high cost of capital equipment. Furthermore, most of the processes developed thus far produce a chemical product that must again compete with imported high-grade ore. An exception is the production of electrolytic manganese, which is based on hydrometallurgical leach and produces manganese metal that can be sold competitively at a price high enough to make the operation profitable.

Pyrometallurgy has shown considerable promise, although it also requires high equipment costs. The recovery of manganese from basic open-hearth slag by smelting and selective oxidation has been under investigation by the Bureau of Mines, in cooperation with the American Iron and Steel Institute, and has been carried to the point where the method has been declared to be technically feasible and a semicommercial plant constructed. Besides upgrading the manganese content, the problem is to restore the proper manganese-iron ratio and to eliminate phosphorus to obtain a synthetic ore. Not only is manganese produced in a usable form, but there is every possibility that a high-iron steel-furnace melting stock, and possibly a phosphate fertilizer material, can be reclaimed to assist in making commercial application of the process possible.

The possibility of recovering manganese from open-hearth slags is of particular interest because of the large quantities of raw material (slag) that have accumulated and are being currently produced. It is estimated that the full application of the process could supply half of the United States current industrial manganese demand. No long-haul transportation of raw material is needed nor does the process involve any change in present steel-mill practice other than selective dumping of slags. It is conceivable that the process also may make possible the use of some low-grade domestic deposits that present difficult manganese-extraction problems by other means.

Pyrometallurgy also offers interesting possibilities for the direct smelting of low-grade or offgrade manganiferous materials to silicomanganese, ferromanganese, etc., and research has been and is being pursued along these lines.

INTERCHANGEABILITY OF MANGANESE ALLOYS

The use of spiegeleisen, a ferroalloy containing about 20 percent manganese, as a substitute for standard ferromanganese is well known in the steel industry. It has been estimated that, under emergency conditions, spiegeleisen could be substituted for at least 20 percent of the ferromanganese requirements. Some low-grade domestic ores are suitable for the production of spiegeleisen, but the cost of the contained manganese probably would be higher than that in ferromanganese, since the coke requirements alone would be about 4:1. Production of spiegeleisen, however, would involve a standard metallurgical process with standard metallurgical equipment.

The primary difficulty in substitution is one of bulk; about four times as much spiegeleisen would have to be added compared with ferromanganese. This would increase the chilling effect on the molten bath of steel to such an extent that the added material probably would have to be brought to a molten condition in a cupola furnace. Further, the quantities of carbon, phosphorus, and other undesirable elements would be increased, requiring longer furnace time and reduced capacity to reduce the residual quantities of these elements in the bath before manganese additions.

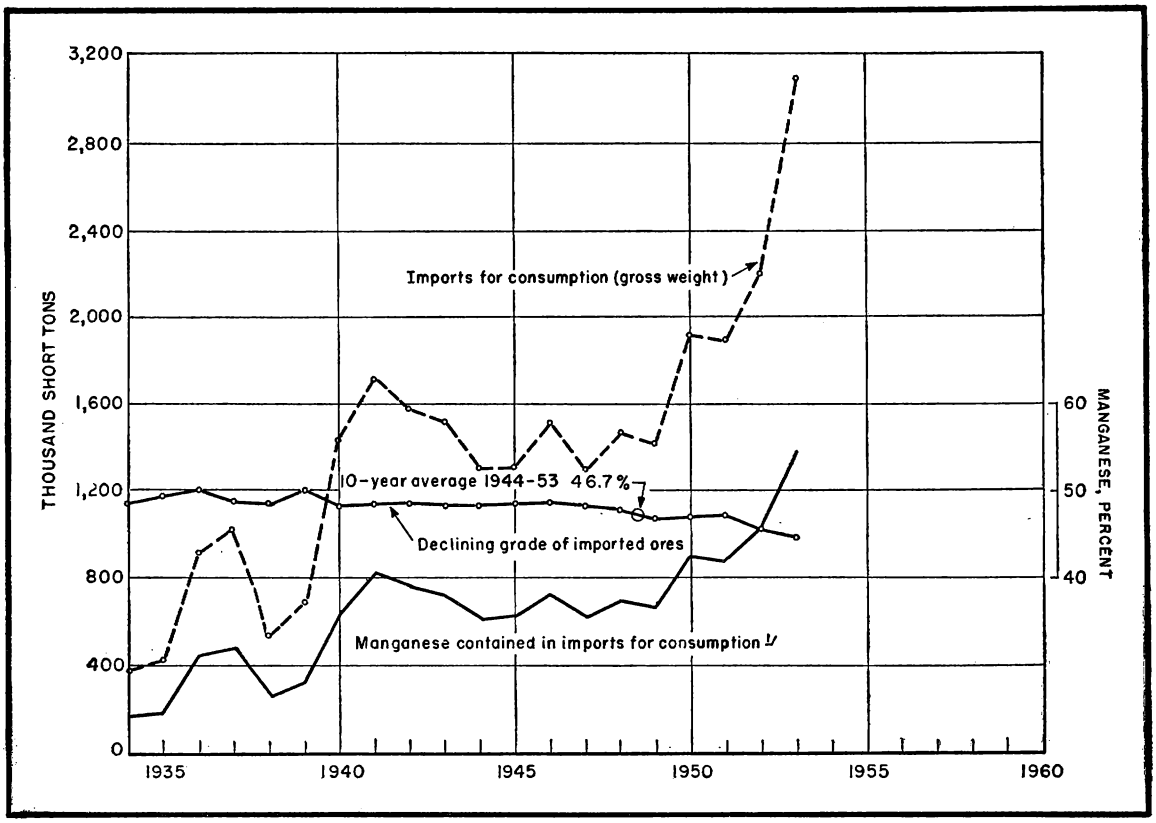
Substitution of silicomanganese for ferromanganese is possible to a certain extent and is advantageous in some killed or semi-killed steels. The production of silicomanganese increases the opportunities for utilizing certain offgrade ores, and progress is being made in electric-furnace smelting to a usable product, ores of high silica and low manganese content. An acceptable silicomanganese has been obtained by the Bureau of Mines in work of this nature.

OUTLOOK

CURRENT TRENDS AND PRACTICES

With respect to foreign ore sources, it is to be noted that producing countries are becoming interested in industrialization and conservation of resources for home use.

Probably the most significant trend in manganese has been the drop in the average grade of imported ores in recent years (fig. 2). The grade of ore necessary to produce standard ferromanganese of 78- to 82-percent manganese



[1] Thousand short tons.

FIGURE 2.—Declining Grade of Manganese Ores Imported Into the United States for Consumption, 1934–53.

content is no longer available either in industry stocks or in the present incoming supplies. Consequently, industry has been forced to lower the grade of ferromanganese to 74 to 76 percent.

This change is significant both as regards costs and the actual practice of using and producing the alloy. The lower grade ores require transportation and handling of greater quantities of gangue content. Because of the resultant larger furnace burdens necessary to produce an equivalent number of manganese units in alloy, smelting costs go up and existent furnace capacity becomes inadequate. Recovery drops due to larger losses of the metal in larger slag volumes, and industry must make closer and higher specification steels with lower grade alloys.

NEW USES

Although new uses for manganese in large quantities are not foreseen at present, prospects in this regard are unpredictable. Very little is known of the actual characteristics of metallic ingot manganese.

It is likely that increased manganese specifications in steel will develop, and substitution for other alloying elements may increase.

In the chemical field, new uses may be found at almost any time. Ordinarily, they are important in themselves but have little effect on the volume of overall consumption. Prospective new chemical uses could be virtually ignored, except for the fact that the grade of ore usually required is high and in relatively short supply.

NEW SOURCES

Prospective new sources of manganese are the natural deposits in underdeveloped areas, including French Equatorial Africa and Brazil. The same problems of control, development, and overseas shipping will apply to these sources as to the major producing areas.

Prospective new domestic sources are the large, low-grade, refractory deposits and the basic open-hearth slags. Under present technology these sources require for their development greatly advanced prices to justify heavy outlays of capital for large processing plants. It is possible, however, that byproducts and coproducts can contribute appreciable compensating credits to any operations designed to utilize these materials. Ores from the tremendous reserves of the U. S. S. R., which were recently withheld from the free-world market for a period of 2 years or more, have again entered this market. This has been the only latent source that could throw large tonnages of high-grade ore on the market in a short time at competitive prices. Its existence has had a braking effect on the development of new sources of supply.

FORECAST OF REQUIREMENTS

Projected United States steel production of 150 million short tons a year by 1975 has been used as a base for estimating future manganese-ore requirements. Assuming that the historical pattern of 35 to 40 pounds of manganese ore consumed for each ton of steel produced will continue, then approximately 3 million short tons a year appears to be the requirement by 1975.

PROBLEMS

The problem facing the domestic manganese industry is basically that of procuring large enough quantities of good-grade manganese ore at reasonable prices; at the same time, any sizable reserve of good-grade domestic ores upon which to draw is lacking. A closer analysis of the problem is obtained by separating it into the three factors of quantity, quality, and price. These are complicated by interests of the consuming and producing portions of the industry.

The demand for manganese rises with the production of steel; consequently, new sources of raw material are needed at an ever-increasing rate to pace furnace consumption as the country's economy expands.

Experience has demonstrated that the need for larger reserves is met and supplied by lower grades of raw material. This decreased quality of the material supplied accentuates the quantity factor. Reduction in grade requires the mining, movement, and processing of even greater tonnages of material to supply the same quantity of metal, resulting in higher capital investment and costs.

Thus the third factor, price, is introduced. Increases in price do not make any change in the aggregate quantities of existing manganese but do add to the availability of commercial reserves.

Adequate data are not available on the character and extent of the country's deposits of manganiferous materials, and the information should be adequately cataloged to guide present and future investigations.

The fact that the United States is more than 90 percent dependent upon foreign sources in underdeveloped areas, some with strong nationalistic ambitions, presents a set of problems involving foreign taxation, restrictions on trade, and the industrial demands of countries other than the United States.

Another problem is the procurement of adequate statistical data concerning foreign manganese sources, production, and distribution, as they concern both ore and alloy; as a proper attack upon the entire manganese problem calls for the existence of adequate statistical data and background information pertaining to all its phases.

In time of war, a serious problem of protecting shipping lanes and plant facilities exists.

In times of peace, provision must be made for future emergencies. Consideration must be given to the acquisition of stockpiles adequate to assume the import load during an emergency or until other sources of supply can take over. The latter requires the development of new deposits in relatively short time with large expenditures of capital. Determination of stockpile policy is a major problem.

Finally, there is the problem of drastic market fluctuations, which restrict the expansion of manganese-ore production.

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MERCURY

By

Helena M. Meyer[1]

UNUSUAL physical and chemical properties give mercury (quicksilver) an industrial and military importance out of proportion to the small physical size of its production industry. The concentration of sources in a very few areas, principally foreign, makes the metal an attractive product for speculation and cartel manipulation. United States producers are at a disadvantage in competitive position because the domestic deposits are characteristically small and low grade.

Summary

The mercury industry is a small one. The sources are relatively limited, and the dominant world producers—in Spain and Italy—have for lengthy periods operated through a cartel. The nature of consumption is such that demand tends to be erratic. Certain uses require substantial quantities for new installations, for example, and only minor quantities for operation of the plants. The foregoing characteristics and the speculation that is likely to accompany them lead to violent fluctuations in prices for the metal. These widely fluctuating prices discourage production in the United States, where the grade of ore produced is low and production costs are high and handicap consumers as well. Government action, through stockpiling, tariff, floor prices, exploration assistance, or other means, has been taken to assure United States supplies of this strategic commodity.

The uses of mercury are widely diversified, based upon its properties of liquidity at ordinary temperatures, high specific gravity and electrical conductivity, the toxic qualities of the metal and compounds, and other characteristics. Although substitutes are known or could be developed for most of the uses, mercury's efficiency and relative cost, as well as industrial habit, prevent large-scale substitution except under conditions of severe and continued shortage. If the alltime record prices of 1954 continue, substitution doubtless will be encouraged.

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[1] Assistant chief, Branch of Base Metals, Bureau of Mines.

BACKGROUND

SIZE OF INDUSTRY

The mercury industry in the United States and the world is a small one, as determined by quantity produced, value of production, and number of producers. It regularly ranks lower than 40th in value in relation to other minerals and fuels produced in the United States. In the post-World War II period, the maximum number of domestic producers in any year was 51, and many of them produced only a few flasks. Only two producers (New Idria Mining & Chemical Co. and Buckman Laboratories, Inc.) in the United States can be considered vertically integrated companies. Similarly, abroad, there is little or no corporate relationship between producers and consumers.

DISTRIBUTION OF PRODUCTION

Mercury is produced in only a few countries and, except for the United States and Mexico, in only one to a few mines in each area. The United States was the outstanding source of mercury in the world for much of the latter half of the 19th century, but mines in Spain and Italy now dominate world production and with rare exceptions have done so for many years. Virtually all of the Spanish production comes from one large mine (the Almaden, Province of Ciudad Real), and Italian production is concentrated in the Monte Amiata area, where 2 large mines are preeminent. Output in the United States over a period of years has come from a relatively large number of small properties, if those that produced only a few flasks are included; as many as 33 produced over 85 percent of the total for the country in 1941, and altogether 197 produced some mercury. In 1953, 10 producers out of a total of 48 supplied 96 percent of the United States total. California is the largest source by far in the United States, both in the alltime total and in recent production; important quantities have come since 1940 from Nevada, Oregon, and Idaho and smaller quantities from Alaska, Arizona, Arkansas, Utah, Texas, and Washington. Under an adverse price-cost relationship, output in the United States dropped in 1950 to the lowest point in the 100 years. In that year only 16 properties produced any mercury at all, and all but 4 produced less than 80 flasks each.

Other large producers of mercury in the world are the U. S. S. R., Yugoslavia, and Mexico. Canada was a substantial source of mercury in World War II and can produce again under adequate stimulation.

Production by important countries is given in table 1.

TECHNOLOGY (GEOLOGY, MINING, REFINING)

Mercury ore is found in rocks of all geologic ages and all classes. The common host rocks are limestone, calcareous shales, sandstone, serpentine, chert, andesite, basalt, and rhyolite.

TABLE 1.—*Mercury produced in the world and in the principal countries, 1940–53*

[In 76-pound flasks]

(Compiled by Helen L. Hunt)

Year	United States	Canada	Mexico	Spain	Italy [1]	Yugoslavia	Other [2]	World
1940	37, 777	2, 024	11, 653	52, 214	91, 230	(1)	20, 102	215, 000
1941	44, 921	7, 057	23, 137	86, 473	94, 161	(1)	19, 251	275, 000
1942	50, 846	13, 630	32, 443	72, 288	75, 921	(1)	19, 872	265, 000
1943	51, 929	22, 240	28, 321	47, 756	58, 004	(1)	27, 750	236, 000
1944	37, 688	9, 682	26, 063	34, 349	28, 705	(1)	26, 513	163, 000
1945	30, 763	----------	16, 443	40, 694	25, 410	(1)	17, 690	131, 000
1946	25, 348	----------	11, 661	41, 801	50, 822	8, 876	15, 492	154, 000
1947	23, 244	----------	9, 700	55, 608	53, 984	9, 457	16, 007	168, 000
1948	14, 388	----------	4, 786	22, 684	38, 233	10, 936	15, 973	107, 000
1949	9, 930	----------	5, 250	32, 289	44, 527	12, 764	16, 240	121, 000
1950	4, 535	----------	3, 713	51, 808	53, 346	14, 368	15, 230	143, 000
1951	7, 293	----------	8, 064	44, 480	53, 839	14, 649	18, 675	147, 000
1952	12, 547	----------	8, 731	39, 135	55, 869	14, 620	20, 098	151, 000
1953	14, 337	----------	11, 632	45, 000	51, 373	14, 272	24, 386	161, 000

[1] Output of Idria mine (Yugoslavia) included with Italy through 1945.

[2] Chiefly U. S. S. R. for which only estimates are available.

TABLE 2.—*Mercury produced in the United States, 1940–53, by States*

[In 76-pound flasks]

Year	Alaska	Arizona	Arkansas	California	Idaho	Nevada	Oregon	Texas	Utah	Washington	Other	Total
1940	162	740	1, 159	18, 629	(1)	5, 924	9, 043	(1)	53	(1)	2, 067	37, 777
1941	(1)	873	2, 012	25, 714	(1)	4, 238	9, 032	(1)	19	(1)	3, 033	44, 921
1942	(1)	701	2, 392	29, 906	(1)	5, 201	6, 935	(1)	(1)	(1)	5, 711	50, 846
1943	786	541	1, 532	33, 812	4, 261	4, 577	4, 651	1, 769	-------	-------	-------	51, 929
1944	(1)	548	191	28, 052	(1)	2, 460	3, 159	1, 095	-------	-------	2, 183	37, 688
1945	(1)	(1)	(1)	21, 199	627	4, 338	2, 500	(1)	-------	-------	2, 099	30, 763
1946	699	95	11	17, 782	868	4, 567	1, 326	-------	-------	-------	-------	25, 348
1947	127	-------	-------	17, 165	886	3, 881	1, 185	-------	-------	-------	-------	23, 244
1948	100	-------	-------	11, 188	543	1, 206	1, 351	-------	-------	-------	-------	14, 388
1949	100	-------	-------	4, 493	-------	4, 170	1, 167	-------	-------	-------	-------	9, 930
1950	-------	-------	-------	3, 850	-------	680	5	-------	-------	-------	-------	4, 535
1951	-------	(1)	-------	4, 282	357	1, 400	1, 177	(1)	-------	-------	77	7, 293
1952	28	-------	-------	7, 241	887	3, 523	868	-------	-------	-------	-------	12, 547
1953	40	-------	-------	9, 290	(1)	3, 254	648	(1)	-------	-------	1, 105	14, 337

[1] Included under "Other".

According to Duschak and Schuette (*5*),[2] two general types of cinnabar ores can be distinguished: (1) Disseminated ore, in which the cinnabar has impregnated a more or less fine-grained or highly brecciated gangue; and (2) ores deposited in fissures and cracks in the country rock. Mercury is recovered almost entirely from the sulfide mineral cinnabar (HgS—86.2 percent mercury and 13.8 percent sulfur), although native metal occurs in some ores and mercury has been obtained from livingstonite, metacinnabarite, and others of the approximately 25 minerals known to contain mercury.

The mineral deposits that constitute the world's largest known mercury mine—the Almaden in Spain—comprise three nearly parallel ore bodies, trending east and west and standing vertically, each consisting of a portion of a bed of quartzite impregnated with cinnabar. The beds of quartzite are separated by beds of slate. Along the south side of the ore bodies and their immediate enclosing rocks is a layer of breccia of peculiar character, locally called frailesca. The rocks with which the ore is associated are all Silurian.

The Monte Amiata district, which contains Italy's largest mines, is characterized geologically, according to Eckel (*6*), by a thick series of sedimentary rocks that range from Triassic to Eocene in age and are thickly capped on Monte Amiata itself by extrusive trachyte. Bodies of travertine and landslide detritus, younger than the trachyte, are rather widespread. The structure is complex and poorly understood, but most of the ore bodies seem to lie along or near a major northtrending fault or on associated cross fractures. Cinnabar is the only important ore mineral, and clay minerals are by far the most abundant alteration products in the sedimentary rocks. A southward increase in the proportion of stibnite, compared to cinnabar, suggests horizontal zoning.

In most mercury deposits in the United States the ore bodies tend to be small and highly irregular. Both surface and underground mining methods are used, but by far the larger part of the ore is produced by the latter. The New Almaden mine (with a depth of 2,450 feet) and the New Idria (with a depth exceeding 1,400 feet) are the deepest to be worked in the United States thus far; a few others have been worked down to 1,000 feet or more. New Idria's lowest present working level is at 1,000 feet. These two-named properties have ranked first and second in total production in the United States from the beginning of output to the present. Because mercury mining is a small-tonnage operation, mining methods requiring elaborate preparatory work are not used. In the United States mercury-mining practice differs somewhat from other mining in that very little ore is blocked out ahead of stoping, and to offset this factor more working faces are required to be maintained.

At the Abbadia San Salvatore mine, Monte Amiata district, Italy, the horizontal cut-and-fill method of stoping is used; fill is obtained from the stope, from other parts of the mine, or from the surface. Parts of some Italian mines are in sound rock which will stand indefinitely with little or no support. By far the greater parts of all mines, however, are in ground so heavy that the workings must be lined with brick, wood blocks, or stone. All workings except stopes are either circular or oval in section, so as to withstand most effectively the essentially fluid pressure of the ground.

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

Mercury is recovered from ores in furnaces and retorts. Preliminary treatment involves sorting and screening, and in a few instances milling, although the last is uncommon. Retorts are used chiefly where the grade of ore is relatively high and the deposit small, at prospects, or in conjunction with furnaces to treat soot. In Europe the bulk of the ore formerly was treated in Cermak-Spirek shaft or tile furnaces, but 2 new Herreshoff furnaces of 100 tons capacity each were installed at the Almaden mine, Spain, in 1952–53. Two new Gould rotary furnaces of 150 tons capacity each are to be installed at the Monte Amiata mine, Italy, in 1955 to augment current production from Pit and Cermak-Spirek furnaces, according to a report by Gordon I. Gould, December 1954. In the United States ores are treated chiefly in mechanical furnaces of the rotary-kiln or multiple-hearth types. In Bureau of Mines Bulletin 335, Schuette (*16*) says:

> Furnaces for treating quicksilver ores are of two kinds, direct-fired furnaces and retorts with indirect firing. * * * Unless cinnabar is heated above 300° C. little quicksilver is given off. This is the limiting temperature below which ore can be dried without danger of loss or of poisoning the operators * * * The treatment of quicksilver ore is carried on at a temperature somewhat higher, say at red heat or about 700° C., or 1,300° F. In direct-fired furnaces oxygen combines with the sulphur of the mercuric sulphide or cinnabar to form sulphur dioxide and the quicksilver is set free in the form of a vapor which condenses to the familiar liquid metal in cooling. In retorts where no oxygen from the air can enter, lime is added to take up the sulphur and so liberate the quicksilver.

Two types of mechanical furnaces are in use, the rotary-kiln and the multiple-hearth. Both types, as well as various types of retorts, are described by Schuette in the report quoted and also by Gould (*7*).

Because of the generally small size of mercury deposits and comparative simplicity of mercury furnaces and retorts, virtually every mine has its own treatment facilities, and only occasionally are mercury ores shipped for treatment even to adjacent properties. Thus the output of mercury mines is marketable metal, in contrast with the situation in the vast majority of metal-mining operations. The metal produced at the mines is pure enough for most consumption purposes.

Schuette (*17*) claims that, with an ample condensing system, stack losses should not exceed 2 to 3 percent of the metal in the furnace charge and that a well-designed plant should recover 95 percent of the furnace input. Recoveries at efficient plants currently meet this standard, but recoveries ranging down to as low as 60 percent are indicated to have been made at some locations.

RECLAIMED MERCURY

The reclamation of mercury from scrap has not appealed particularly to researchers, chiefly because mercury scrap has been available in quantities too small to have much commercial interest. Reclamation processes are known, however, and scrap has been reclaimed on a relatively small scale for many years. The quantities of scrap or dirty mercury available for treatment, however, are increasing. Scarcity of new metal and recent high prices give impetus to scrap reclamation.

MERCURY POISONING

Poisoning may occur in mining and recovering mercury and in any industry in which mercury is used, as a result of handling it or being exposed to its vapors or dust. The Bureau of Mines and many others have made studies and issued reports on the subject. Precautions that may be taken at mines, retorts, furnaces, and industrial plants are known, so that with care the mining and handling of mercury can be made relatively safe.

UNITED STATES DEPARTMENT OF THE INTERIOR EXPLORATION PROGRAM

The Strategic Minerals Development Program was inaugurated by the United States Department of the Interior through the Bureau of Mines and Geological Survey as a result of the passage of the Strategic Materials Act of 1939 and the Stockpiling Act of 1946. Mercury was on the original list of seven metals for investigation in 1939. Between July 1939 and June 1949 over 332 examinations of mercury occurrences were made, 43 deposits were explored (18 projects), and ore was developed on 38 of them; 370,000 tons of ore averaging 16.2 pounds of mercury per ton and 1,220,000 tons averaging 2.5 pounds were developed, and 285,000 tons averaging 1.6 pounds was indicated by dump sampling (*11*). The exploration was done chiefly by surface trenching and diamond drilling. In most instances information of some positive value was obtained, although only a few projects were successful in discovering commercial ore. The program for mercury contributed to production in World War II. Government exploration stimulation since the outbreak of war in Korea is outlined under Defense Program.

CONSUMPTION, USES, AND SUBSTITUTES

Before World War II and during a substantial part thereof, Germany and Japan were 2 of the world's largest consumers of mercury; the

United States and the United Kingdom were others ranking among the first 4 in importance. The collapse of the former two countries as major consumers, the availability of surplus supplies of metal in these countries under conditions of reduced demand, and the urgent need for dollars in both areas were major factors contributing to abundant postwar supplies of mercury in contrast to the stringent supply conditions applying to most strategic commodities. Both countries have resumed importation of mercury but as yet on a reduced scale.

Data on total consumption and on consumption by uses outside of the United States are not available, although the United Kingdom's figures on imports and reexports indicate that the country is consuming considerably less than half of the recent United States average annual quantity.

In the United States uses of mercury are widely diversified. During wars consumption has been largely for the same uses that furnish a market for mercury in times of peace, although the proportions are different. Consumption in the United States by uses, is shown in table 3.

Mercury is used in redistilled form for many of the same purposes that consume prime virgin metal—the output of mines. Following redistillation mercury is usually bottled in smaller containers. Redistilled mercury is used for those instruments or other items that require a higher degree of purity.

In 1944 a new dry-cell battery was developed, which used mercury as an important component, and before the war ended the expectation was that this new use would absorb more mercury than previously had ever been taken by all other uses combined. Features of the cell are its ability to stand up under high and low temperatures and high humidity, its long shelf life, its ability within its rated current range to deliver the same ampere-hours of service whether operated intermittently or continuously, and its much greater power for the size of unit used. The peak total annual consumption for all uses was over 62,000 flasks in 1945. In the postwar period the new battery consumed only a small part of the quantity anticipated for war. The use, nonetheless, has important future possibilities.

In the past 2 or 3 years a number of new plants using mercury cells for the electrolytic preparation of chlorine and caustic soda have been constructed, and capacities of some of the existing plants of this type have been expanded. Initial metal requirements for this type of plant and for plants that use mercury-vapor boilers for generating power are not dissipated; the metal may be reclaimed and reused if the plants are dismantled. Plant requirements for such new installations may range from one to several thousand flasks. Only metal actually consumed in chlorine manufacture is shown opposite this class in the table; metal taken from supplies for new chlorine and mercury boiler plant constructions is shown opposite "Other."

In the pharmaceutical field, substitutes for mercury include the sulfa drugs, iodine and other antiseptics, and other disinfectants. Lead azide and organic initiators, such as diazo-dinitro-phenol, are substitutes for mercury fulminate. Plastic paint and copper oxide paint are used to protect ship bottoms both

TABLE 3.—*Consumption of mercury in the United States, by uses, 1945, 1950–53*

[In 76-pound flasks]

Use	1945	1950	1951	1952	1953
Pharmaceuticals	11, 166	5, 996	2, 761	1, 395	1, 858
Dental preparations	537	[1] 1, 458	[1] 803	[1] 1, 027	[1] 1, 117
Fulminate for munitions and blasting caps	1, 115	289	494	337	39
Agriculture	2, 862	4, 504	7, 737	5, 886	6, 936
Antifouling paint	1, 661	3, 133	2, 500	1, 178	655
Electrolytic preparation of chlorine and caustic soda	597	1, 309	1, 543	2, 507	2, 380
Catalysts	3, 650	2, 743	2, 635	1, 048	826
Electrical apparatus	[1] 24, 468	[1] 12, 049	[1] 10, 250	[1] 8, 018	[1] 9, 630
Industrial and control instruments	[1] 3, 776	[1] 5, 385	[1] 6, 158	[1] 6, 412	[1] 5, 546
Amalgamation	205	192	154	151	200
General laboratory	337	646	524	629	1, 241
Redistilled	[1] 9, 712	[1] 7, 600	[1] 8, 776	[1] 7, 547	[1] 7, 784
Other	2, 343	3, 911	12, 513	6, 421	14, 047
Total	62, 429	49, 215	56, 848	42, 556	52, 259

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[1] A partial breakdown of the "redistilled" classification for 1945 showed that 53 percent was for electrical apparatus and 28 percent for instruments; for 1950, 53 percent was for instruments, 14 percent for dental preparations, and 21 percent for electrical apparatus; for 1951, 48 percent was for instruments, 9 percent for dental preparations, and 34 percent for electrical apparatus; for 1952, 48 percent for instruments, 5 percent for dental preparations, and 37 percent for electrical apparatus; and for 1953, 47 percent for instruments, 7 percent for dental preparations, and 36 percent for electrical apparatus.

TABLE 4.—*Estimated mercury reserves of the United States, including Alaska, as of January 1944, by States*

[In 76-pound flasks]

State	Workable at $195 a flask			Workable at $300 a flask [1]		
	Measured and indicated [2]	Inferred	Total	Measured and indicated [2]	Inferred	Total
Alaska	9, 500	14, 200	23, 700	14, 500	30, 200	44, 700
Arizona	3, 500	3, 500	7, 000	3, 500	3, 500	7, 000
Arkansas	200	3, 500	3, 700	200	3, 500	3, 700
California	72, 400	107, 400	179, 800	90, 300	160, 600	250, 900
Idaho	14, 300	3, 700	18, 000	34, 000	5, 400	39, 400
Nevada	25, 000	50, 000	75, 000	29, 000	75, 000	104, 000
Oregon	10, 000	6, 800	16, 800	13, 000	7, 800	20, 800
Texas	2, 500	3, 000	5, 500	3, 000	6, 000	9, 000
Washington	------------	500	500	------------	2, 000	2, 000
Total	137, 400	192, 600	330, 000	187, 500	294, 000	481, 500

[1] Cumulative totals.

[2] Separation of "measured" and "indicated" ore is not made because of lack of detailed information at some mines. Ore at most mines is indicated and inferred because measured ore is extracted during development.

alone and as extenders for mercury. Metal powders and porcelain replace mercury for some dental uses. For agricultural uses copper compounds, formaldehyde, and other products may be substituted in part for mercury. Other processes are available to replace the mercury used in electrolytic reduction, as a catalyst, or for the generation of power.

RESERVES

The Bureau of Mines and Geological Survey prepared data on reserves of mercury and other minerals in the United States as of January 1, 1944 (*10*). The figures were broken down to show reserves at $100 a flask, at $195, and $300, but only the second and third classes are shown here because no reserves are believed available at the first-named price. Since the date to which the reserve estimates apply, the wholesale index of the Bureau of Labor Statistics has moved up from 67.6 in 1944 to 110.1 in 1953 (1947–49=100). A price in 1953 of $300 thus would represent more nearly $185 a flask in terms of 1944 purchasing power. Moreover, since 1943, 180,000 flasks of mercury has been produced by domestic mines.

A more recent estimate (Report of the President's Materials Policy Commission, vol. II, p. 155, June 1952) gave measured and indicated reserves in ore containing 5 pounds or more of mercury as 150,000 flasks and inferred reserves of this grade as 445,000 flasks, as well as additional quantities in ore containing less than 5 pounds per ton. In the 5 years 1949–53, inclusive, annual output in the United States was from ores averaging 6.5 to 10.3 pounds per ton.

The source report gave reserves of commercial ore (measured only) in Spain, Italy, Canada, and Mexico as 2,000,000 ± flasks and submarginal ore as 1,750,000 flasks. With regard to these reserves, the report states:

> Only a part of the known foreign mineralized areas have been explored, and it is expected that enough new ore (of grade comparable to that now being mined and richer than in the larger United States mines) will be discovered to extend known reserves two to five times . . .

PRICES AND COSTS

Average quoted prices of mercury at New York and London, 1940–53, inclusive, and the excess or deficit of New York as compared with London are given in table 5. The large excess

TABLE 5.—*Average yearly prices per flask (76 pounds) of mercury at New York and London, and excess of New York price over London, 1940–53*

	New York [1]	London [2]	Excess of New York over London [3]
1940	$176. 87	$201. 10	[4] $24. 23
1941	185. 02	194. 20	[4] 9. 18
1942	196. 35	227. 87	[4] 31. 52
1943	195. 21	281. 44	[4] 86. 23
1944	118. 36	281. 44	[4] 163. 08
1945	134. 89	242. 45	[4] 107. 56
1946	98. 24	120. 39	[4] 22. 15
1947	83. 74	73. 02	10. 72
1948	76. 49	62. 35	14. 14
1949	79. 46	73. 28	6. 18
1950	81. 26	61. 94	19. 32
1951	210. 13	203. 37	6. 76
1952	199. 10	194. 89	4. 21
1953	193. 03	192. 49	. 54

[1] Engineering and Mining Journal, New York.

[2] Quotations in pounds sterling converted to dollars.

[3] Imports into the United States are subject to a duty of $19 a flask.

[4] London excess.

of London prices (dollar equivalents) over New York prices in the early years shown, despite the United States tariff of $19, was caused by the abnormal marketing conditions that prevailed during those years. Prices on both markets were subject to Government controls for most if not all of the World War II period. There was a return to the normal New York excess in 1947, but the excess has been substantially under the United States tariff of $19 a flask. The United States price for mercury was subject to Government control for most of 1951.

Very little information is available on costs of production either in the United States or abroad; but for comparative purposes it is noteworthy that recent production in the United States was from ore averaging less than 0.5 percent mercury, the Italian grade was about twice that in the United States, and the grade at the Almaden mine, Spain, 3 to 5 times that in Italy. Even with more efficient operation United States mines could offset, only to a small degree, the grade advantages at Spanish and Italian properties and the lower wages paid in those countries.

S. H. Williston reported in December 1954 that the item of labor makes up half of the cost of mercury mining and that wage rates for miners were only 70 cents an hour in 1938–39, with few if any fringe benefits, but that his company was at present (December 1954) paying $1.89 an hour, with $2 a day in fringe benefits. Other costs, he said, had risen similarly, so that a price of $330 would represent only about $110 in pre-World War II buying power. The Bureau of Labor Statistics wholesale price index (1947–49=100), on the other hand, rose from only 50.1 in 1939 to 110.3 in 1954, or about doubled during the period.

Certain representations have been made on behalf of industry concerning the prices required for domestic production. In 1918 United States producers stated that the mining industry would require a stable price of about $100 a flask to produce a substantial part of United States needs. Following the increases in costs that accompanied higher levels of prices for all commodities and for labor and the high production levels of the World War II period, it was indicated in 1951 that it would take a price of above $200 a flask for a period of 2 years to induce the large American producers to reopen their mines.

Some of the large mines resumed production in 1951 when the price exceeded $200. Production increases, however, were insignificant as compared with those following the outbreak of World War II, despite the fact that the 1951 average price of $210 was the highest annual price on record; those of $199 and $193 for 1952 and 1953 were far above the range in annual prices ($118 to $196) in World War II. The reasons that output did not again respond to higher prices were partly that higher grade reserves were mined during the war period; that since then the wholesale commodity index had risen sharply, cutting the real price for the metal; and that producers, having witnessed a collapse in prices more than once in World War II, claimed to fear a recurrence of conditions that caused many to lose money in that period. As a matter of fact, the price fell to $183 a flask in October 1953; it then turned upward suddenly and spectacularly in 1954 and reached a new alltime peak of $325–$330 in October 1954. Factors that caused the scarcity of mercury for sale in quantity lots and the attendant sharp price rise involved to some extent classified activities, such as stockpiling.

TARIFF

A duty of 25 cents a pound ($19 a flask) on imports of mercury has been in effect since 1922. This duty was 10 percent of the average price per flask of $193.03 in New York in 1953 but amounted to 24 percent of the average price in 1949, the last full year before the outbreak of war in Korea.

SPANISH-ITALIAN CARTEL

Mercurio Europeo, a cartel of Spanish and Italian producers, was formed in 1928 when world stocks were excessive. Aims of the organization were distribution of sales, control of production, and stabilization of prices. Spanish and Italian producers controlled over 80 percent of world production at the time; sales were to be allocated 55 percent to Spain and 45 percent to Italy. As a result of the maintenance of a high price, following organization of the cartel, mining in the United States (sharply reduced following World War I) was revitalized, and in 1931 the United States produced enough for its own needs for the first time in 14 years and in addition had a surplus for export. Then the price dropped again, and the United States resumed its 20th century position as a net importer of mercury; it was, however, jolted out of this position again briefly in World War II, in response to the record-breaking prices of the early part of the war. The Spanish-Italian cartel was formally disbanded January 1, 1950, following a large purchase of Italian mercury by the United States Government with counterpart funds. Reports in the press from time to time that the cartel was to be reconstituted have been denied by the principals. Spain and Italy produced 60 percent of the total world output in 1953.

STRATEGIC CONSIDERATIONS

The United States now normally produces only a small part of its needs for mercury; in the past 6 years 74 to 93 percent of the United States new supply was imported. It will be seen from table 6 that recent supplies were obtained largely from overseas sources, an adverse factor in case of war. The longtime need to draw upon foreign sources for mercury led to establishment of a substantial strategic stockpile of mercury.

DEFENSE PROGRAM

The Defense Materials Procurement Agency (General Services Administration) and its predecessor, Defense Minerals Administration, found that the supply-requirements situation in mercury did not justify expenditure of funds under the Defense Production Act for expansion of production. The agency planned to review the situation from time to time to ascertain if new conditions called for new policy. The Government, however, included mercury with the minerals that were to get exploration aid through the Defense Minerals Exploration Administration; mercury was one of the minerals for which the Government supplied 75 percent of costs ot approved exploration projects. By the end of 1953 the DMEA had entered into 20 contracts for exploration aid to mercury companies. These involved $1,204,159, of which the Government share was $903,119. Mercury reserves have been developed at a number of locations as a result of the DMEA assistance.

Early in June 1954 the Bureau of Foreign Commerce announced new export restrictions on mercury, effective June 5. Exports were to require licenses for shipment to all destinations except Canada.

A 3-year guaranteed-price program for mercury was announced by General Services Administration early in July 1954. The announcement stated that it was designed to "broaden the mobilization base of the metal and increase its supply for defense and industrial purposes." The procurement program calls for buying a maximum of 125,000 flasks of domestic mercury and 75,000 flasks of Mexican metal, or until December 31, 1957, whichever occurs first. The guaranteed price was $225 for domestic and Mexican metal, that for the latter being duty paid. It was stated that, in addition, from time to time individual contracts with specific foreign producers, particularly those in Canada, may be consummated.

Mercury prices were controlled by the General Ceiling Price Regulation of the Office of Price Stabilization, from January 26, 1951, until this metal and some other strategic items were released from price control on August 10 of the same year.

ADEQUACY OF STATISTICAL INFORMATION

Current domestic mercury statistics are believed adequate; but if a more critical situation develops the frequency of reporting should be advanced from quarterly to monthly, and additional detailed information may be required.

More detailed and complete data on reserves in the known areas of occurrence should be compiled.

Statistics on production for all foreign areas are not adequate, and information on consumption other than in the United States is entirely lacking. Total consumption by various use classifications would be of interest and help.

TABLE 6.—*United States self-sufficiency in mercury, 1947–53*

[In 76-pound flasks]

	1947	1948	1949	1950	1951	1952	1953
United States production	23, 244	14, 388	9, 930	4, 535	7, 293	12, 547	14, 337
Imports (general):							
Italy	1, 516	4, 994	84, 628	18, 073	17, 633	26, 025	37, 827
Mexico	1, 824	4, 063	3, 506	3, 986	4, 989	7, 971	13, 637
Spain	2, 161	27, 114	2, 225	29, 439	13, 707	24, 333	28, 303
Yugoslavia	1, 500	1, 691	3, 753	5, 980	6, 525	10, 186	5, 765
Other	3, 227	3, 870	2, 806	3, 086	2, 073	171	252
Total	10, 228	41, 732	96, 918	60, 564	44, 927	68, 686	85, 784
Total new supply	33, 472	56, 120	106, 848	65, 099	52, 220	81, 233	100, 121
Domestic percent	69	26	9	7	14	15	14
Foreign percent:							
Europe	16	60	85	82	72	75	72
Other	15	14	6	11	14	10	14
Total	31	74	91	93	86	85	86

OUTLOOK

As a result of recent alltime peak prices and possibly more particularly because of the Government (GSA) floor-price program, the outlook is that there will be a substantial rise in domestic production in the near future. The Government's program extends to the end of 1957 or until 125,000 flasks of domestically mined mercury have been purchased, whichever comes first. Currently (December 1954) the market price is nearly 50 percent higher than the floor price; as a consequence, producers are not selling to GSA, or are they likely to do so if current conditions continue. Thus it would appear that the program will not be terminated until the end of 1957. It is not clear at this time that conditions at the end of 1957 will support a higher level of production.

The GSA program guarantees a floor price for Mexican metal until 75,000 flasks have been purchased or until the end of 1957. This program and the current alltime peak prices should stimulate noteworthy increases in production in that country. Even if the current production rate—about 12,000 flasks annually—is doubled there is little chance of that country's meeting the purchase maximum before the end of 1957.

With installation of modern equipment at the Almaden mine, Spain, production was expected to advance above the average rates for recent years—45,100 flasks annually in 1950–53, and Italian production will surely rise upon completion of planned capacity, as mentioned under Technology.

Maintenance of current high prices for mercury doubtless would result not only in expanded production in Spain and Italy but in other mercury-producing countries as well. The world produced 275,000 flasks in 1941 contrasted with 161,000 in 1953. The 1941 rate can be attained again and even surpassed, but this accomplishment would depend upon stimulation, such as continued high prices or, more desirable still, some assurance that demand for mercury in the future would require the larger quantities.

The near future of mercury may be said to include a wide range of possibilities from overproduction and falling prices on one hand to sharp expansion in consumption, at least temporary scarcity of supplies, and rising prices on the other. The far future, as with most other products, appears to include increased consumption and production and a generally uptrending price.

PROBLEMS

The current and continuing problems of the mercury industry are:

1. Domestic deposits contain small and low-grade reserves particularly as compared with those in Spain and Italy; United States producers cannot compete on even terms with those in Spain and Italy.
2. World production is highly concentrated, particularly in a few mines in Spain and Italy. This concentration is conducive to combined action for industry domination. Producers in Spain and Italy acted together through a cartel for most of the period from 1928 to 1950; the cartel was disbanded at the beginning of 1950.
3. The nature of uses for mercury tends to make consumption erratic.
4. Because of the foregoing, price fluctuations tend to be severe. The tendency of mercury prices to vary more sharply than those of most metal commodities is deplored by producers and consumers alike.
5. The toxic quality of the metal and its compounds is responsible for some of mercury's uses but also presents a health problem.
6. There is inadequate knowledge of substitutes that can be used with minimum or no loss of efficiency in times of insufficient supply.

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MICA

By

Waldemar F. Dietrich[1] **and Robert D. Thomson**[2]

SHEET MICA has grown in importance from its use as stove and furnace windows to a vital material for the electronic and electrical industries. Scrap and flake mica are important in the production of ground mica for the manufacture of roofing materials, wallpaper, plastics, paint, and glass-bonded mica. Few laymen are aware of the part that mica plays in their daily activities and that without it their lives would be greatly changed.

Summary

Mica is the group name for nine silicate minerals that crystallize in a booklike form with exceptionally well-developed cleavage that permits splitting the natural blocks into extremely thin sheets. Muscovite, the potash mica, and phlogopite, the magnesia mica, are the important commercial minerals that are germane to this program.

The only commercial occurrence of muscovite sheet mica is in pegmatites, associated principally with feldspar and quartz and in many deposits with beryl, columbite-tantalite, lithium minerals, and others. Phlogopite is usually found with crystalline dolomite, marble, or serpentine but rarely in granitic pegmatites.

The United States has been virtually dependent on foreign sources for high-quality sheet mica, due primarily to the more extensive deposits in India and Brazil and to comparatively low labor costs there. India is the principal source of muscovite sheet mica (block, film, and splittings). Brazil is also an important exporter of high-quality muscovite block to the United States. The United States supply of phlogopite is obtained almost entirely from Madagascar and Canada. Historically, domestic production has been principally punch and circle mica, and only a small fraction of the United States requirements of high-quality mica is produced domestically. The high labor factor required to mine, process, and prepare sheet mica has restricted the domestic mica industry. Sizable quantities of high-quality mica have been produced domestically only in emergency periods through Government subsidization.

In 1944 the Federal Geological Survey estimated the domestic indicated and inferred reserves of sheet mica to be approximately 32 million pounds. These reserves are meager in proportion to the apparent consumption in 1953 of 14,412,900 pounds of block, film, and splittings.

Mica, in the form of sheets, is important to the electrical and electronic industries. The higher qualities of block and film mica, of sufficient crystal size to yield laminae 1 square inch and larger in area, are essential in the

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[1] Chief, Branch of Ceramic and Fertilizer Materials, Bureau of Mines.
[2] Commodity-industry analyst, Bureau of Mines.

manufacture of certain capacitors and electronic tubes. Lower qualities of block mica, including punch and circle mica, are used in toasters, water heaters, electric flatirons, and other electrical appliances. Mica splittings are used principally in making built-up mica (micanite), which is used as an insulating material in the electrical industry.

Scrap and flake mica, usually the dominant products of the United States mica-mining industry both in volume and value, are ground and used in manufacturing roofing materials, wallpaper, paints, plastics, and other commodities.

Research on synthetic mica, largely by the Bureau of Mines, has resulted in development of a commercially feasible method of manufacturing small crystals of synthetic mica as a potential raw material for machinable hot-pressed mica dielectrics, glass-bonded mica ceramics, reconstituted mica, and other industrial uses. Although the possible applications for synthetic mica at present are mainly where flake or ground mica can be used, further research possibly will result in the synthesis of larger mica crystals.

Recent developments by industry permit the use of factory scrap mica, and, to a limited degree, mine scrap, in making reconstituted mica. This new product, with certain superior qualities, can be used in applications now using builtup mica made from imported splittings. Further developments of reconstituted mica may relieve the United States of its dependence on remote foreign sources of splittings.

The outlook is for a vast expansion of the electronics industries that are just entering the great field of servo-mechanisms for automation of industrial processes. Mica and other dielectrics will be required in ever-increasing quantities.

To minimize the hazards of a high degree of dependence on distant foreign sources of sheet mica, the United States must encourage domestic sheet-mica production; expand the production and the area of utilization of reconstituted mica from domestic scrap; and intensify research on synthetic mica and on nonmica substitutes.

BACKGROUND

Mica is a group name for nine minerals differing in chemical composition and physical properties but characterized by perfect basal cleavage and ease of splitting into thin sheets. These minerals are muscovite, phlogopite, biotite, vermiculite, lepidolite, roscoelite, paragonite, zinnwaldite, and lepidomelane. Only muscovite (potassium mica) and phlogopite (magnesium mica) have any significant commercial value; muscovite has major importance.

This commodity chapter is concerned principally with muscovite and/or phlogopite, as indicated by the context, except as otherwise noted. The Lithium chapter considers lepidolite, and roscoelite is covered in the Vanadium chapter. Vermiculite is classed as a construction material.

In ancient times mica was marketed for decorative and medicinal purposes and to serve the purpose of glass. In modern times, before the use of electricity, it was used chiefly for glazing in stove windows, doors, and lanterns. Large quantities also were utilized in lamp chimneys and lampshades. Only fairly clear sheets with good cleavage were suitable. With introduction of electrical appliances in the latter years of the 19th century, the consumption of sheet mica, comprised of block, film, and splittings, in the electrical industry soon greatly exceeded all other uses. With the development and growth of electronics, mica became the preferred material as the dielectric in capacitors and for electronic tube spacers.

Paralleling the expansion of sheet-mica consumption, scrap and flake mica used in producing ground mica grew in importance for a wide variety of uses.

The current world production of mica can be divided into two main classifications: (1) Sheet mica and (2) scrap and flake mica, which comprise the greater part of the total quantity produced in the world. These two classes of mica differ widely in their uses and in the forms in which they are marketed; they are produced by separate divisions of the mica industry.

Certain grades and qualities of sheet mica are classified as strategic materials and are stockpiled by the United States Government. The United States is the world's largest consumer of sheet mica but produces only a small part of its requirements. Over 90 percent of the total quantity of muscovite sheet mica used in the United States is imported, principally from India and Brazil. Madagascar and Canada are the important sources of phlogopite.

In the United States scrap and flake mica, mostly from domestic sources, constitute about 90 percent of the tonnage and 12 percent of the value of total mica consumption.

GEOLOGY

Muscovite sheet mica is recovered only from pegmatites. Pegmatites are coarse-grained granites often referred to as giant granites, occur principally as dikes, and contain quartz, feldspar, and mica as the principal constituents; all may vary in size and proportions. Pegmatites also may contain garnet, tourmaline, apatite, beryl, lepidolite, columbite-tantalite, and spodumene. Feldspar constitutes by far the greatest tonnage and value of the products of domestic pegmatite mining, with scrap mica a strong second. The lithium minerals—lepidolite, amblygonite, and spodumene—have increasing importance, and some pegmatites are being mined solely for these minerals. Few domestic pegmatites are minable solely for beryl or columbite-tantalite except in emergencies or in rare instances where exceptionally high concentrations of these minerals occur.

Many pegmatite deposits are zoned, resulting in concentrations of the various minerals. One zone may contain a substantial portion of excellent-quality mica; another may be rich in beryl; another may be composed almost entirely of feldspar and quartz, etc. In mica-bearing pegmatites, muscovite mica is distributed sporadically throughout the deposit as books or crystals of various sizes and thicknesses. It has not been possible to formulate criteria that will distinguish a productive pegmatite dike from a barren one or to establish the exact location of the mica within a productive dike.

Throughout the world the occurrence of higher quality and larger grades of sheet mica is irregular and spotty. In all areas, domestic and foreign, low-quality mica is associated with high-quality mica.

LOCATION AND STRUCTURE OF DOMESTIC MICA INDUSTRY

Known mica deposits in the United States are confined to pegmatites in a few well-defined areas and have been found in 20 or more States. The largest region in the United States extends from central Virginia southward through western North and South Carolina, northern Georgia, and east-central Alabama (*12*, *19*, *20*).[3] A second region lies discontinuously in the New England States, with New Hampshire, Connecticut, and Maine each having mica-bearing pegmatites (*7*). A third region has districts confined to the Black Hills of South Dakota in the vicinity of Keystone and Custer (*23*) and in Idaho, Colorado, and New Mexico (*15*, *18*, *27*).

Commercial production of mica in the United States was begun in 1803 at the Ruggles mine, Grafton County, N. H., and until 1868 New Hampshire furnished all the domestic mica (*22*). Mica mining was begun on a large scale in North Carolina soon after 1868, and mining in New Hampshire declined (*26*). Since then, the major portion of the domestic production has come from the Spruce Pine, Franklin-Sylva, and Shelby-Hickory districts in North Carolina. The most important area has been the Spruce Pine district in Avery, Mitchell, and Yancey Counties. New Hampshire, Connecticut, Idaho, and South Dakota have been major producers.

Many of the older and larger sheet-mica mines yield feldspar as their major product, although some of them, especially in New England, are primarily engaged in producing mica. In addition to production by organized enterprises, substantial quantities of mica are mined in off-seasons by persons engaged in other occupations, particularly by farmers. During 1944, when sheet mica was in great demand, the number of mines operating in the United States ranged from 430 to 765. In 1950 fewer than 50 mines with a significant production operated regularly, although 200 to 300 mines operated intermittently. The small, part-time mining operations supply over half of the total United States output.

The preliminary processing of domestic sheet mica consists of rifting, trimming, and classification, as described later, to produce trimmed block or film mica. This work is done in small rifting shops at a mine or accessible to a group of mines. The processed mica from these small shops is transferred by truck or rail to a fabricator. Many fabricators maintain a trimming department if further preparation of the mica is necessary.

The fabricator stamps or punches trimmed mica, foreign and domestic, into specified shapes and sizes, in accordance with specifications of the ultimate consumer. This portion of the industry is highly specialized and is centered around two types of fabricators: (1) Firms that fabricate the mica to supply the electronic and electrical industry and (2) firms that manufacture electronic or electrical equipment and fabricate for own consumption. The major portion of the fabricating plants for block and film mica are principally in electrical manufacturing centers lying outside the productive mica regions, but there are several in the North Carolina mica district. Distribution of the fabricating plants in 1953 is shown in table 1.

TABLE 1.—*Fabrication of muscovite ruby and nonruby block and film mica in the United States, July–December 1953, by States*

State	Number	Quantity (pounds)
Illinois and Ohio	4	32, 062
Massachusetts, Pennsylvania, Rhode Island, and Virginia	7	1, 120, 859
New Jersey	5	145, 094
New York	8	466, 867
North Carolina	3	149, 979
Total	27	1, 914, 861

Virtually no mica splittings are produced in the United States, but a strong domestic industry has developed for the manufacture of builtup mica products from imported splittings. The domestic industry can compete with foreign producers because it is highly mechanized. In 1953 consumption of splittings was reported by 16 companies, with 5 in New York, North Carolina, and Virginia; 4 in Michigan, Ohio, and Wisconsin; 4 in Indiana and Pennsylvania; and 3 in Massachusetts and New Hampshire.

Scrap mica, not sheet mica, usually has been the dominant product of the United States mica industry, both in volume and value. The emphasis on scrap mica results from a large domestic market for ground mica. The mica-grinding plants are usually near the sources of scrap mica. In 1953, 10 plants in North Carolina reported grinding mica; 3 in Arizona; 2 in Georgia; and 1 each in Colorado, Illinois, Massachusetts, New Hampshire, New Jersey, Pennsylvania, Tennessee, Texas, and Virginia.

DOMESTIC RESOURCES

There are no measured reserves of mica in the United States. Most of the domestic mines are small and shallow. The practice has always been to mine rich pockets and zones, largely by hand methods, and to abandon a deposit as soon as it became unprofitable. Nearly all the mines are developed from surface exposures, and little or no development work is undertaken before production. Because of this haphazard mining practice, development is tantamount to production.

The industry has no record from which the production of strategic mica can be predicted, except from operations under Government subsidy. Reserves stated herein were esti-

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[3] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

mated by the Geological Survey, based on conditions prevailing in June 1944. Given comparable conditions in the future, the strategic mica recoverable from reasonably assured extensions of known deposits (indicated reserves) probably would total about 2 million pounds, distributed as follows (*4*):

	Pounds
Southeastern States	1, 270, 000
New England	575, 000
South Dakota and other Western States	355, 000

Considering that most mica mines are shallow and the deposits relatively undeveloped, that numerous potentially productive pegmatites have not been extensively explored, and that the present trend is toward the use of greater quantities of lower quality material for strategic uses, the inferred reserves were estimated to total 30 million to 40 million pounds of strategic mica from the established districts.

PROSPECTING AND MINING

Mica prospecting has peculiar problems wherein geology in its present stage does not yield conclusive help. The only practical prospecting method known to the mica industry is to sink a trial pit in a pegmatite to determine if that pegmatite is rich enough in mica to warrant development. Even in the famous Bihar mica field of India, it has been stated that only 2 or 3 percent of the pegmatite dikes are rich enough to permit development on an extensive scale (*24*).

Mining for mica involves great financial risk and is based upon current values, size, shape, and attitude of the deposits. Most sheet-mica mines are operated on a small scale. Usually, a miner follows the mica zone with little or no regard to systematic mining methods or future operations. Minimum equipment includes hand drills, picks, shovels, and explosives.

Throughout the mica districts of North Carolina and New England virtually all of the large productive mines have histories of alternate periods of activity and idleness. The chief reason for the frequent stoppages is that the margin of profit is nearly always so small that the industry is on a subsistence basis and cannot afford extensive development work or expensive equipment for operating at depth or under adverse water, ventilation, or haulage conditions. Furthermore, the very nature of mica itself handicaps systematic operation. When large books of mica are encountered by a drill bit, drilling not only becomes difficult, but the book itself is ruined by penetration of the drill. Blasting operations require light charges of explosives to avoid excessive shattering of books, yet of enough force to loosen the rock and permit separation of the mica from the enclosing rock.

In the United States and Canada, large companies made several attempts to develop and operate promising mica fields on a relatively large scale. None resulted in establishment of a permanent enterprise, although in a few instances the production substantially alleviated a shortage of the better qualities of mica for a short period (*29*). A few comparatively large properties remain that can be considered as valuable reserves capable of production on fairly short notice, in case of an extreme shortage of strategic-quality mica and disregarding normal price ranges.

PREPARATION

Preparation of sheet mica for market depends entirely on hand labor, and power-driven machinery is not used in processing crude sheet mica (*24*). Crude mica is usually in the form of crystals or rough books of mica of various shapes and sizes and is designated as "mine run," "run-of-mine," or "book" mica. The rough books are first cobbed to remove dirt, adhering fragments of quartz, feldspar, and other foreign materials, and defective mica laminae. Next, the cobbed mica is "rifted" or split into forms of suitable thicknesses, and the major flaws are removed. Products of the rifting operation are broadly classified by visual methods as (1) "untrimmed-sheet" mica and (2) "scrap" mica. After rifting, the edges of the sheet mica are trimmed by knife, sickle, shears, or fingers to remove the broken and ragged ends and thus facilitate further splitting. The "uncut" sheet mica in the "full-trimmed" condition is rifted mica trimmed on all sides with all cracks, reeves, and crossgrains removed. "Half-trimmed" mica is rifted mica trimmed on two adjacent sides, with no cracks extending into the usable area.

In the broadest sense, sheet mica is any mica other than scrap mica that is flat and free enough from structural defects, which can be punched or stamped into products of specified shapes. More specifically, sheet mica is classified as "block," "film," and "splittings," based on the thickness and quality of the mica. Block mica is knife-dressed mica not less than 0.007 inch thick, with a minimum usable area of 1 square inch. Film mica is split from the better qualities of knife-dressed mica to specified thicknesses ranging from 0.0012 to 0.004 inch. Sheets split from block and film with a maximum thickness of 0.0012 inch are known as "splittings." About 70 percent of the weight of the closely trimmed mica sheets is lost in manufacturing because of the irregular shapes of the stock. Smaller size, low-quality block mica can be subdivided further as "punch," 'circle," and "washer." Punch mica is thumb-trimmed material of large enough area to yield a circular disk of at least 1½ inches in diameter for stained-quality mica and 1¾ to 2 inches for clear-quality. Circle mica is larger than punch and should yield sheets 2 inches in diameter, and washer mica (sometimes called "small punch") will yield 1-inch sheets. Often, punch mica is used as a general term, including punch, circle, and washer.

"Scrap" mica includes mica that, because of size, color, and quality, is below specifications for sheet mica; mica obtained from pegmatite mining as a sole product or as a byproduct; and mica derived from preparation of sheet mica and from factory waste. "Flake mica" includes finely divided mica recovered from mica and sericite schists and as a byproduct of feldspar and kaolin beneficiation.

The great amount of hand labor required in producing and preparing block mica, film mica, and splittings gives countries with extremely low wages, such as India, a very great competitive advantage over those with high wage scales. In 1953 wages in India for experienced mica miners were reported to be from 1 rupee (21 cents) to 1½ rupees per day compared with a minimum of $6 per day in the North Carolina districts. Women, children, and old men in India do most of the work of fabricating and preparation for market at daily wages considerably lower than 21 cents compared with a minimum wage in the United States of $6 per day. Even in Brazil, wages in the mica-mining areas in 1953 were reportedly $0.60 to a maximum of perhaps $1.25 per day. India has the added competitive advantage over all other producing countries in that much of the low-quality and small-size material unsuitable for electronic uses is utilized in manufacturing splittings, which constitute about 90 percent of the total value of Indian mica production. It is extremely doubtful, in view of comparative labor costs, if the United States mica industry can compete in normal times with low-cost imported materials, even if domestic deposits were developed of adequate size and mica content to be amenable to low-cost mining and block-recovery methods.

Mine scrap and flake mica usually are prepared for market by dry grinding (*1*, *5*, *31*). The whitest crude mine scrap and factory scrap are preferred by producers of wet-ground mica. The procedures and machinery employed in preparing dry-ground mica vary considerably according to the personal preference of the mill operator. Usually, scrap is carried by a jet of water from the stockpile through a series of alternate roll crushers and trommel screens, which separate the mica from the gangue. The mica discharge from the last screen is lifted by a bucket elevator to a draining bin. From this bin, mica is fed by a screw feeder to a rotary oil-fired drier. The dried material, after passing through a hammer mill and over a multiple-deck vibrating screen, is bagged for shipment, usually as products ranging from 16- to 100-mesh.

Wet grinding usually is done in chaser mills, consisting of wooden rollers that rotate in a cylindrical steel tank with an end-grain wood-block bottom (*11*). Scrap mica is fed through hoppers to the grinding mill, and water is added gradually to form a stiff paste. After grinding has been completed to the desired fineness, the ground charge is sluiced from the mill into settling bins. After the gritty impurities have settled, the overflow carries the fine mica to vats, where it settles. The mica sludge then is put through a centrifuge, and drying of the pulp is completed in a steam drier. The dry mica is screened to remove heavy particles that would be injurious to the silk-cloth reels in the bolting machine and bolted to produce ground mica ranging from 160- to 325-mesh. Extremely fine ground mica is obtained in "micronizing" mills.

Wet-ground mica differs from dry-ground in that it retains its sheen, and edges of the flakes are smooth and rounded in contrast to the torn and hackled edges and abraded surfaces of the dry-ground material.

CLASSIFICATION

No other mineral product is as difficult to classify as sheet mica or approaches it in the multiplicity of qualities and grades (sizes) (*14*, *16*, *24*). The nearest approach to a generally accepted standard is the ASTM Standard Specification D-351-53T, based on the Indian system and corresponding closely to Federal Specifications No. HH-I-536 and the International Organization for Standardization's proposed methods for mica classification (*2*, *3*).

Muscovite block mica, based on the visual properties in accordance with ASTM D-351-53T and shown in table 2, is classed into 11 qualities, ranging from densely stained quality to clear quality. However, it has been customary in the United States, due to economic reasons, to classify domestically produced mica into two primary divisions—"clear" and "stained." Clear mica is subdivided into "No. 1," "No. 2," and "No. 2 inferior." Black-stained and spotted material is referred to as "stained," "electric," or "No. 3." Interpretations of visual standards differ somewhat from one observer to another, but basically visual classification depends on the degree of mineral, vegetable, and air-staining and foreign inclusions, cracks, buckles, and other similar defects. Clear mica, which is hard, of uniform color, flat, and free from all stains and other defects, is the finest quality of mica.

TABLE 2.—*Quality classification of muscovite block mica 0.007-inch minimum thickness based on visual properties*

(American Society for Testing Materials)

ASTM visua lquality classification	Clear—uniform basic color[h]	Air inclusions				Smoky stains	Light dots (mineral)	Sooty stains (mineral)	Black stains (mineral)[e]	Black stains (mineral)[g]	Red stains (mineral)[e]	Green stains (vegetable type)[f]	Clay stains	Waviness				Hardness		Stones	Buckles	Reeves	Ridges	Tears	Cracks	Hairlines	Wedge	Tangle sheet	Herringbones	Ribboned or ruled
		Very slight[a]	Slight[b]	Medium[c]	Heavy[d]									Nearly flat	Slight	Medium	Heavy	Hard	Soft											
V-1, Clear	✓	x	x	x	x	x	x	x	x	x	x	x	x	✓	x	x	x	✓	x	x	x	x	x	x	x	x	x	x	x	x
V-2, Clear and Slightly Stained	✓	✓	x	x	x	x	x	x	x	x	x	x	x	✓	x	x	x	✓	x	x	x	x	x	x	x	x	x	x	x	x
V-3, Fair Stained	✓	✓	✓	x	x	x	x	x	x	x	x	x	x	✓	✓	x	x	✓	x	x	x	x	x	x	x	x	x	x	x	x
V-4, Good Stained	✓	✓	✓	✓	x	x	x	x	x	x	x	x	x	✓	✓	✓	x	✓	x	x	x	x	x	x	x	x	x	x	x	x
V-5, Stained A Quality	✓	✓	✓	✓	✓	✓	x	x	x	x	x	✓	x	✓	✓	✓	✓	✓	x	x	x	x	x	x	x	x	x	x	x	x
V-6, Stained B Quality	✓	✓	✓	✓	✓	✓	x	x	✓	x	✓	✓	✓	✓	✓	✓	✓	✓	x	x	✓	x	x	x	x	x	x	x	x	x
V-7, Heavy Stained	✓	✓	✓	✓	✓	✓	x	x	✓	x	✓	✓	✓	✓	✓	✓	✓	✓	S	x	✓	x	✓	x	x	x	x	x	x	x
V-8, Black Dotted	✓	✓	✓	✓	✓	✓	x	x	✓	x	✓	✓	✓	✓	✓	✓	✓	✓	S	x	✓	x	✓	x	x	x	x	x	x	x
V-9, Black Spotted	✓	✓	✓	✓	✓	✓	✓	✓	✓	x	✓	✓	✓	✓	✓	✓	✓	✓	S	x	✓	✓	✓	x	x	x	x	x	x	x
V-10, Black Stained	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓	S	✓	✓	✓	✓	✓	✓	✓	✓	x	✓	✓
V-11, Densely Stained	✓	✓	✓	✓	✓	✓	✓	✓	✓	x	x	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓	✓

Symbols:
✓—Permissible.
x—Not permissible.
S—Permissible only if specified.
[a] Few and tiny in one-fourth of usable area.
[b] In one-half usable area.
[c] In two-thirds usable area.
[d] Uniformly distributed.
[e] Slight.
[f] Medium.
[g] Heavy.
[h] Crystallographic discoloration is permitted to a limited extent in V-4 Good-Stained quality. It is not permitted in qualities higher than V-4 Good-Stained. In qualities below V-4 Good-Stained, it is permitted without limit.

NOTE 1.—The visual properties of block mica usually are judged under the following light conditions:
For stains and inclusions.—Transmitted daylight or equivalent.
For air inclusions.—Reflected daylight or equivalent.
For waves, buckles, ridges, etc.—Reflected daylight or equivalent where distortion of parallel and vertical lines of reflected image such as a window frame can be judged.

NOTE 2.—The hardness or mechanical properties of block mica are usually judged by a sharp, clear ring when mica is dropped on a hard surface.

NOTE 3.—Muscovite mica occurs in various colors which are more pronounced the thicker the block. Some typical colors are ruby, white, light green, rum, etc.

NOTE 4.—Cracks, tears, stones or pinholes are judged by transmitted daylight or equivalent.

NOTE 5.—Rigidity is judged by the relative stiffness when flexing with the fingers.

Muscovite film is classified under three quality designations: First quality, equivalent to Fair-Stained block mica; second quality, equivalent to Good-Stained block; and third quality, equivalent to Stained block.

A system of classifying the quality of block and film mica for fixed mica-dielectric capacitors, based on a combination of the electrical and physical properties of the mica, is covered by ASTM D-748-52T. The specifications provide for the use of a Q-meter for determining the reciprocal of the power factor. A simple spark test may be used to detect conducting stains.

Trimming operations result in irregular mica, and mica grades or sizes refer to the dimensions of the largest rectangle of usable mica that can be cut from a piece of mica. Sizes are universal, but several countries use different designations (table 3). In accordance with ASTM specifications, block and film are classed into 12 grade groups, ranging from No. 6 (1 inch square) to OOEE Special (100 square inches). The minimum strategic grade is No. 6.

Mica also is classified according to color. There is no sharp division between color groups, which range from pinkish buff and drab through shades of brown and green to pale yellowish green. The Geological Survey has arranged mica into 7 main color categories and 3 commercial groups, as shown below (*19*):

Main color categories	*Commercial groups*
Pinkish buff and drab	
Cinnamon brown	Ruby.
Brown	
Brownish olive	Rum.
Yellowish olive	
Yellowish green	Green.
Green	

The National Bureau of Standards has established two color categories of mica, namely, ruby and nonruby. These standards are used by the Government for purchasing mica for the National Stockpile. The terms "rum" and "green" mica are commonly used by industry, however.

Ruby mica is the preferred color because its Q factor is fairly constant within acceptable ranges. The Q factor of nonruby, especially green mica, may vary considerably; therefore there is strong reluctance to accept nonruby mica as a substitute for ruby mica in electronic applications. Nonruby mica from a specific

TABLE 3.—*Grade classification of muscovite block and film mica*

Area of minimum rectangle (square inches)	Grade (size) classification					Minimum dimensions on one side (inches)
	ASTM	United States (domestically produced mica)	Madagascar	India	Brazil	
100	OOEE Special	8 x 12	0000	OOEE Special	OEE Special	4
80	OEE Special	8 x 10	000	OEE Special	EE Special	4
60	EE Special	6 x 10	00	EE Special	E Special	4
48	E Special	6 x 8	0	E Special	Special	4
36	A1 Special	6 x 6	A1	Special	A1	3½
24	1	4 x 6	1	1	1	3
15	2	3 x 5	2	2	2	2
10	3	3 x 3 and 3 x 4	3	3	3	2
6	4	2 x 3	4	4	4	1½
3	5	1½ x 2 and 2 x 2	5	5	5	1
2¼	5½	Circle	6	5½	5½	⅞
1	6	Punch	6	6	6	¾

source may be of uniformly high quality but may vary from piece to piece or mine to mine, and general acceptance of nonruby mica is considered hazardous without adequate electrical testing. Research is needed to determine the usability of nonruby mica in an effort to conserve a potential source of strategic mica.

PRODUCTION

The United States is the largest consumer of mica and the world's largest producer when all classes of mica are considered. However, the United States produces only a small fraction of its requirements of high-quality sheet mica. Historically, punch and circle micas, used principally in household electric appliances and electric equipment controls, have been the dominant sheet-mica products in volume. Unfortunately, the electrical and electronic industries, which are so essential to the highly mechanized way of life in the United States, must rely upon foreign sources for most of their sheet-mica supply. The salient statistics of the United States mica industry from 1936 to 1953 are given in table 4, and specific data for sheet, scrap, and flake mica sold or used in the United States from 1920–53 in table 5.

IMPORTS

The United States is virtually dependent on foreign sources for sheet mica (*30*). Most of the muscovite sheet-mica imports into the United States have come from India, the principal source since 1885. In recent years Brazil has become an important source of high-quality muscovite mica. The United States supply of phlogopite is obtained principally from Madagascar and Canada.

Imports of muscovite block, film, and splittings and phlogopite splittings compiled by the United States Tariff Commission from official documents of the United States Bureau of Customs, by quality and principal source, for 1952–53 are given in tables 6 and 8. Before 1952 such information was not tabulated, and a method for estimating sheet-mica imports was determined from figures compiled by the United States Department of Commerce (table 7). Tariff and Commerce data are compared in table 8.

Most mica imports are made through established importers. New York is the principal port of entry for imported mica, with substantial quantities also reaching Newport News, Va.; and Boston, Mass.

USES

Such characteristics as high dielectric strength, low power-factor losses, flexibility, transparency, perfect cleavage, and low unit cost make mica usable in many industries and in many diversified applications. Sheet mica is important in the electrical and electronic industries as single pieces split from individual crystals or as composite pieces in a built-up form. The higher qualities of block and film mica are essential as dielectrics in the manufacture of certain capacitors and electronic tubes. Lower qualities of block mica, including punch and circle mica, are used as electrical insulation in flatirons, toasters, and other electrical appliances. Data on the fabrication of muscovite ruby and nonruby block and film mica in the United States during July–December 1952 and 1953 by end-product use are shown in table 9.

Civilian peacetime requirements place the emphasis on the types of mica suitable for use

TABLE 4.—*Salient statistics of the mica industry in the United States, 1936–53*

Mica	1936	1937	1938	1939	1940	1941	1942	1943	1944
Domestic mica sold or used by producers:									
Total sheet:									
Pounds	1,319,233	1,694,538	939,507	813,708	1,625,437	2,666,453	2,761,844	3,448,199	1,523,313
Value	$203,879	$285,244	$139,333	$138,963	$291,685	$566,858	$725,030	$3,228,742	$3,262,711
Average per pound	$0.15	$0.17	$0.15	$0.17	$0.18	$0.21	$0.26	$0.94	$2.14
Scrap and flake mica:									
Short tons	20,955	25,196	20,257	24,672	22,386	32,500	43,262	46,138	51,727
Value	$260,594	$354,737	$256,382	$311,895	$314,565	$442,789	$671,165	$738,025	$1,089,072
Average per ton	$12.44	$14.08	$12.66	$12.64	$14.05	$13.62	$15.51	$16.00	$21.05
Total sheet, scrap, and flake mica:									
Short tons	21,615	26,043	20,727	25,079	23,199	33,833	44,643	47,862	52,489
Value	$464,473	$639,981	$395,715	$450,858	$606,250	$1,009,647	$1,396,195	$3,966,767	$4,351,783
Total ground:									
Short tons	25,585	27,245	27,086	30,924	27,984	43,419	46,979	51,582	52,713
Value	$722,416	$839,812	$924,554	$1,156,333	$1,016,628	$1,532,351	$1,653,358	$1,990,144	$1,914,709
Consumption of splittings:									
Pounds	3,518,058	4,347,435	1,667,806	3,423,044	4,918,861	7,297,628	6,636,639	8,413,362	8,816,965
Value	$846,393	$1,257,645	$612,465	$1,089,683	$1,725,522	$2,832,939	$2,835,421	$3,518,822	$4,657,730
Imports for consumption:									
Total sheet:									
Pounds	860,253	1,004,950	391,125	902,598	1,534,188	2,016,852	3,244,857	5,501,745	5,032,983
Value	$239,378	$296,235	$113,403	$271,072	$576,565	$1,119,584	$2,141,465	$6,313,900	$3,921,078
Scrap:									
Short tons	3,893	6,723	4,450	4,279	3,061	1,251	2,179	2,048	2,412
Value	$22,666	$36,355	$28,590	$29,493	$22,611	$12,791	$25,879	$27,102	$32,688
Total sheet and scrap:									
Short tons	4,323	7,226	4,646	4,730	3,828	2,259	3,801	4,799	4,929
Value	$262,044	$332,590	$141,993	$300,565	$599,176	$1,132,375	$2,167,344	$6,341,002	$3,953,766
Manufactured:									
Short tons	2,355	4,113	1,115	1,550	3,860	6,041	7,493	8,960	2,314
Value	$943,524	$1,735,009	$522,426	$758,745	$1,884,952	$3,282,656	$6,860,434	$8,513,064	$3,707,718
Total imports:									
Short tons	6,678	11,339	5,761	6,280	7,688	8,300	11,294	13,759	7,243
Value	$1,205,568	$2,067,599	$664,419	$1,059,310	$2,484,128	$4,415,031	$9,027,778	$14,854,066	$7,661,484
Exports (all classes of mica):									
Short tons	1,478	1,795	1,772	1,827	903	1,163	1,001	693	619
Value	$170,011	$216,858	$183,889	$226,364	$191,550	$280,810	$303,526	$653,889	$526,824

Mica	1945	1946	1947	1948	1949	1950	1951	1952	1953
Domestic mica sold or used by producers:									
Total sheet:									
Pounds	1,298,587	1,078,867	415,589	270,042	513,994	578,818	594,884	697,989	849,394
Value	$737,342	$217,955	$116,110	$45,940	$132,097	$125,928	$160,322	$908,135	$2,153,584
Average per pound	$0.57	$0.20	$0.28	$0.17	$0.26	$0.22	$0.27	$1.30	$2.54
Scrap and flake mica:									
Short tons	41,060	53,602	49,797	52,157	32,856	69,360	71,871	75,236	73,259
Value	$812,322	$1,041,423	$1,095,578	$1,091,698	$795,782	$1,742,616	$1,884,087	$1,954,286	$1,823,840
Average per ton	$19.78	$19.43	$22.00	$20.93	$24.22	$25.12	$26.21	$25.97	$24.90
Total sheet, scrap, and flake mica:									
Short tons	41,709	54,141	50,005	52,292	33,113	69,650	72,168	75,585	73,684
Value	$1,549,664	$1,259,378	$1,211,688	$1,137,638	$927,879	$1,868,544	$2,044,409	$2,862,421	$3,977,424
Total ground:									
Short tons	51,806	62,113	64,540	64,642	56,393	72,250	70,122	74,806	73,072
Value	$1,995,969	$2,516,018	$2,967,713	$3,232,632	$2,860,956	$3,935,697	$3,842,628	$4,278,103	$4,192,420
Consumption of splittings:									
Pounds	7,897,402	7,815,989	9,309,981	7,917,365	8,114,804	10,783,198	13,379,295	10,220,671	10,346,159
Value	$3,415,696	$4,259,478	$6,680,753	$6,300,581	$7,096,365	$8,631,421	$11,760,617	$9,729,099	$7,902,232
Imports for consumption:									
Total sheet:									
Pounds	4,208,130	4,499,672	1,754,419	2,829,335	2,466,546	3,334,652	3,563,242	2,481,669	2,599,007
Value	$4,148,737	$2,288,897	$1,150,958	$2,477,598	$2,111,095	$3,094,616	$3,855,063	$3,520,922	$4,279,273
Scrap:									
Short tons	3,612	6,207	5,109	7,124	1,758	4,402	5,885	6,531	3,927
Value	$41,950	$75,846	$66,408	$107,540	$21,740	$59,014	$93,357	$106,475	$72,100
Total sheet and scrap:									
Short tons	5,716	8,457	5,986	8,539	2,991	6,069	7,667	7,772	5,226
Value	$4,190,687	$2,364,743	$1,217,366	$2,585,138	$2,132,835	$3,153,630	$3,948,420	$3,627,397	$4,351,373
Manufactured:									
Short tons	3,695	5,487	5,699	9,357	9,747	12,441	11,250	5,276	5,763
Value	$2,173,133	$4,754,583	$6,251,613	$12,960,918	$17,212,419	$20,506,774	$18,568,148	$11,053,579	$10,911,101
Total imports:									
Short tons	9,411	13,944	11,685	17,896	12,738	18,510	18,917	13,048	10,989
Value	$6,363,820	$7,119,326	$7,468,979	$15,546,056	$19,345,254	$23,660,404	$22,516,568	$14,680,976	$15,262,474
Exports (all classes of mica):									
Short tons	981	1,542	1,493	1,403	1,108	1,547	1,894	2,472	2,402
Value	$377,473	$709,109	$970,326	$720,359	$676,752	$859,796	$1,101,917	$911,076	$1,109,865

TABLE 5.—*Mica sold or used by producers in the United States, 1920–53*

Year	Sheet mica						Scrap mica [2]		Flake mica [3]		Total scrap and flake mica		Grand total	
	Uncut mica larger than punch and circle		Uncut punch and circle mica [1]		Total sheet mica									
	Pounds	Value	Pounds	Value	Pounds	Value	Short tons	Value	Short tons	Value	Short tons	Value	Short tons	Value
1920	529, 940	$431, 202	1, 153, 540	$115, 770	1, 683, 480	$546, 972	5, 723	$167, 017	(4)	(4)	5, 723	$167, 017	6, 565	$713, 989
1921	66, 058	50, 997	675, 787	67, 516	741, 845	118, 513	2, 577	56, 849	1, 200	$37, 262	3, 777	94, 111	4, 148	212, 624
1922	166, 535	123, 174	911, 433	71, 127	1, 077, 968	194, 301	6, 641	114, 045	913	23, 157	7, 554	137, 202	8, 093	331, 503
1923	269, 430	176, 225	1, 793, 749	134, 955	2, 063, 179	311, 180	8, 054	129, 695	1, 505	65, 484	9, 559	195, 179	10, 591	506, 359
1924	147, 450	97, 282	1, 313, 447	114, 753	1, 460, 897	212, 035	4, 709	87, 242	3, 429	56, 154	8, 138	143, 396	8, 868	355, 431
1925	1, 003, 682	246, 434	790, 183	75, 528	1, 793, 865	321, 962	9, 695	173, 537	2, 665	64, 544	12, 360	238, 081	13, 257	560, 043
1926	329, 189	172, 131	1, 842, 970	228, 053	2, 172, 159	400, 184	7, 043	136, 643	2, 434	70, 000	9, 477	206, 643	10, 563	606, 827
1927	197, 750	117, 471	1, 314, 742	95, 011	1, 512, 492	212, 482	6, 430	119, 883	3, 107	48, 595	9, 537	168, 478	10, 293	380, 960
1928	213, 295	140, 025	1, 468, 482	90, 931	1, 681, 777	230, 956	9, 557	159, 788	3, 521	53, 079	13, 078	212, 867	13, 919	443, 823
1929	283, 084	187, 332	1, 752, 044	98, 989	2, 035, 128	286, 321	6, 253	117, 901	8, 595	71, 545	14, 848	189, 446	15, 866	475, 767
1930	211, 703	116, 077	1, 253, 782	61, 230	1, 465, 485	177, 307	6, 732	109, 100	2, 183	42, 031	8, 915	151, 131	9, 648	328, 438
1931	205, 306	78, 513	757, 647	33, 317	962, 953	111, 830	6, 621	99, 415	2, 727	22, 722	9, 348	122, 137	9, 829	233, 967
1932	80, 485	37, 906	258, 512	7, 976	338, 997	45, 882	7, 040	83, 777	3, 626	38, 380	10, 666	122, 157	10, 835	168, 039
1933	111, 297	42, 980	253, 243	10, 199	364, 540	53, 179	8, 751	98, 159	4, 729	61, 280	13, 480	159, 439	13, 662	212, 618
1934	158, 372	74, 172	425, 156	16, 096	583, 528	90, 268	7, 719	99, 791	4, 218	66, 831	11, 937	166, 622	12, 229	256, 890
1935	266, 306	132, 763	670, 327	28, 387	936, 633	161, 150	12, 185	132, 606	6, 667	111, 345	18, 852	243, 951	19, 320	405, 101
1936	300, 773	155, 493	1, 018, 460	48, 386	1, 319, 233	203, 879	12, 697	133, 251	8, 258	127, 343	20, 955	260, 594	21, 615	464, 473
1937	381, 638	214, 751	1, 312, 900	70, 493	1, 694, 538	285, 244	14, 640	205, 100	10, 556	149, 637	25, 196	354, 737	26, 043	639, 981
1938	165, 386	93, 767	774, 121	45, 566	939, 507	139, 333	13, 707	169, 780	6, 550	86, 602	20, 257	256, 382	20, 727	395, 715
1939	147, 953	99, 756	665, 755	39, 207	813, 708	138, 963	14, 661	202, 996	10, 011	108, 899	24, 672	311, 895	25, 079	450, 858
1940	220, 132	175, 598	1, 405, 305	116, 087	1, 625, 437	291, 685	12, 712	176, 417	9, 674	138, 148	22, 386	314, 565	23, 199	606, 250
1941	324, 216	359, 911	2, 342, 237	206, 947	2, 666, 453	566, 858	16, 917	257, 303	15, 583	185, 486	32, 500	442, 789	33, 833	1, 009, 647
1942	336, 199	442, 130	2, 425, 645	282, 900	2, 761, 844	725, 030	22, 781	355, 358	20, 481	315, 807	43, 262	671, 165	44, 643	1, 396, 195
1943	757, 116	2, 754, 787	2, 691, 083	473, 955	3, 448, 199	3, 228, 742	22, 025	423, 174	24, 113	314, 851	46, 138	738, 025	47, 862	3, 966, 767
1944	619, 569	3, 108, 566	903, 744	154, 145	1, 523, 313	3, 262, 711	29, 620	603, 505	22, 107	485, 567	51, 727	1, 089, 072	52, 489	4, 351, 783
1945	118, 357	568, 040	1, 180, 230	169, 302	1, 298, 587	737, 342	26, 014	487, 807	15, 046	324, 515	41, 060	812, 322	41, 709	1, 549, 664
1946	91, 850	91, 909	987, 017	126, 046	1, 078, 867	217, 955	38, 405	750, 883	15, 197	290, 540	53, 602	1, 041, 423	54, 141	1, 259, 378
1947	68, 756	68, 603	346, 833	47, 507	415, 589	116, 110	35, 199	709, 745	14, 598	385, 833	49, 797	1, 095, 578	50, 005	1, 211, 688
1948	53, 248	22, 012	216, 794	23, 928	270, 042	45, 940	(5)	(5)	(5)	(5)	52, 157	1, 091, 698	52, 292	1, 137, 638
1949	63, 159	59, 521	450, 835	72, 576	513, 994	132, 097	24, 942	526, 268	7, 914	269, 514	32, 856	795, 782	33, 113	927, 879
1950	32, 385	39, 253	546, 433	86, 675	578, 818	125, 928	58, 250	1, 401, 411	11, 110	341, 205	69, 360	1, 742, 616	69, 650	1, 868, 544
1951	50, 838	51, 893	544, 046	108, 429	594, 884	160, 322	59, 514	1, 475, 059	12, 357	409, 028	71, 871	1, 884, 087	72, 168	2, 044, 409
1952	72, 496	790, 238	625, 493	117, 897	697, 989	908, 135	57, 201	1, 452, 174	18, 035	502, 112	75, 236	1, 954, 286	75, 585	2, 862, 421
1953	182, 153	2, 055, 574	667, 241	98, 010	849, 394	2, 153, 584	49, 215	1, 305, 134	24, 044	518, 706	73, 259	1, 823, 840	73, 684	3, 977, 424

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[1] Includes small quantities of splittings in certain years.

[2] Includes the mica, except sheet mica, obtained from pegmatite mining as a sole product or as a byproduct.

[3] Includes finely divided mica recovered from mica and sericite schist and as a byproduct of feldspar and kaolin beneficiation.

[4] Separate figures for flake mica not available.

[5] Figure withheld to avoid disclosure of individual company operations.

TABLE 6.—*Muscovite block and film mica, United States general imports, 1952–53, by quality and principal source* [1] [2]

[Pounds]

Quality	Total		Country					
			India		Brazil		Other	
	1952	1953	1952	1953	1952	1953	1952 [3]	1953 [4]
Block:								
Good-Stained and Better	352, 237	527, 967	117, 589	298, 387	197, 459	191, 402	37, 189	38, 178
Stained	2, 019, 543	2, 395, 272	599, 870	1, 189, 206	1, 389, 055	1, 135, 899	30, 618	70, 167
Heavy-Stained	573, 227	989, 576	37, 933	276, 062	524, 757	675, 425	10, 537	38, 089
Lower	665, 601	551, 353	118, 291	236, 541	510, 006	296, 753	37, 304	18, 059
Total	3, 610, 608	4, 464, 168	873, 683	2, 000, 196	2, 621, 277	2, 299, 479	115, 648	164, 493
Film:								
First-quality	101, 394	116, 212	101, 363	116, 210			31	2
Second-quality	196, 809	329, 426	196, 360	327, 003	330	1, 267	119	1, 156
Other-quality	4, 167	5, 176	3, 932	4, 976	235	200		
Total	302, 370	450, 814	301, 655	448, 189	565	1, 467	150	1, 158
Block and film:								
Good-Stained and Better [5]	650, 440	973, 605	415, 312	741, 600	197, 789	192, 669	37, 339	39, 336
Stained [6]	2, 023, 710	2, 400, 448	603, 802	1, 194, 182	1, 389, 290	1, 136, 099	30, 618	70, 167
Heavy-Stained	573, 227	989, 576	37, 933	276, 062	524, 757	675, 425	10, 537	38, 089
Lower	665, 601	551, 353	118, 291	236, 541	510, 006	296, 753	37, 304	18, 059
Total	3, 912, 978	4, 914, 982	1, 175, 338	2, 448, 385	2, 621, 842	2, 300, 946	115, 798	165, 651

[1] Compiled by the U. S. Tariff Commission from official documents of the U. S. Bureau of Customs.

[2] Does not include imports of mixed grades and qualities from Angola, Austria, Port of East Africa, and Northern Rhodesia in 1952 because detailed breakdown from these sources is not available. Imports from these sources totaled 48,350 pounds. For 1953, does not include imports from Angola, Australia, Brazil, Canada, and Union of South Africa. Imports from these sources totaled 32,359 pounds.

[3] Includes imports from Angola, Argentina, Canada, Colombia, Southern Rhodesia, Tanganyika, and United Kingdom.

[4] Includes imports from Argentina, Canada, Mexico, Northern Rhodesia, Southern Rhodesia, and Tanganyika.

[5] Includes first- and second-quality film.

[6] Includes other-quality film.

TABLE 7.—*Muscovite block, film, and splittings and phlogopite splittings imported for consumption, 1940–53* [1]

[Thousand pounds and dollars]

Year	Block [2]		Film [3]		Splittings			
					Muscovite [4]		Phlogopite [5]	
	Pounds	Value	Pounds	Value	Pounds	Value	Pounds	Value
1940	[6] 968	$513	343	$196	6, 283	$1, 357	732	$210
1941	[6] 1, 391	1, 043	500	417	10, 695	2, 487	403	182
1942	[6] 2, 489	2, 070	1, 884	1, 209	10, 228	2, 996	888	346
1943	4, 261	6, 018	2, 170	3, 903	14, 248	3, 888	257	207
1944	3, 746	3, 389	899	1, 805	1, 980	856	863	499
1945	3, 186	3, 987	1, 211	408	5, 526	1, 511	187	71
1946	2, 649	2, 037	373	508	8, 835	3, 658	538	246
1947	1, 349	1, 112	360	526	8, 426	5, 137	637	311
1948	2, 265	2, 465	138	272	15, 254	11, 691	[7] 819	488
1949	1, 970	2, 076	205	531	17, 539	15, 716	[7] 783	434
1950	3, 230	3, 381	594	1, 102	22, 136	17, 844	[7] 918	502
1951	3, 770	4, 548	1, 050	3, 044	18, 328	12, 791	1, 297	712
1952	3, 111	4, 791	789	1, 878	7, 246	6, 041	703	368
1953	3, 240	5, 369	1, 299	3, 648	7, 600	3, 651	[7] 745	374

[1] Compiled from records of the U. S. Department of Commerce.

[2] Includes unmanufactured mica valued above 15 cents per pound, minus phlogopite valued above 15 cents per pound, plus manufactured films and splittings, not cut or stamped to dimensions, over 12/10,000 inch thick, from Brazil.

[3] Manufactured films and splittings, not cut or stamped to dimensions, over 12/10,000 inch thick, from India.

[4] Manufactured films and splittings, not cut or stamped to dimensions, not above 12/10,000 inch thick, from India.

[5] Manufactured films and splittings, not cut or stamped to dimensions, not over 12/10,000 inch thick, from Canada, Mexico, and Madagascar.

[6] Includes unmanufactured phlogopite valued above 15 cents per pound.

[7] Includes imports from France.

TABLE 8.—*Mica block, film, and splittings imported into the United States, 1952–53, by varieties and principal sources*

[Pounds]

	U. S. Tariff Commission data		U. S. Department of Commerce data	
	1952	1953	1952	1953
Muscovite block:				
India	873, 683	2, 000, 196	524, 174	1, 016, 540
Brazil	2, 621, 277	2, 299, 479	2, 432, 275	2, 126, 498
Other	[1] 115, 648	[2] 164, 493	154, 396	96, 541
Total	3, 610, 608	4, 464, 168	[3] 3, 110, 845	[3] 3, 239, 579
Muscovite film:				
India	301, 655	448, 189	[4] 788, 734	[4] 1, 298, 890
Brazil	565	1, 467	------	------
Other	150	1, 158	------	------
Total	302, 370	450, 814	788, 734	1, 298, 890
Muscovite splittings:				
India	6, 759, 201	7, 619, 603	[5] 7, 245, 780	[5] 7, 599, 952
Other	[6] 16, 600	[6] 6, 245	------	------
Total	6, 775, 801	7, 625, 848	7, 245, 780	7, 599, 952
Phlogopite splittings:				
Madagascar	677, 688	703, 281	697, 712	739, 018
Other	[7] 1, 084	[8] 2, 795	5, 613	6, 179
Total	678, 772	706, 076	[9] 703, 325	[10] 745, 197

[1] Includes imports from Angola, Argentina, Canada, Colombia, Southern Rhodesia, Tanganyika, and United Kingdom.

[2] Includes imports from Argentina, Canada, Mexico, Northern Rhodesia, Southern Rhodesia, and Tanganyika.

[3] Includes imports of unmanufactured mica valued above 15 cents per pound, minus phlogopite valued above 15 cents per pound, plus imports from Brazil of manufactured films and splittings, not cut or stamped to dimension, over 12/10,000 inch in thickness.

[4] Manufactured films and splittings, not cut or stamped to dimensions, over 12/10,000 inch thick, from India.

[5] Manufactured films and splittings, not cut or stamped to dimensions, not over 12/10,000 inch thick, from India.

[6] Includes imports from Brazil and Mexico.

[7] Includes imports from Canada and Mexico.

[8] Includes imports from Mexico.

[9] Manufactured films and splittings, not cut or stamped to dimensions not over 12/10,000 inch thick, from Canada, Madagascar, and Mexico.

[10] Manufactured films and splittings, not cut or stamped to dimensions, not over 12/10,000 inch thick, from France, Madagascar, and Mexico.

in the heavy electrical industries, for household appliances, for radio and television tubes, and, to a smaller degree, for capacitors and other uses. For the civilian uses of block and film mica, the quantities of high-quality mica required are not excessive in proportion to total supplies. On the other hand, military requirements for mobilization and war place a heavy emphasis on the highest qualities of mica for electronic uses. Manufacturers of less essential products are then forced to use lower qualities of mica; even then, there may be a surplus of low-quality mica that cannot be absorbed. The effect of this is inevitably to raise the price of the higher qualities.

Mica splittings are used principally in making built-up mica board, tape, and paper. Built-up mica (micanite) consists of alternate layers of a binder (shellac, alkyl, or silicone resin) and irregularly arranged and partly overlapped splittings. After heating, pressing, and trimming, the built-up mica constitutes a raw material from which numerous shapes are stamped or cut. Built-up mica products are the most important insulating medium in the electrical industry. Electric motors, generators, and transformers either could not operate or could operate only with greatly reduced efficiency without suitable built-up mica.

Data on consumption of splittings in 1927–53 for making built-up mica are shown in table 10.

Dry-ground mica is used in large tonnages in the manufacture of roofing materials and wallboard joint cements. Smaller quantities are used by numerous industries, including paint, rubber, plastics, well drilling, pipeline enamel, and welding rods. Wet-ground mica is used by paint, wallpaper, and tire manufacturers. Table 13 shows the ground mica sold by producers from 1926–53.

PRICES

In view of the wide range of qualities and grades of sheet mica, the market values for domestically produced sheet mica range from a few cents to over $20 per pound. Prices prevailing in the North Carolina district, as of August 5, 1954, quoted by E&MJ Metal and Mineral Markets, were as follows:

Clear quality

Grade (size):	*Price per pound*
Punch mica	$0. 10–$0. 16
1½ x 2 inch	. 70– 1. 60
2 x 2 inch	1. 10– 1. 60
2 x 3 inch	1. 60– 2. 00
3 x 3 inch	1. 80– 2. 30
3 x 4 inch	2. 00– 2. 60
3 x 5 inch	2. 60– 3. 00
4 x 6 inch	2. 75– 4. 00
6 x 8 inch	4. 00– 8. 00

Stained or electric quality—sold at approximately 10 to 15 percent lower than clear sheet.

In July 1952 General Services Administration announced the schedule of prices for domestic mica to be purchased under the provisions of the domestic mica-production expansion program (table 14). The subsidized prices were about 4 to 6 times the market price. "Good-Stained and Better" and "Stained" were retained for the stockpile. "Heavy-Stained" and other were reserved for sale to industry.

TARIFF

Mica imports into the United States are subject to duty under Paragraph 208 of the Tariff Regulations under eight classifications. A summary of the tariff rates for unmanufactured and manufactured mica, as provided in the Tariff Acts of 1922 and 1930, as well as the present rates (1952), is shown in table 15.

TABLE 9.—*Fabrication of muscovite ruby and nonruby block and film mica in the United States, July–December 1952 and 1953, by end-product use*

[Pounds]

Quality use	1952		1953					
	July–December		January–June		July–December		Total	
	Quantity (pounds)	Percent of grand total	Quantity (pounds)	Percent of grand total	Quantity (pounds)	Percent of grand total	Quantity (pounds)	Percent of grand total
Good-Stained and Better:[1]								
Electronic:								
Capacitors	107, 813	5. 6	147, 364	6. 2	87, 760	4. 6	235, 124	5. 5
Tubes	62, 395	3. 2	69, 627	2. 9	56, 720	3. 0	126, 347	2. 9
Other	1, 235	. 1	2, 374	. 1	2, 156	. 1	4, 530	. 1
Total	171, 443	8. 9	219, 365	9. 2	146, 636	7. 7	366, 001	8. 5
Nonelectronic	4, 378	. 2	5, 386	. 2	12, 668	. 6	18, 054	. 4
Total	175, 821	9. 1	224, 751	9. 4	159, 304	8. 3	384, 055	8. 9
Stained:[2]								
Electronic:								
Capacitors	14, 989	. 8	11, 068	. 4	10, 343	. 5	21, 411	. 5
Tubes	1, 161, 648	60. 0	1, 224, 650	51. 2	732, 267	38. 3	1, 956, 917	45. 5
Other	9, 232	. 5	18, 826	. 8	13, 217	. 7	32, 043	. 7
Total	1, 185, 869	61. 3	1, 254, 544	52. 4	755, 827	39. 5	2, 010, 371	46. 7
Nonelectronic	73, 722	3. 8	92, 554	3. 9	59, 183	3. 1	151, 737	3. 5
Total	1, 259, 591	65. 1	1, 347, 098	56. 3	815, 010	42. 6	2, 162, 108	50. 2
Lower than Stained:								
Electronic:								
Capacitors	185	(3)	153	(3)	161	(3)	314	(3)
Tubes	126, 848	6. 6	248, 415	10. 4	292, 372	15. 3	540, 787	12. 6
Other	4, 063	. 2	27, 946	1. 2	4, 719	. 2	32, 665	. 8
Total	131, 096	6. 8	276, 514	11. 6	297, 252	15. 5	573, 766	13. 4
Nonelectronic[4]	367, 226	19. 0	542, 375	22. 7	643, 295	33. 6	1, 185, 670	27. 5
Total	498, 322	25. 8	818, 889	34. 3	940, 547	49. 1	1, 759, 436	40. 9
Grand total	1, 933, 734	100. 0	2, 390, 738	100. 0	1, 914, 861	100. 0	4, 305, 599	100. 0

[1] Includes first- and second-quality film.

[2] Includes other-quality film.

[3] Less than 0.1 percent.

[4] Includes punch mica.

TABLE 10.—*Built-up mica*[1] *sold or used in the United States, by kind of products, 1927–53*

Year	Molding plate		Segment plate		Heater plate		Flexible (cold)		All other (tape, etc.)		Total	
	Pounds	Value	Pounds	Value	Pounds	Value	Pounds	Value	Pounds	Value	Pounds	Value
1927	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	2, 925, 941	(2)
1928	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)
1929	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	4, 042, 478	(2)
1930	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)
1931	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	1, 702, 938	(2)
1932	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	922, 936	(2)
1933	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	1, 308, 924	(2)
1934	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	(2)	1, 839, 888	(2)
1935	817, 995	$703, 688	783, 870	$934, 654	194, 171	$251, 502	176, 345	$183, 026	413, 242	$875, 672	2, 385, 623	$2, 948, 542
1936[3]	481, 810	414, 388	462, 174	542, 342	124, 400	165, 608	114, 161	119, 035	270, 189	574, 263	1, 452, 734	1, 815, 636
1937	1, 405, 056	(2)	1, 399, 014	(2)	290, 564	(2)	258, 620	(2)	780, 873	(2)	4, 134, 127	(2)
1938	531, 661	505, 000	479, 273	605, 000	209, 814	280, 000	119, 440	160, 000	417, 466	650, 000	1, 757, 654	2, 200, 000
1939	1, 099, 066	1, 090, 000	1, 135, 555	1, 610, 000	369, 677	815, 000	239, 582	290, 000	581, 515	925, 000	3, 425, 395	4, 730, 000
1940	1, 315, 000	1, 410, 000	1, 510, 000	2, 024, 000	561, 000	878, 000	330, 000	383, 000	697, 000	1, 116, 000	4, 413, 000	5, 811, 000
1941	1, 873, 822	1, 854, 000	2, 317, 364	2, 599, 000	539, 410	789, 000	705, 532	754, 000	1, 199, 361	2, 089, 000	6, 635, 489	8, 085, 000
1942	1, 599, 000	1, 552, 000	2, 045, 000	2, 367, 000	212, 000	303, 000	649, 000	743, 000	1, 628, 000	2, 794, 000	6, 133, 000	7, 759, 000
1943	1, 785, 000	1, 912, 000	2, 388, 000	2, 846, 000	235, 000	462, 000	707, 000	831, 000	1, 716, 000	3, 222, 000	6, 831, 000	9, 273, 000
1944	1, 792, 052	1, 816, 920	2, 521, 527	3, 073, 074	364, 773	372, 683	817, 255	1, 004, 719	1, 942, 642	3, 876, 477	7, 438, 249	10, 143, 873
1945	2, 015, 993	2, 096, 267	2, 441, 076	3, 074, 435	390, 598	716, 903	775, 342	955, 126	1, 571, 900	3, 003, 988	7, 194, 909	9, 846, 719
1946	1, 742, 835	2, 061, 588	1, 860, 173	2, 460, 860	685, 580	1, 283, 908	553, 274	746, 600	1, 644, 693	3, 732, 906	6, 486, 555	10, 285, 862
1947	1, 660, 883	1, 832, 779	1, 920, 875	2, 513, 205	1, 248, 461	2, 351, 901	677, 801	973, 247	1, 388, 094	3, 741, 913	6, 896, 114	11, 413, 045
1948	1, 545, 401	2, 435, 709	2, 008, 924	3, 614, 521	1, 033, 995	2, 126, 367	339, 509	575, 066	1, 020, 989	3, 792, 278	5, 948, 818	12, 543, 941
1949	1, 579, 846	2, 131, 727	1, 727, 212	3, 041, 809	1, 033, 035	1, 965, 678	431, 660	677, 753	1, 523, 515	5, 386, 887	6, 295, 268	13, 203, 854
1950	2, 114, 502	3, 860, 049	2, 548, 442	4, 928, 870	898, 333	2, 416, 478	711, 412	1, 914, 911	1, 773, 912	7, 120, 539	8, 046, 601	20, 240, 847
1951	2, 184, 654	3, 898, 117	2, 778, 482	5, 488, 492	1, 140, 404	2, 901, 670	917, 326	2, 596, 787	2, 439, 289	11, 457, 814	9, 460, 155	26, 342, 880
1952	1, 682, 742	3, 137, 011	2, 094, 397	3, 972, 515	511, 120	1, 419, 575	721, 037	2, 002, 263	2, 139, 670	10, 916, 674	7, 148, 966	21, 448, 038
1953	1, 704, 644	3, 323, 141	2, 106, 226	4, 054, 997	822, 207	2, 221, 995	559, 671	1, 713, 996	2, 455, 761	9, 410, 204	7, 648, 509	20, 724, 333

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[1] Consists of a composite of alternate layers of a binder and irregularly arranged and partly overlapped splittings.
[2] Figures not available.
[3] Figures for 6 months (January-June 1936).

TABLE 11.—*Ground mica sold by producers in the United States to various industries, 1952–53*

Use	1952			1953		
	Short tons	Percent of total	Value	Short tons	Percent of total	Value
Roofing	30, 922	41	$887, 700	32, 389	44	$935, 208
Wallpaper	583	1	79, 673	598	1	79, 522
Rubber	5, 126	7	457, 194	5, 668	8	547, 654
Paint	16, 566	22	1, 549, 671	15, 258	21	1, 435, 294
Plastics	1, 959	3	181, 889	1, 641	2	153, 440
Pipeline enamel	2, 668	4	85, 537	(1)	(1)	(1)
Welding rods	1, 749	2	102, 934	1, 538	2	85, 665
Well drilling	4, 847	6	245, 504	4, 347	6	195, 152
Miscellaneous [2]	10, 386	14	688, 001	11, 633	16	760, 485
Total	74, 806	100	4, 278, 103	73, 072	100	4, 192, 420

[1] Included with "Miscellaneous" to avoid disclosure of individual company operations.

[2] Includes mica used for molded electric insulation, house insulation, Christmas-tree snow, manufacture of axle grease and oil, annealing, and other purposes.

TABLE 12.—*Ground mica sold by producers in the United States, by methods of grinding, 1923–53*

Year	Dry-ground		Wet-ground		Total	
	Short tons	Value	Short tons	Value	Short tons	Value
1923	2, 830	$123, 450	2, 102	$250, 170	4, 932	$373, 620
1924	2, 592	101, 700	2, 868	331, 410	5, 460	433, 110
1925	2, 522	101, 680	2, 402	279, 940	4, 924	381, 620
1926	2, 375	98, 290	2, 490	206, 370	4, 865	304, 660
1927	2, 594	90, 968	3, 199	332, 511	5, 793	423, 479
1928	2, 871	95, 151	3, 317	358, 458	6, 188	453, 609
1929	1, 819	62, 029	2, 697	328, 332	4, 516	390, 361
1930	5, 956	190, 635	1, 575	161, 623	7, 531	352, 258
1931	5, 363	168, 783	2, 444	267, 653	7, 807	436, 436
1932	5, 253	126, 714	2, 452	184, 126	7, 705	310, 840
1933	6, 439	135, 178	3, 392	263, 503	9, 831	398, 681
1934	6, 824	156, 046	2, 723	247, 284	9, 547	403, 330
1935	15, 178	341, 825	3, 145	201, 148	18, 323	542, 973
1936	20, 800	457, 042	4, 785	265, 374	25, 585	722, 416
1937	21, 150	457, 879	6, 095	381, 933	27, 245	839, 812
1938	19, 757	466, 959	7, 329	457, 595	27, 086	924, 554
1939	23, 222	547, 539	7, 702	608, 794	30, 924	1, 156, 333
1940	21, 809	515, 930	6, 175	500, 698	27, 984	1, 016, 628
1941	31, 914	733, 559	11, 505	798, 792	43, 419	1, 532, 351
1942	36, 369	805, 163	10, 610	848, 195	46, 979	1, 653, 358
1943	40, 256	1, 027, 781	11, 326	962, 363	51, 582	1, 990, 144
1944	47, 023	1, 382, 147	5, 690	532, 562	52, 713	1, 914, 709
1945	43, 686	1, 243, 075	8, 120	752, 894	51, 806	1, 995, 969
1946	53, 908	1, 582, 974	8, 205	933, 044	62, 113	2, 516, 018
1947	55, 731	1, 852, 768	8, 809	1, 114, 945	64, 540	2, 967, 713
1948	55, 494	2, 035, 618	9, 148	1, 197, 014	64, 642	3, 232, 632
1949	49, 133	1, 850, 400	7, 260	1, 010, 556	56, 393	2, 860, 956
1950	61, 139	2, 374, 089	11, 111	1, 561, 608	72, 250	3, 935, 697
1951	59, 200	2, 294, 620	10, 922	1, 548, 008	70, 122	3, 842, 628
1952	62, 465	2, 526, 407	12, 341	1, 751, 696	74, 806	4, 278, 103
1953	60, 127	2, 438, 628	12, 945	1, 753, 792	73, 072	4, 192, 420

TABLE 13.—*Ground mica sold by producers in the United States to various industries, 1926–53*

Year	Roofing			Wallpaper			Rubber			Paint			Plastics		
	Short tons	Percent of total	Value	Short tons	Percent of total	Value	Short tons	Percent of total	Value	Short tons	Percent of total	Value	Short tons	Percent of total	Value
1926	609	12	$24,657	1,260	26	$108,558	1,655	34	$134,893	(1)	--------	(1)	--------	--------	--------
1927	1,292	22	34,347	1,884	33	192,091	1,589	27	144,594	(1)	--------	(1)	--------	--------	--------
1928	1,293	21	32,283	1,614	26	182,247	1,933	31	172,180	(1)	--------	(1)	--------	--------	--------
1929	603	13	18,132	1,614	36	202,386	1,316	29	119,103	(1)	--------	(1)	(1)	--------	(1)
1930	4,825	64	135,076	970	13	104,197	506	7	53,810	(1)	--------	(1)	(1)	--------	(1)
1931	3,591	46	83,017	2,665	34	252,908	1,029	13	76,781	(1)	--------	(1)	(1)	--------	(1)
1932	4,383	57	93,967	1,752	23	135,806	668	9	44,987	558	7	$19,533	(1)	--------	(1)
1933	5,755	58	112,109	2,340	24	194,014	963	10	63,068	(1)	--------	(1)	(1)	--------	(1)
1934	6,594	69	149,082	1,777	19	170,187	858	9	68,631	(1)	--------	(1)	(1)	--------	(1)
1935	14,403	79	324,165	2,051	11	134,457	768	4	46,456	(1)	--------	(1)	(1)	--------	(1)
1936	20,279	79	432,493	2,869	11	166,315	516	2	27,012	1,307	5	71,155	(1)	--------	(1)
1937	21,636	79	457,652	2,623	10	190,127	1,413	5	99,106	1,011	4	69,125	(1)	--------	(1)
1938	18,795	70	402,671	2,926	11	232,870	1,187	4	82,809	1,666	6	117,595	(1)	--------	(1)
1939	19,255	62	406,522	3,586	12	265,359	2,539	8	204,977	1,916	6	144,235	(1)	--------	(1)
1940	18,359	66	385,720	2,915	10	220,995	1,731	6	144,202	1,874	7	141,192	(1)	--------	(1)
1941	25,178	58	498,946	3,219	8	256,067	3,476	8	223,182	4,020	9	253,294	(1)	--------	(1)
1942	28,342	60	561,205	2,168	5	168,178	1,754	4	115,563	6,819	14	418,548	1,868	4	$125,449
1943	29,022	56	682,970	1,814	3	153,937	3,063	6	175,922	7,608	15	431,832	2,969	6	203,699
1944	21,280	40	495,431	1,325	3	123,180	1,137	2	95,689	5,107	10	329,905	1,079	2	95,887
1945	28,407	55	711,175	1,911	4	139,719	3,715	7	241,657	7,570	14	382,992	990	2	83,941
1946	30,102	48	791,639	2,827	5	240,786	4,951	8	381,231	14,452	23	638,598	321	1	33,066
1947	40,012	62	1,228,972	1,724	3	205,454	3,900	6	429,570	8,151	13	560,336	1,374	2	72,836
1948	32,969	51	1,074,322	1,256	2	148,311	4,372	7	474,294	9,172	14	703,558	590	1	63,428
1949	29,481	52	939,587	877	2	118,954	3,856	7	378,411	8,484	15	620,306	1,439	2	103,417
1950	32,594	45	1,083,584	622	1	82,565	5,776	8	580,840	14,386	20	1,102,524	1,542	2	145,599
1951	27,919	40	846,801	865	1	121,065	6,551	9	507,602	11,760	17	1,028,490	1,186	2	138,778
1952	30,922	41	887,700	583	1	79,673	5,126	7	457,194	16,566	22	1,549,671	1,959	3	181,889
1953	32,389	44	935,208	598	1	79,522	5,668	8	547,654	15,258	21	1,435,294	1,641	2	153,440

Year	Pipeline enamel			Welding rods			Well drilling			Miscellaneous [2]			Total		
	Short tons	Percent of total	Value	Short tons	Percent of total	Value	Short tons	Percent of total	Value	Short tons	Percent of total	Value	Short tons	Percent of total	Value
1926	--------	--------	--------	--------	--------	--------	--------	--------	--------	1,342	28	$36,552	4,866	100	$304,660
1927	--------	--------	--------	--------	--------	--------	--------	--------	--------	1,028	18	52,447	5,793	100	423,479
1928	--------	--------	--------	--------	--------	--------	--------	--------	--------	1,348	22	66,899	6,188	100	453,609
1929	--------	--------	--------	--------	--------	--------	--------	--------	--------	983	22	50,740	4,516	100	390,361
1930	--------	--------	--------	--------	--------	--------	--------	--------	--------	1,230	16	59,175	7,531	100	352,258
1931	(1)	--------	(1)	--------	--------	--------	--------	--------	--------	521	7	23,730	7,806	100	436,436
1932	(1)	--------	(1)	--------	--------	--------	--------	--------	--------	344	4	16,547	7,705	100	310,840
1933	(1)	--------	(1)	--------	--------	--------	--------	--------	--------	772	8	29,490	9,830	100	398,681
1934	(1)	--------	(1)	--------	--------	--------	--------	--------	--------	318	3	15,430	9,547	100	403,330
1935	(1)	--------	(1)	--------	--------	--------	--------	--------	--------	1,101	6	37,895	18,323	100	542,973
1936	(1)	--------	(1)	--------	--------	--------	--------	--------	--------	614	3	25,441	25,585	100	722,416
1937	(1)	--------	(1)	--------	--------	--------	--------	--------	--------	562	2	23,802	27,245	100	839,812
1938	(1)	--------	(1)	--------	--------	--------	--------	--------	--------	2,512	9	88,609	27,086	100	924,554
1939	(1)	--------	(1)	--------	--------	--------	--------	--------	--------	3,628	12	135,240	30,924	100	1,156,333
1940	(1)	--------	(1)	--------	--------	--------	--------	--------	--------	3,105	11	124,519	27,984	100	1,016,628
1941	(1)	--------	(1)	--------	--------	--------	(1)	--------	(1)	7,526	17	300,862	43,419	100	1,532,351
1942	(1)	--------	(1)	(1)	--------	(1)	(1)	--------	(1)	6,028	13	264,415	46,979	100	1,653,358
1943	(1)	--------	(1)	(1)	--------	(1)	(1)	--------	(1)	7,106	14	341,784	51,582	100	1,990,144
1944	(1)	--------	(1)	(1)	--------	(1)	(1)	--------	(1)	22,785	43	774,617	52,713	100	1,914,709
1945	(1)	--------	(1)	(1)	--------	(1)	(1)	--------	(1)	9,213	18	436,485	51,806	100	1,995,969
1946	(1)	--------	(1)	(1)	--------	(1)	(1)	--------	(1)	9,460	15	430,698	62,113	100	2,516,018
1947	(1)	--------	(1)	(1)	--------	(1)	(1)	--------	(1)	9,379	14	470,545	64,540	100	2,967,713
1948	(1)	--------	(1)	(1)	--------	(1)	(1)	--------	(1)	16,283	25	768,719	64,642	100	3,232,632
1949	(1)	--------	(1)	(1)	--------	(1)	(1)	--------	(1)	12,256	22	700,281	56,393	100	2,860,956
1950	(1)	--------	(1)	(1)	--------	(1)	(1)	--------	(1)	17,330	24	940,585	72,250	100	3,935,697
1951	6,378	9	$202,741	1,203	2	$78,916	(1)	--------	(1)	14,260	20	918,235	70,122	100	3,842,628
1952	2,668	4	85,537	1,749	2	102,934	4,847	6	$245,504	10,386	14	688,001	74,806	100	4,278,103
1953	(1)	--------	(1)	1,538	2	85,665	4,347	6	195,152	11,633	16	760,485	73,072	100	4,192,420

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[1] Included with "Miscellaneous" to avoid revealing individual company operations.

[2] Includes mica used for molded electric insulation, house insulation, Christmas-tree snow, manufacture of axle grease and oil, annealing, pipeline enamel, oil-well drilling, welding, paint, plastic specialities and other purposes.

TABLE 14.—*Domestic mica-purchasing program prices per pound, full-trimmed and half-trimmed ruby and nonruby block and film mica, f. o. b. Government depot* [1]

Grades	Quality				
	Full-trimmed			Half-trimmed	
	Good-Stained and Better	Stained	Heavy-Stained	Stained	Heavy-Stained
Ruby:					
No. 3 and larger	$70.00	$18.00	$13.00	$12.00	$8.00
Nos. 4 and 5	40.00	8.00	6.00	5.00	4.00
Nos. 5½ and 6	15.00	5.00	3.00	3.00	2.00
Nonruby:					
No. 3 and larger	70.00	14.40	10.40	9.60	6.40
Nos. 4 and 5	40.00	6.40	4.80	4.00	3.20
Nos. 5½ and 6	15.00	4.00	2.40	2.40	1.60

[1] The price to be paid for ruby hand-cobbed mica accepted by the Government and delivered f. o. b. Government depot was $600 per short ton; nonruby, $540.

SUBSTITUTES

Because of the importance of sheet mica, during both peacetime and wartime, considerable research has been conducted during the past three decades in an effort to obtain a substitute for mica (*25*). Several substitutes have been developed and have displaced mica in certain fields; for example, heat-resistant glass in stove windows, glass and paper for certain capacitor uses—but none can replace mica in all its civilian applications or in hardly any of its strategic military applications.

A synthetic mica program was initiated at the Bureau of Mines Electrotechnical Laboratory, Norris, Tenn., in September 1947, in cooperation with the Army Signal Corps, the Bureau of Ships, and the Office of Naval Research. The major accomplishments to 1954 include: (1) Development of a commercially feasible method of manufacturing synthetic flake mica in large quantities by internal electric resistance melting; (2) synthesis of micas by solid-state reaction; (3) development of machinable synthetic mica dielectrics by hot-pressing and phosphate bonding; and (4), development of synthetic mica cermets and methods of producing a reconstituted mica sheet from synthetic mica of controllable thickness and uniformity (*8, 9, 10, 17*).

To 1954 the synthetic mica crystals have been small in size, usually 2 inches in diameter or less. This material has shown its possible usefulness, however, in applications where a fine-grained mica can be employed, such as in glass-bonded mica ceramics, hot-pressed machinable dielectrics, bonding agent in abrasive wheels, and electrical insulation materials. Synthetic mica, in its present state of development, is primarily a new material and is incidentally, and only to a minor degree, a substitute for natural sheet mica. The develop-

TABLE 15.—*Mica, unmanufactured sheet: United States rates of duty in the Tariff Acts of 1922 and 1930, and in 1952* [1]

[Cents per pound; percent ad valorem]

Paragraph (Act of 1930)	Tariff class	Tariff rate in—		
		Tariff Act of 1922	Tariff Act of 1930	1952
208 (a)	Mica, unmanufactured:			
	Valued not above 15 cents per lb	4¢	4¢	[2] 4¢
	Valued above 15 cents per lb	25%	4¢+25%	[3] 2¢+15%
208 (b)	Mica, cut or stamped to dimensions, shape or form	30%	40%	40%
208 (c)	Mica films or splittings, not cut or stamped to dimensions:			
	Not over 12/10,000 inch thick	30%	25%	[3] 12½%
	Over 12/10,000 inch thick	30%	40%	[3] 20%
208 (d)	Mica films or splittings cut or stamped to dimensions	40%	45%	[3] 22½%
208 (e)	Mica plates and built-up mica, and all manufactures of mica, or of which mica is the component material of chief value	40%	40%	[3] 25%
208 (f)	Untrimmed phlogopite mica from which no piece exceeding 2 inches in length or 1 inch in width may be cut	10%	15%	[3] 5%
208 (g)	Mica waste or scrap valued at not more than 5 cents per lb.:			
	Phlogopite mica waste or scrap	[4] 10%	25%	[2] 12½%
	Other	[4] 10%	25%	[3] 15%
208 (h)	Mica, ground or pulverized	20%	20%	[2] 12½%

[1] U. S. Tariff Commission.

[2] Rate bound in trade agreement with Argentina, effective Nov. 15, 1941, and in General Agreement on Tariffs and Trade (Geneva), effective July 31, 1948.

[3] General Agreement on Tariffs and Trade (Geneva), effective July 9, 1948.

[4] Largely dutiable as "waste (mine or mineral), n. s. p. f."

ment of synthetic mica ceramics and reconstituted mica appears to be the key to the potentialities of synthetic mica.

Brush Beryllium Co. of Cleveland, Ohio, and Mycalex Corp. of America, New Jersey, were manufacturing synthetic mica on a commercial basis in 1954 and had active programs of research on synthetic mica. The National Bureau of Standards, Colorado School of Mines, and Horizons, Inc., Cleveland, Ohio, also have conducted research on synthetic mica.

Although the demand by the electronic industry for high-quality sheet mica cannot be satisfied in the foreseeable future through the use of synthetic mica, enough progress has been made to indicate that continuation and intensification of synthetic mica research is fully justified. It is anticipated that synthetic mica may some day replace strategic natural sheet mica and relieve the United States of dependence on foreign sources.

Recent developments by industry use factory scrap mica and to a limited degree, domestic mine scrap mica, to produce a mica paper called "reconstituted mica" (*28*). This new product can be used in applications now using builtup mica (micanite), which is made from imported splittings. Reconstituted mica does not compare with micanite in mechanical strength as it comes from the sheetmaking machine, but in combination with suitable bonding agents, certain mica products can be made from reconstituted mica that are superior to those made from mica splittings. General Electric Co. and Mica Insulator Co. market reconstituted mica products (*13*, *21*). Developments in 1954 indicate that the need for imported splittings for manufacturing builtup mica products will decline, and possibly United States dependence on distant foreign sources for splittings will be eliminated.

WORLD WAR II PROGRAMS

The military requirements for the higher qualities of block and film mica during periods of war or intensive mobilization may be 4 to 6 times the normal civilian consumption rate. It became apparent at the outset of World War II that strategic block and film mica would be in extremely short supply. After entering the war the United States, acting jointly with the United Kingdom, took steps to stimulate production in India and other mica-producing countries and to assure equitable distribution among the Allied countries.

In 1940, under Public Law 117, 76th Congress (1939) preliminary stockpile objectives for mica were established and a purchasing program was begun. Furthermore, block mica, mica splittings, and built-up-mica products were placed under export control.

The Geological Survey, in cooperation with the Bureau of Mines, conducted investigations of domestic deposits. Examination of the Spruce Pine area of North Carolina was completed, and further examination of the New England area was begun.

Under the Office of Production Management during 1941 up to Pearl Harbor, the first available statistics showing consumption by sizes and quality were prepared by the Bureau of Mines and served as a guide to reevaluate and increase the stockpile requirements.

The Office of Production Management recommended that immediate mandatory action be taken to limit mica to essential uses, to provide control over stocks and future imports, actively to promote the production of strategic mica both in the United States and Latin America with appropriate assistance from the Federal Loan Agency, and to establish a Defense Industry Advisory Committee to obtain the most efficient gearing of the domestic mica industry to the war economy. On October 23, 1941, at the request of the Office of Production Management, the Nonmetallic Group of the Advisory Committee on Minerals and Metals, National Academy of Sciences, prepared Report 14461, Mica, Its Conservation and Substitution, which was an important guide to initial action during the period immediately following distribution of the report.

Soon after Pearl Harbor the Mica Section of the Mica-Graphite Division of the War Production Board became active in developing a mica program. The magnitude of the requirements soon made it apparent that production would have to be stimulated in the principal producing areas and that it would be unlikely that the United States could obtain enough mica for its needs unless Government-sponsored programs were established. The Combined Raw Materials Board decided that it was desirable that these be joint programs for the Allies. The United States Government stimulated production in Brazil and Argentina by offering high prices. The United States Government furnished further assistance to Brazil through personnel, equipment, and funds. In India, an Anglo-American Mica Mission functioned during the war to stimulate production and buying and allocate the output, but labor and war disburbances kept Indian production from reaching even the amount it had achieved in 1937 and 1938.

In the United States the Colonial Mica Corporation was created as buying agent for domestic strategic mica for the Metals Reserve Company. It also aided domestic producers by advancing funds against future production and by leasing mining equipment. The corporation paid high prices for domestic mica of strategic quality and resold to users at much lower prices, thus subsidizing the mica-mining industry. The net result of the domestic program was that approximately 15 percent of the total block- and film-mica requirements of the United States was met from domestic sources during the war period. However, only 4 percent of the requirements for strategic quality mica acceptable for stockpiling was produced domestically. The cost of mica procured through the efforts of the Colonial Mica Corporation was approximately four times the cost of the same grades and qualities of mica obtained from foreign sources. In spite of this very substantial support to the domestic mica mining industry, most mining operations were conducted at a loss. Less than 25 percent of the mines yielded more than 75 percent of the mica produced. By the end of 1944 the mica-supply situation materially improved when increased imports were received from India and Brazil; and Colonial Mica Corporation's purchases were restricted to ruby mica. It had been found difficult to persuade manufacturers to use nonruby mica.

POST-WORLD WAR II GOVERNMENT ACTIVITIES

At the end of World War II all Government efforts to improve mica supplies were discontinued. The mandatory basic series of statistical data on the consumption of sheet mica by grades, qualities, and uses was abandoned.

Mica was designated as a commodity for accumulation under the Stockpiling Act (Public Law 520) of 1946. Specifications were prepared and objectives established for the purchase of four varieties of material: (1) Muscovite block and film, Good-Stained and Better, grade No. 6 and larger; (2) Stained muscovite block (radio grade); (3) muscovite splittings; and (4) phlogopite splittings. In 1953 the first and second categories were revised to read: (1) Muscovite block, Stained and Better, grade No. 6 and larger; and (2) muscovite film, First and Second qualities. Before the outbreak of fighting in Korea in 1950, several hundred contracts were made for stockpiling mica. Purchases were made, and there were some transfers of surplus World War II stocks.

In 1950, under provisions of the Defense Production Act, the Defense Minerals Administration was established to encourage the exploration, development, and mining of critical and strategic metals and minerals, including strategic mica. With dissolution of DMA in early 1952, the exploration functions were delegated to Defense Minerals Exploration Administration; and the programing, development loans, facilities loans, and tax-amortization functions were transferred to Defense Minerals Procurement Administration. The encouragement of exploration of unknown or undeveloped sources of strategic or critical metals and minerals through financial assistance established as a DMA program in 1951 became the function of DMEA.

On September 31, 1954, DMEA had executed 143 contracts for mica exploration. Of these, 114 were canceled or terminated, and 29 were in progress. Of the terminated contracts 31 certificates of discovery valued at $251,195 were issued. Twenty-five certifications were in North Carolina, 2 in Alabama, and 1 each in Georgia, Idaho, New Hampshire, and South Dakota.

A domestic mica-purchasing program was announced by General Services Administration on March 14, 1952. The program was to run until June 30, 1955, or when 25,000 short tons of hand-cobbed mica (or 2¼ million pounds of full-trimmed block and film mica) was delivered to and accepted by the Government. In 1954 the program was extended to June 30, 1957. In July 1952 General Services Administration activated buying depots at Spruce Pine, N. C., and Custer, S. Dak. A third depot was opened at Franklin, N. H., in August. The prices paid are given in the section on prices. From mid-1952 to November 30, 1954, a total of 1,586,197 pounds of mica had been purchased as follows: Spruce Pine, ruby block—232,456, nonruby block—42,991, and ruby hand-cobbed—33,167; Custer, ruby block—20,888, nonruby block—102, ruby hand-cobbed—868,422, and nonruby hand-cobbed—19,718; and Franklin, ruby block—44,498, nonruby block—149, ruby hand-cobbed—301,406, nonruby hand-cobbed—22,366, and film—34.

The Government program contained provisions for expanding imports through purchase contracts for muscovite block and film for the National Stockpile, to be negotiated by Emergency Procurement Service, General Services Administration. Funds were not available for this program until the latter part of February 1952, after which important contracts were negotiated for block and film mica from India, Brazil, and South Africa. In 1953–54 a sharp decline in demand for nonstrategic mica, which constitutes by far the largest portion of production, caused a decline in the production of strategic mica. Tiein purchases of nonstrategic mica with purchases of strategic mica for stockpiling partly arrested the decline in production.

SOURCES OF STATISTICAL DATA

Three Government agencies and three trade organizations compile statistics on various phases of the mica industry. The Bureau of Mines collects statistics from domestic producers of crude mica, fabricators of sheet mica, and mica grinders. Data on imports are compiled by the Bureau of the Census, Department of Commerce, and the United States Tariff Commission and on exports by the Bureau of the Census. National Electrical Manufacturers Association, Mica Fabricators Association, and Wet-Ground Mica Association issue monthly releases on the member companies' business transactions.

OUTLOOK

The demand for muscovite block and film of capacitor and tube quality is expected to continue high. It is unlikely that the supply of the best qualities of muscovite block and film will be adequate to meet all requirements for which mica is the preferred material, hence there probably will be an increasing trend toward the use of substitute dielectrics whereever possible. On the other hand, the development of jet engines and related equipment for civilian and military use created new demands for high-quality muscovite block and film for which no substitutes are known. Electronic components for industrial servomechanisms also may increase greatly the requirements for mica.

No shortage of muscovite or phlogopite splittings is anticipated. Muscovite splittings are essentially a byproduct of Indian muscovite block and film production, and the supply is largely a question of price. The supply of phlogopite splittings, which is obtained principally from Canada and Madagascar, is adequate. Improvements in processes for making builtup muscovite-mica products from domestic scrap probably will lead to gradual expansion of reconstituted mica production. If the supply of muscovite splittings from overseas, is curtailed, an expansion program for reconstituted mica would have a strong chance of success to eliminate the need for Indian splittings. The outlook for the replacement of Madagascar phlogopite splittings is not clear. Although Canadian phlogopite splittings are not widely acceptable as a substitute for Madagascar phlogopite splittings, World War II experience showed that Canada could fill a large part of the gap created by the curtailment of receipts from Madagascar. The Canadian splittings production also was supplemented by splitting Canadian phlogopite block in Mexico. Canada probably again could expand its phlogopite-splittings production at an appropriate price level, and reconstituted scrap-phlogopite products might be developed to satisfy at least a substantial part of the phlogopite-splittings requirements.

Domestic production of strategic qualities and grades of muscovite block mica probably will be negligible without heavy subsidization. The production of domestic punch, circle, and washer mica and of flake and scrap mica should continue to supply most of the United States requirements.

Continued research on synthetic mica and on mica substitutes should result in improved dielectric materials and finally in successful synthesis of large mica crystals. However, the need for large mica crystals may be minimized by development of reconstituted synthetic mica products.

Improvement of testing techniques may result in greater utilization of the lower qualities of mica for purposes now believed to require only the best qualities. This would release more high-quality mica for uses for which no substitutes are available.

PROBLEMS

The problems of the mica industry are considered in two separate categories—sheet-mica problems and scrap-mica problems.

SHEET-MICA PROBLEMS

The most important sheet-mica problems are concerned with the need for insuring adequate supplies of strategic mica or acceptable substitutes to meet essential military and civilian requirements. The dependence of the United States on uncertain and possibly indefensible sources for the greater part of its strategic requirements for mica block, film, and splittings forces consideration of all feasible means of expanding supplies from domestic and nearby sources, of utilizing most efficiently the available supplies, and of developing substitutes.

It has been shown that, even under heavy subsidies, only a small part of the United States requirements of strategic-quality mica can be produced from domestic sources. It is necessary therefore to establish an economic and strategically sound balance among subsidized domestic production, standby resources and facilities for an emergency, stockpiling, peacetime and wartime foreign production expansion programs, and import policies.

Solution of the problems of both strategic and nonstrategic sheet mica require more comprehensive economic and technologic studies to obtain data on domestic and foreign mica resources, to determine the most efficient methods of prospecting, exploration, mining, and preparation and to develop improved methods of recovering and utilizing byproducts and coproducts.

Conservation of the limited supplies of the higher qualities and larger sizes of sheet mica may be accomplished by developing adequate scientific methods for selecting mica for specific applications, thus confining consumption of the higher qualities to the most essential purposes. Conservation also would be favored by adjusting price differentials between the grades and qualities of mica to induce consumers to conserve the better qualities and larger sizes for more strategic end uses.

The problem may also be attacked by redesigning electronic equipment to use lower qualities of sheet mica. There is a need for more precise data on the performance characteristics of such mica under the conditions it is required to meet.

Consideration must also be given to the possible need for maintaining trained personnel to split and process sheet mica in an emergency that might cut off the supply of sheet mica now prepared for fabrication by low-cost foreign labor.

Continued research on synthetic-mica and sheet-mica substitutes is needed to develop materials to compete in cost and quality with natural sheet mica, thereby lessening the United States dependence on sources of supply that may be inaccessible in an emergency.

It is logical to assume that the development of substitutes will lessen the demand for the lower mica qualities and cause a rise in the price of strategic mica to compensate for the loss of the market for splittings and offgrade mica, since the present relatively low price of strategic mica from foreign sources depends heavily upon the marketing of splittings and nonstrategic qualities of block mica. Unquestionably this is a possibility that must be carefully considered and emphasizes the importance of making every effort to develop substitutes for strategic sheet mica rather than substitutes for mica of lower qualities.

SCRAP-MICA PROBLEMS

Scrap-mica problems revolve principally around the more economic production of scrap mica and the development of new markets. Research is needed to develop improved mining and processing methods for decreasing production costs and for recovering salable coproducts and byproducts.

Although known domestic resources of scrap and flake mica are adequate for many years at current levels of consumption in the long-range view, there is need for the evaluation of domestic scrap-mica resources, with special reference to the requirements for specific industries, to the amenability of the deposits to economical mining and milling techniques, and to the potential recovery of coproducts and byproducts. For example, the specifications of scrap mica for use in the production of reconstituted mica probably will differ from those for scrap mica for other uses.

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MINERALS RESEARCH—UNCLASSIFIED

By

Sidney Gottley[1]

ORGANIZED research has made possible the vast technologic complex that characterizes civilization in the 20th century. To maintain and advance that complex, the secrets of unexplored fields of science must be probed. Minerals research delves into some of these fields to insure adequate supplies of minerals, metals, and fuels—the basic raw materials of industry. Such research seeks new mineral raw materials, improved utilization of currently used materials, and the development of latent resources.

Summary

Unclassified minerals research attacks problems not directly related to specific commodities.

The growing complexity of the technology on which modern civilization is based demands the diligent pursuit of minerals research to meet the mounting requirements of the future for supplies of minerals, metals, and fuels. The ultimate objective of minerals research is to discover means of extracting all needed values from the earth's crust, the atmosphere, and the oceans. Many possible scientific and technologic fields may be explored by unclassified minerals research.

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[1] Executive Assistant to the Assistant Director—Programs, Bureau of Mines.

BACKGROUND

Unclassified minerals research is restricted to the solution of problems not directly attributable to a commodity described elsewhere in this bulletin or to problems whose solutions are applicable to many, if not all, mineral commodities. Although this category of research is predominantly fundamental or basic in its character—that is, it seeks the solutions to problems which will add knowledge to various fields of science without reference to specific applications—it may also include activities conducted to solve definite technologic problems arising from the production, treatment, or utilization of minerals, metals, and fuels.

IMPORTANCE OF MINERALS RESEARCH

Minerals, metals, and fuels have been used to maintain and improve the standards of living of mankind since prehistoric times. During the past century the quantity of mineral commodities used in this country has increased tremendously, largely because of the discovery of new and better usage. Per capita consumption of various commodities continues to multiply, pacing the increasing mechanization of modern society. This has been made possible by improved and augmented technology in utilizing minerals, metals, and fuels.

The mineral industries, because of their basic position as suppliers of raw materials, have heavy responsibilities. Not only must they provide the everyday raw materials needed by industry but also materials of continually higher quality and with new and better properties to satisfy the requirements of technologic advances in the host of items used in a complex civilization. At the same time, the economics of the industries, like those of any other business, must be so maintained as to preserve the competitive earnings position of the producing companies. The mineral industries must do their best to keep their investment and operating costs low, so that their products can also be kept low in price.

Today, greater emphasis is being placed upon technology than ever before. A far greater effort is going into the development of new products and into research that has yielded new technology. By this means, this country has been able to step up the output of its mines and mills to meet the demands of an expanding economy.

Technology, however, cannot advance without basic scientific knowledge. The advances that mark the high level of American civilization—its industry and weapons, its machinery and transportation—would have been impossible without a foundation of knowledge accumulated by means of fundamental research.

The availability of reserves of usable source material is a primary factory in maintaining healthy, flourishing mineral industries. To the extent that minerals research provides additional reserves by making it possible to employ sources now unusable, ranging from only slightly submarginal deposits to portions of the earth's crust that contain only very small percentages of various elements, minerals research contributes to the well-being of the mineral industries. Because such research will impart much greater value to hitherto valueless material, it contributes to the wealth of the Nation.

TRENDS IN MINERALS RESEARCH

There is no need to review here the inventions and discoveries in minerals, metals, and fuels that have marked the progress of civilization. It should be noted that systematic methods of scientific research did not appear until the 17th century. Outstanding developments in the 19th century created whole new fields of scientific thought, and scientific institutions of learning came into being to meet the demands of the changing environment.

Technologic advances in the mineral industries have frequently resulted from empirical methods of study. Technology persists today that is based on scientific theory and principles whose origin is literally lost in antiquity. This has been particularly true of mining and mineral dressing. Metallurgy, while equally old, shifted to investigation of more fundamental scientific theory in the past half century.

Accordingly, it does not suffice to confine minerals research to preconceived applications, because applied research too soon runs out of ideas to develop. Industry is now using basic information from fundamental research as soon as it becomes available. Continuing fundamental research is needed to provide a reservoir on which applied research can draw. Obviously, such fundamental research must be relatively free of detailed control and the restrictions of narrow assignments. The degree to which many scientific fields have already been explored in the course of the rapid development of our industrial civilization since 1900 makes fundamental scientific research in the mineral industries imperative. The obvious has already been prospected. Scientists must turn to new fields for the findings that will permit utilization of more and more of the materials to be found in the earth's crust, its atmosphere, and its oceans.

The peculiar problems of the mineral industries make it essential that research of a self-contained, specialized aspect be carried on almost independently of that done in other fields. That is, mineral industries must depend upon mineral-industry research, using, of course, developments in other fields that may be valuable.

OBJECTIVES OF INDUSTRIAL RESEARCH

The basic criteria for industrial research in the mineral industries, as in all industries, are of relatively short-range economic value. Industry expects that its research programs will eventually increase sales of old or new products and decrease production costs of such products. Industry must make a profit.

OBJECTIVES OF GOVERNMENT RESEARCH

Since Government research lacks a profit motive, federally sponsored and financed research on minerals, metals, and fuels is directed toward serving the national interest. It may have long-range or nonspecific product goals. It may seek to develop and insure immediate and future supplies of strategic commodities; to foster conservation; and to develop supplies, markets, and uses for hitherto unused minerals and metals with peculiarly valuable properties to the point where industry can profitably take over.

ULTIMATE OBJECTIVES OF UNCLASSIFIED MINERALS RESEARCH

The ancient argument that the last frontier has been reached and there is no room for further growth and expansion may be true, as far as opening up new lands in this country is concerned. It may be true—though some authorities are dubious—for the possibilities of finding vast, rich, new mineral deposits that will bring a rebirth to the tremendous minerals industries founded on the bonanzas of the 19th century. It is probably true for many depleted deposits, worked by timeworn methods, producing materials designed for an outmoded technology.

But it will never be true for scientific research in the mineral industries. Dwindling reserves, growing population, expending industry, new and improved technology in other industries, all the increasing complexities of advancing civilization challenge research workers to move forward across new frontiers of scientific knowledge in search of the solutions to the ever-lengthening list of problems.

Horizons must be broadened to the ultimate degree if these challenges are to be met. The concept of "ore" must be changed to fit the requirements of an expanding civilization that is continually becoming more complex. The eventually acceptable concept might well read:

> The mere presence of recognizable aggregations of metallic or nonmetallic compounds does not, in itself, constitute a mineral deposit, since such aggregations may only be termed ore when the occurrence is distributed in a manner that will permit controlled use in a natural state or in concentrations rich enough to permit their adaptation to man's needs, real or imaginary, at a civilization's then current cultural level.

Thus far, science and technology have learned to use only a fraction of the elements and substances that constitute the physical world. The problems that must be solved by the mineral industries in the future will necessitate constant expansion of the basic sources of mineral raw materials.

The ultimate goal will be achieved when means have been found to extract and utilize all elements and compounds from even the commonest and most abundantly widespread rocks, soils, and waters and from the atmosphere.

OUTLOOK

Technology may well be the deciding factor in the future of most minerals, metals, and mineral fuels. Since the results of research determine future technology, noncommodity-minerals research will decide whether or not latent resources, new mineral raw materials, and improved utilization of existing raw materials can be developed.

Research expenditures today are many times greater than they were 20 years ago, and the trend continues upward. Representative increases (approximate) in mineral-industry spending for research in 1952 over 1951 include: Petroleum and coal products, stone, clay and glass—14 percent; and primary metals—11 percent. Analysis of research-growth curves, attitudes of industry, Government, and the public, planning for future industrial operations, and other factors point to continued and probably increased emphasis on research.

It is not unreasonable to assume that fundamental and unclassified minerals research will receive its share of attention.

PROBLEMS

A number of fields of unclassified minerals research are described below to illustrate the broad avenues of approach that exist. These are, in the main, problems in applied research, but in attacking them, innumerable opportunities for basic research will arise.

Labor-force problems are many in the industry, as in other industries. Because operating costs are rising, minerals research, on mechanization in mining particularly, is imperative to increase productivity per man, decrease costs, and make working conditions in mining more attractive.

New methods of *mining narrow veins* of various minerals must be developed. It cannot be done simply by applying known methods to mining conditions at specific locations.

Longwall mining may have additional applications in several places in the United States, notably the Carlsbad potash field and the White Pine mine in Michigan.

Much excellent work has been done on the problems of *mining thick-bedded deposits* at the Bureau of Mines Experimental Oil Shale Mine, Rifle, Colo. Many procedures developed there may have application in mineral mining.

Solution mining may be applicable to other soluble minerals and perhaps to metals also, as well as to those now mined by such methods.

Quarrying methods for producing dimension stone have been relatively static for many years. The application of new principles could do much to improve efficiency and lower costs.

There have been important developments in *block-caving methods* in the past 20 years, but there are further applications to be developed where block caving, as now practiced, is considered unsafe or uneconomic. Mining research is extending its application to a broader field.

Research could do much to improve efficiency in *transportation of ore* by various methods. Transportation through pipelines is particularly promising.

Research on mine supports offers a promising field. *Rock bolting in metal and nonmetallic mines* is gaining in use and further research will extend the application of the practice. *Roof bolting in coal mines* has made practicable the adoption of more efficient mining practices.

In the 1920's and 1930's, the Bureau of Mines issued many publications on *mining methods and costs* that proved to be of great value to the industry. Because of the great changes that have taken place in mining practice, studies of mining methods and costs in current practice would be very valuable. Data on haulage, pumping, underground loading, development practices, drilling and blasting, etc., would be most useful. Studies of trends in mining and metallurgical practices may prove particularly enlightening.

Improvement in *mine lighting underground* could make conditions safer and more attractive for labor. In English coal mines, lighting conditions have been improved remarkably, with a very favorable effect on the working force.

There is an excellent field for auxiliary research projects in which existing information in the form of *abstracts, bibliographies, and similar reports* is compiled to summarize the present state of knowledge or present practice in a particular phase of the minerals industry.

The Bureau of Mines and others have investigated the most effective use of *diamonds in core drilling,* but many problems remain unsolved.

Despite many years of research on the design and testing of *drilling equipment* and improved *drilling techniques* for varying conditions, there is much room for improvement in core recovery and drilling efficiency. Blasting techniques also invite investigation.

Abroad, the practice of *mineral sampling* and the mathematical-statistical theory of sampling are recognized as being related. In the United States, relatively little interest in sampling theory has been displayed, although it is widely employed in agriculture, biology, and in various large manufacturing and business enterprises. Instead, mineral deposits and metallurgical processes are sampled according to established practice, usually of an empirical nature.

Development of *new methods of exploration* is essential to offset the declining grade of ore and the increasing depths at which new deposits may be found that are making present methods obsolete. The development of data on *technologic and economic trends in exploration and development* is a valuable function, providing background information for research and for industrial practice.

A variety of current research projects seek to apply *physical principles to mining.* These include: (1) Mine-structure analysis; (2) determination of the physical properties of mine rock; (3) blasting research; and (4) research on the development of instruments for research on mining problems. These projects are concerned largely with mineral mining, in which there is a very broad and fertile field for such fundamental research.

When other industries require materials with unique properties, minerals research is charged with the location and development of such materials. Because of this, increased recovery of minor metals from ores and concentrates of other metals can be expected. As the search for more and more mineral raw materials is intensified, *low-grade sources,* such as mine waters, dumps, crude oils, industrial wastes, and sea water, may move into the category of economically acceptable sources. Research on recovery processes is an essential phase of the attack on these problems.

Surface changes, such as those that may be induced by boiling or dry heat treatment, can change the *flotation characteristics* of some minerals. The effects of heat treatment and controlled dehydration on the floatability of minerals, in an attempt to discover selective procedures, should be investigated.

Investigations to obtain more information on the settling, thickening, and filtering of *gelatinous chemical precipitates* and products would be desirable. Other mineral-dressing research projects worthy of attention are studies that will relate *flocculation and dispersion of ore pulps* to the selective recovery of certain mineral particles. *Dry methods of ore beneficiation* should be investigated, since many mineral deposits occur in arid or semiarid localities. Improvements in dry grinding, air classification, and dry tabling would be helpful.

More attention should be paid to decreasing the costs of fine crushing and grinding by coarse-mesh *scalping of low-grade impure bulk concentrates before flotation* to permit discard of the bulk of the feed material at an early stage. This is now done effectively in heavy-medium separations. Requirements for liberation of valuable minerals may restrict this technique, but much wider application should be possible. At the opposite end of the range of sizing problems is that of *recovering slimes* (below 20 microns in size), which cannot be achieved by flotation. This is a most serious problem.

The organic chemical industry is growing rapidly, and new chemicals are constantly being placed on the market. Research to determine the usefulness of *new chemicals in mineral dressing* is indicated.

Grinding, perhaps the most expensive and least efficient phase of ore preparation, is still relatively primitive, since it is based on the crude principle of striking a piece of ore with another piece or with a piece of another material. Application of an entirely new principle, possibly based upon research in *solid-state physics,* is needed.

Preconditioning the surfaces of minerals has broadened the field of application of *electrostatic separation.* Further study holds promise.

Some aspects of *hydrometallurgy* are currently under study. *Ion exchange, liquid-liquid extraction, chromatographic separation,* and high-temperature *pressure-leaching* or *pressure-digestion,* are processes on which additional research should be rewarding.

Several other techniques are promising areas for research. These include:

(1) Recovery of pure metals or alloys by high-temperature treatment under vacuum.

(2) Development of basic techniques, such as distillation, filtration, and centrifugation for separating metals and alloy constituents.

(3) Study of the properties of fused electrolytes, including the properties of conductance and resistance and the mechanics of electrical deposition and/or transfer of ores and metals.

(4) Measurement and correlation of the basic thermodynamic data of minerals, metals, and fuels and their application.

(5) Production of pure powder metal by electrolysis.

(6) Studies of crystal growth in nonmetallic, inorganic systems, including the relationship of physical properties to crystal structure and isomorphism.

(7) The fundamentals of mineral separation in various types of existing or still-to-be-designed types of equipment.

(8) Development of faster and more accurate methods of quantitative and qualitative analysis of mineral-industry products.

(9) Research on roasting of ores.

(10) Fundamental studies of catalysis and catalysts.

(11) Investigation of the fundamentals of corrosion.

(12) Cryogenic research on metals and minerals.

Some new tools of research have possible applications in several fields of minerals research.

Ultrasonics, the science of acoustic or compressional waves at frequencies above the limit of human hearing (approximately 20,000 cycles per second), applies frequencies up to millions of cycles per second that can bring about changes in molecular structures and other properties. There are many potential applications in the mineral industries.

With the help of *radioactive isotopes,* the location of materials and conditions can be accurately traced through the various steps of a process. In minerals research, radioactive tracers can be useful in diffusion studies, investigation of alloy systems, studies of synthetic substitutes and crystal systems, and following the role played by reagents or the course taken by particular minerals in a separation. Numerous other possible applications exist.

The *effects of radiation* from nuclear fission on minerals, metals, and fuels should be studied.

Rheology, the study of the deformation and flow of materials, should not be overlooked for possible applications in the minerals industries.

Research on the *transfer of heat* from one substance to another in any state is needed to increase process efficiency. Of value also, would be investigation of the *fundamentals of the production of heat* by combustion, friction, etc.

Economic research that will overcome the problem of predicting patterns of technologic trends, costs, prices, transportation, and related areas is much needed. The results of such research will serve not only as a guide to mineral-industry operations and technologic research but also as a basis for establishing Federal Government policies and programs.

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MOLYBDENUM

By

Robert W. Geehan [1]

ALTHOUGH a little molybdenum goes a long way, the United States, which produces the lion's share of the world's supply, has recently completed a production-expansion project to meet emergency needs.

Summary

Molybdenum, a metal largely used as an alloying addition in steel and cast iron, is an exception to the domestic supply-demand pattern of most ferroalloy metals because the United States produces far greater quantities than all other nations combined, and exports rather than imports are the rule. Use of this element has increased spectacularly since World War I, when information on alloys containing molybdenum first became well known. Production is from molybdenum mines and from copper mines as a byproduct. Outstanding among the former is the vast deposit at Climax, Colo., where large-scale caving methods are used in mining; the leading byproduct source is the Bingham, Utah, copper mine, where ores are mined by opencut methods on an even larger scale. Both sources produce a molybdenite (MoS_2) concentrate, using the flotation process; nearly all such concentrate is roasted before final use. The price of molybdenum concentrate has been quoted at 60 cents per pound of MoS_2 since 1950. At present, production is ample to cover defense and essential civilian needs and allow shipments to the National Strategic Stockpile and to friendly nations.

A large degree of substitution is practical in high-speed steels, either molybdenum for tungsten or the reverse. Boron steels can be substituted for certain molybdenum steels. About 70 percent of the molybdenum used in the United States is consumed in steels and about 15 percent in cast iron and malleable castings; the remainder is distributed among nonferrous alloys, metallic molybdenum, and compounds.

A Government program designed to assure ample supplies for an all-out emergency is in progress; the chief factor is a long-term purchase contract with Climax Molybdenum Co., which has provided for a large additional capacity to produce. A slight decrease in the molybdenum content of several widely used alloy steels has been an effective conservation measure.

The long-term outlook is for a surplus of productive capacity, except during periods of major war demands; this probably will be true for several years, even though an increased industrial demand is expected.

Wide swings in demand create several of the problems of the molybdenum industry. Those of a more technical nature include the need for improved methods to produce concentrates and primary products and a satisfactory method for protecting molybdenum from oxidation at high temperatures.

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[1] Assistant chief, Division of Minerals, Bureau of Mines.

BACKGROUND

Molybdenum is a metal with atomic weight 96.0 and is element number 42. It was discovered by Scheele in 1778, and metallic molybdenum was first produced in 1782. Some experimentation with molybdenum alloys was conducted late in the 19th century, but no appreciable use of the metal developed until World War I. Table 1 indicates the spectacular growth of the industry in the United States, and table 2 lists salient statistics from 1939 to the present.

TABLE 1.—*Production of molybdenum ore and concentrates in the United States, before 1914 through first 10 months of 1954*

[Thousand pounds of contained molybdenum]

Year	Production	Year	Production	Year	Production	Year	Production
Before 1914	[1] 350	1931	3, 133	1939	30, 324	1947	27, 047
1914–18	[1] 1, 602	1932	2, 431	1940	34, 313	1948	26, 706
1919	397	1933	5, 682	1941	40, 363	1949	22, 530
1920–25	[1] 2, 000	1934	9, 362	1942	56, 942	1950	28, 480
1926	1, 393	1935	11, 512	1943	61, 667	1951	38, 855
1927	2, 299	1936	17, 186	1944	38, 679	1952	43, 259
1928	3, 428	1937	29, 419	1945	30, 802	1953	57, 243
1929	4, 021	1938	33, 297	1946	18, 218	1954: Jan.–Oct. inclusive	46, 984
1930	3, 723						

[1] Estimate.

TABLE 2.—*Salient statistics of molybdenum concentrates in the United States, 1939–54*

[Thousand pounds of molybdenum contained in concentrates]

Year	Production	Shipments (including exports)	Exports [1]	Imports for consumption	Consumption	Stocks at end of year [2]
1939	30, 324	32, 415	(3)	26	(3)	(3)
1940	34, 313	25, 329	6, 585	----------	(3)	(3)
1941	40, 363	38, 377	7, 640	4	[4] 16, 891	21, 226
1942	56, 942	66, 437	11, 597	756	56, 388	12, 540
1943	61, 667	53, 955	10, 071	1, 572	49, 891	17, 993
1944	38, 679	39, 423	5, 985	2, 354	31, 520	19, 321
1945	30, 802	33, 683	2, 863	204	32, 696	16, 883
1946	18, 218	16, 787	565	(5)	14, 994	19, 275
1947	27, 047	22, 190	2, 607	----------	20, 221	23, 661
1948	26, 706	29, 669	3, 525	----------	25, 156	21, 206
1949	22, 530	23, 280	4, 287	48	19, 960	19, 159
1950	28, 480	44, 544	5, 386	3	26, 029	4, 326
1951	38, 855	37, 955	3, 270	4	33, 691	5, 058
1952	43, 259	42, 717	5, 290	50	32, 715	6, 856
1953	57, 243	53, 823	5, 893	None	31, 193	11, 326
1954: 1st 10 months	46, 984	55, 526	10, 450	None	19, 576	----------

[1] Exports include roasted concentrates 1940–46, shipments of raw concentrates for export 1947–54.

[2] At mines and at plants making molybdenum products.

[3] Data not available.

[4] 6 months, July–December.

[5] 10 pounds.

Molybdenum production is concentrated in the United States (92 percent in 1953) and Chile (5 percent in 1953). Consumption, gaged by 1953 shipments is about as follows: United States 74 percent, United Kingdom 13 percent, France 4 percent, Germany 4 percent, Japan 2 percent, and Canada, Netherlands, and Sweden each 1 percent.

NEW SOURCES

Byproduct molybdenum may be produced from the Gaspé copper deposit, Canada, after 1955; no reliable estimate of the quantity expected is available. Production at the La Corne mine, Quebec, will be expanded because of a Government contract, but production will probably require favorable prices. Deposits have been reported to exist in Mexico that may be worked in future years; a small output was begun in 1953. Eventual development of sulfide ore at the Chuquicamata mine, Chile, may lead to byproduct-molybdenum production. New domestic byproduct sources may include the San Manuel and Mineral Hill-Plumed Knight mines in Arizona.

TABLE 3.—*Salient statistics of molybdenum products*[1] *in the United States, 1945–54*

[Thousand pounds of contained molybdenum]

Year	Production	Shipments		Stocks (producers), end of year
		To domestic destinations	Exports [2]	
1945	32, 406	26, 977	1, 327	10, 176
1946	15, 039	16, 502	442	8, 211
1947	20, 660	19, 878	866	8, 126
1948	24, 445	23, 809	1, 216	7, 547
1949	19, 624	15, 019	1, 314	10, 838
1950	25, 348	32, 736	1, 955	1, 495
1951	32, 775	29, 845	1, 388	3, 037
1952	32, 383	30, 211	1, 844	3, 373
1953	30, 283	29, 595	1, 107	3, 894
1954: 1st 10 months	19, 250	18, 846	1, 170	----------

[1] Comprises ferromolybdenum, molybdic oxide, and molybdenum salts and metal.

[2] Reported by producers to the Bureau of Mines.

GEOLOGY

The only important production at present is from deposits containing the mineral molybdenite (molybdenum sulfide, MoS_2); however, some powellite (calcium tungstomolybdate, $Ca[MoW]O_4$) is treated to recover molybdenum, and in the past deposits containing wulfenite (lead molybdate, $PbMoO_4$) were worked.

Molybdenum deposits can be classified into the following types: (1) Veins, (2) massive silicified replacement, and (3) copper deposits containing molybdenum. Nearly all the production is from types 2 and 3. Questa, New Mexico, and Knaben, Norway, are examples of vein deposits; Climax, Colo., is typical of massive silicified replacement; Bingham, Utah, and Braden, Chile, are examples of the third type. All important deposits have been described in readily available publications listed in the bibliography.

MINING

Mining methods used range from small cut-and-fill stopes in vein deposits to large-scale caving at massive, silicified, replacement ore bodies. The caving system in use at Climax is considered one of the best examples of that method now in use. The nature of the ore body led to caving inclined areas over a vertical distance exceeding 800 feet; this required the use of long raises for ore passes and for development before undercutting of the upper zone. The use of the slusher system for transporting ore in slusher drifts is standard practice, as is extensive concreting of these drifts and the lower portions of finger raises.

Development work recently completed at the Climax mine provides for daily production of 27,000 tons of ore; output comes from two main haulage levels.

MILLING

The flotation system is universally used in the production of molybdenite concentrates. At Climax a rough concentrate is produced with a ratio of concentration of 10 or 12:1; it is reground and refloated to produce a final concentrate. The overall ratio of concentration is about 190:1. A recent addition to the Climax mill has been a byproduct plant for recovering tungsten.

At plants producing molybdenum as a byproduct of copper mining a concentrate containing both copper and molybdenum is first produced; the molybdenite is then separated from the copper minerals by differential flotation. As copper is considered an undesirable impurity in molybdenite concentrates, this requires a well-designed and efficiently operated unit.

Government specifications for molybdenite concentrates are as follows:

Molybdenum disulfide, MoS_2 (molybdenite)

Each lot of molybdenite shall conform to the following:

		Percent by weight molybdenite
Molybdenum disulfide (MoS_2)		80.00 minimum
Copper (Cu)		1.00 maximum
Lead (Pb)		.30 maximum
Phosphorus (P)	Combined	.20 maximum
Tin (Sn)		
Arsenic (As)		

The molybdenite content of concentrates shipped in 1951 ranged from 57 to 94 percent; 90 percent is considered as standard grade for price quotations.

MANUFACTURE OF MOLYBDENUM PRODUCTS

Essentially all molybdenite concentrates are roasted to produce technical-grade oxide. Large quantities of this material are charged to steel, and the balance is the raw material used to make other molybdenum products. Government specifications for this type of oxide are as follows:

Molybdic oxide

Each lot of molybdic oxide shall conform to the following:

	Percent by weight molybdic oxide
Molybdenum (Mo)	55.00 minimum
Copper (Cu)	1.00 maximum
Phosphorus (P)	.05 maximum
Sulfur (S)	.25 maximum

Molybdic oxide of higher purity is produced by subliming the technical-grade oxide or from ammonium molybdate.

Ferromolybdenum is manufactured from the oxide. At the plant of Climax Molybdenum Co. at Langeloth, Pa., where the thermite method is used, a typical charge is 1,300 pounds of molybdenum in oxide, 116 pounds of 93-percent aluminum, 1,122 pounds of 50-percent ferrosilicon, 618 pounds of high-grade iron ore, 160 pounds of lime, and 50 pounds of high-grade fluorspar. This mixture is ignited in a bottomless shell set over a sand pit and covered by a dust hood. The reaction is complete in 20 minutes; after the slag is drained, the button is cooled, quenched, and crushed to desired size. Government specifications for this product are as follows:

Ferromolybdenum

Each lot of ferromolybdenum shall be either Grade A or Grade B and conform to the following:

	Percent by weight	
	Grade A	*Grade B*
Molybdenum (Mo)	55.00 minimum	55.00
Carbon (C)	2.50 maximum	.25
Copper (Cu)	1.00 maximum	1.00
Phosphorus (P)	.10 maximum	.10
Silicon (Si)	1.50 maximum	1.50
Sulfur (S)	.25 maximum	.25

BYPRODUCTS AND RELATIONSHIP TO OTHER PRODUCTS

The most important byproduct of molybdenum is tungsten; this is produced only at Climax, Colo. In 1953, 413,000 pounds of tungsten was produced along with some pyrite, tin concentrate, monazite, and topaz. Rhenium is obtained from molybdenum plants as a byproduct; the original source is copper ore containing molybdenite.

Output of domestic mines operated solely or almost solely for molybdenum was 37,421,000 pounds in 1953, whereas byproduct concentrates from copper and tungsten operations totaled 19,822,000 pounds. Chilean production was entirely a byproduct of copper mining; in 1953 this was estimated at 3,031,000 pounds.

Fluctuations in demand are largely paralleled by fluctuations in production at molybdenum mines; the output of the byproduct sources is not directly related to demand for molybdenum. During periods when molybdenum consumption is low, byproduct sources may produce much of the molybdenum used.

SUBSTITUTES

Molybdenum is substituted for tungsten in certain high-speed steels; the reverse is also true. Boron steels have been developed that replace steels containing molybdenum used primarily for its contribution to hardenability; the degree of such substitution that eventually will result is subject to debate. In general, the tendency has been to use more steels containing molybdenum in defense periods, partly because the supply situation led to its use as a substitute for tungsten and partly because of greatly expanded use of alloys containing molybdenum.

SECONDARY SOURCES

High-speed steel scrap represents over half of the charge at plants producing high-speed tool steel. Scrap is not a very important source of molybdenum at the plants producing other alloy steel. Some molybdenum is added to high-speed steel as a minor constituent of tungsten concentrates. This is not a true secondary source but is metal and is not covered in statistics of molybdenum production. All published data of the Bureau of Mines on molybdenum consumption refer to primary metal.

ORE RESERVES

The domestic reserves estimated by the Bureau of Mines and Geological Survey, as of 1943, were as follows:

	Molybdenum (metal) content, thousand pounds
Availability:	
1. Commercial under conditions similar to those of 1943	2, 600, 000
2. Potentially commercial and marginal	900, 000
3. Submarginal, more than	3, 500, 000

In group 1, the grades of ore generally exceed 0.5 percent of molybdenite (molybdenum sulfide, MoS_2), except for the byproduct molybdenum in copper mining, in which the grades range from 0.02 to 0.2 percent of MoS_2. Although the grade of some ores included in group 2 exceeds 0.5 percent of MoS_2, most of the ore is below this grade, but none with a cutoff less than 0.25 percent of MoS_2 is included, except in the byproduct class. In group 1 and 2 the reserves are considered measured and indicated, except insofar as the byproduct reserves of the larger copper deposits may be classed as inferred.

The reserves in group 3, which are largely inferred, include large deposits in several Western States and Alaska containing 0.05 to 0.25 percent of molybdenite, vanadiferous shales, and partly explored areas adjacent to the principal known molybdenum deposits.

The commercial, potentially commercial, and marginal reserves of molybdenum in the United States are equivalent to over 400 years' supply at the average rate of domestic consumption from 1935 to 1939 and nearly 100 years' supply at the wartime rate. They are equivalent to the output of about 140 years at the prewar rate of production.[2]

The statistics listed above are still usable, as ore mined since 1943 has largely been replaced by newly developed ore.

USES

Of the total molybdenum used in the United States, it is estimated that about 70 percent is now consumed in steels, 15 percent in cast iron and malleable castings, and the remaining 15 percent in the pure metal form and as compounds. The pure metal is used in electrical, electronic, and high-temperature applications. Compounds are used as fertilizers, in pigments, in the ceramic industry, and as catalysts.

Molybdenum is used so universally in alloy steels that it is difficult to list the types and applications. Modern usage ordinarily calls for remarkably small quantities of this element per ton of steel; an exception is high-speed steel, where 6 percent Mo is a popular composition. Additions in the range 0.1 to 0.5 percent Mo are common in many alloy steels that also contain chromium and nickel. An important class of molybdenum-bearing alloy steel is designed for service at moderately high temperatures, such as those prevailing at oil refineries; these alloys contain 0.5 to 1.5 percent Mo in most instances.

Molybdenum is used in steels to promote uniform hardness and strength, to reduce softening when tempering, to reduce the tendency of certain steels to become brittle after tempering, to increase strength and creep resistance at elevated temperatures, to retard embrittlement in steels subjected to stress at temperatures ranging from 600° to 1,000° F., to aid corrosion resistance, to impart red-hardness to high-speed steels, and, along with other alloying elements, for many special applications. In cast irons molybdenum increases the tensile strength, promotes uniform strength in castings with light and heavy sections, improves high-temperature strength, improves resistance to chipping, and improves hardenability of heat-treated castings. In both iron and steel the ability to use a very small quantity of molybdenum to yield the desired effect minimizes the cooling action at the time the alloying element is added to the melt.

SOURCES OF STATISTICAL INFORMATION

The Bureau compiles data on production and consumption of concentrates, and production, shipments and stocks of primary products. The American Iron and Steel Institute publishes data on the molybdenum used by its members and information regarding alloy steels produced and shipped.

SELF-SUFFICIENCY AND STRATEGIC CONSIDERATIONS

The United States produced 92 percent of the estimated world production in 1953. Contracts designed to increase production and capacity to produce at Climax, Colo., and La Corne, Quebec, Canada, coupled with expanded production from byproduct sources, have led to a threefold increase in available supply.

These emergency programs are considered necessary because of the high level of consumption estimated for an all-out emergency period. Compared to most other ferroalloy metals, the United States supply of molybdenum is not critical because there is no uncertainty over imports; however, the United States may again be called on to export substantial quantities during a war, as was the case in World War II.

PRICES

Prices of molybdenum products have remained remarkably stable over a long period. For example, molybdenum concentrates (per pound of MoS_2) were quoted at 45 cents from 1938 to 1949, at 54 cents from 1949 to 1950, and at 60 cents since that date.

Prices of the principal molybdenum products are based on a pound of contained molybdenum, f. o. b. producer's plant. Prices in effect since 1950 are as follows: Molybdic oxide $1.14, calcium molybdate $1.15, ferromolybdenum $1.32, and molybdenum metal $3.00 to $5.50, depending on its form and quality.

TARIFF

The duty on molybdenum ore and concentrates is 35 cents a pound on the metallic molybdenum contained. The General Agreement on Tariff and Trade, Torquay, effective June 6, 1951, reduced the duty on ferromolybdenum, molybdenum metal and powder, calcium molybdate, and other compounds and alloys of molybdenum to 25 cents a pound of molybdenum contained plus 7.5 percent ad

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[2] Subcommittee of Committee on Public Lands, U. S. Senate, Hearings: 80th Cong., 1st Sess., May 1947, p. 269.

valorem; the former rate was 50 cents and 15 percent ad valorem.

TRANSPORTATION

All of the molybdenum mined in the United States comes from the Western States; nearly all the consumption is in the Eastern States. Transportation is seldom a problem in the sense of limiting production or shipments.

CONSERVATION PRACTICES

At the mines, efficient mining and milling have led to lowering the minimum grade of ore worked, and improved metallurgical practices have led to better recovery at both byproduct and molybdenum mines. A Government contract with Climax Molybdenum Co. provides for working low-grade fringe areas of the Climax deposit.

Dust-recovery units at primary products plants have increased overall recovery; however, the sulfur in the concentrates roasted is not recovered.

It has been found that slight decreases in the quantities of molybdenum added to some steels do not result in undesirable products; however, the ultimate cost of end products may be increased because of problems of fabrication when lean alloys are used. Research in this field is still in progress.

RESEARCH

At least part of the spectacular growth of the molybdenum industry can be attributed to research sponsored by the producing firms. The need for metals able to withstand the high temperatures of modern jet engines has led to much investigation of production of large ingots of very pure molybdenum and to an intensive study of protective coatings. The Bureau of Mines has recently investigated new techniques for producing exceptionally pure metal.

In addition to the constant investigation of alloys, much research continues on the use of molybdenum in lubricants, fertilizers, and catalysts.

OUTLOOK

TRENDS

The long-term trend for consumption is upward. The President's Materials Policy Commission report[3] estimates that consumption in the United States will increase to 70 million pounds by 1975 and that consumption in the rest of the Free World by 1975 will be 27 million pounds. Domestic consumption in 1953 was 31 million pounds.

Actually, estimates of consumption in future years are difficult because of the uncertainty regarding factors that may lead to use of more or of less molybdenum as related to other elements of the Nation's economic growth. For example, changes in relative use of boron steels, tungsten steels, specially heat-treated carbon steels, steels for high-temperature applications, and nonferrous alloys can be expected; but the degree and direction of the changes are impossible to predict.

Other free countries doubtless will greatly expand their consumption of molybdenum in future years. Many alloy steels containing the element that are well established in the United States are just now coming into use in Europe. The need for dollar exchange to cover purchases has held back foreign consumption, particularly in the high-speed steel field. Some nations find that tungsten can be obtained in nondollar areas with less strain on their economy than molybdenum from the United States, even though the prices would appear to favor the latter by a wide margin. However, exports during the first half of 1954 were at a rate more than at any time since 1943.

NEW USES

The use of molybdenite as a lubricant is a new development that may require 1,000,000 pounds (Mo content) by 1960. Present usage is about 200,000 pounds a year.

Many new uses for pure metal are in a development stage; if technical difficulties are surmounted, these may consume several hundred thousand pounds a year. A significant factor from a production standpoint is the ability to produce large ingots that can be forged, extruded, or rolled to large shapes for high-temperature applications.

The use of molybdenum in fertilizers is not new but is in a growth stage. These fertilizers are now in wide use in molybdenum-deficient areas of Australia; the officials of that country regard a supply for this use as more important than the alloy field. The quantities involved are very small; Australia used about 100 tons in 1952 for all purposes.

NEW SOURCES

Byproduct molybdenum may be produced from copper mines under development in Canada and the United States. A relatively small output from molybdenum mines in Canada and Mexico can be expected.

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[3] Vol. II, June 1952, p. 26.

GENERAL

After stockpile purchases are terminated, capacity to produce molybdenum will exceed demand, barring a full-scale war emergency. Byproduct output will depend on the level of copper production; molybdenum mines will limit output to cover the remaining demand. A gradual increase in requirements seems certain, so, after a period of several years, present productive capacity may be inadequate.

PROBLEMS

PRODUCTION

The needed ability to shift production rates to meet wide swings in demand is a major problem to producers. Retention of the present facilities for high production in standby condition for future needs will not be difficult, but skilled labor and adequate supervisory personnel may not be available on short notice after any extended periods of curtailed production.

The ability to expand production from known ore reserves leads producers into research designed to stimulate demand.

Ore-dressing methods, particularly at byproduct plants, are subject to constant study leading to improved recovery of a better grade product. Lack of a market for molybdenite concentrates containing a high percentage of copper may prevent byproduct recovery at some mines and decrease the percentage of recovery at others.

PRIMARY PRODUCTS

The same problem of capacity to overproduce applies at mines and at plants producing primary products.

A method of recovering sulfur from dilute mixtures of SO_2 and air now being wasted from the roasting of MoS_2 would create a new source of income and would be good conservation practice. New roasting techniques might serve the same end.

Production and fabricating methods for very pure metal and methods of protecting it from oxidation at critical temperatures are problems so important to producers and consumers that intensive research has been sponsored and much progress is being made.

From a long-term standpoint, conservation or productive capacity for future emergencies and expanded needs, conservation of resources, export policy, and maintenance of technical leadership are important.

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NICKEL

By

Hubert W. Davis [1]

EVER SINCE the superiority of armorplate made from nickel steel was discovered about 1889, nickel has been an important element in the construction of war materiel; but it is also a metal of peace, and some of the greatest developments in its use have been for things of everyday life.

Summary

The United States currently has been producing less than 1 percent of its nickel requirements but was expected to be producing nearly 5 percent by mid-1955. This unfavorable position with respect to production could improve if further exploration of certain sulfide deposits discloses substantially larger reserves and an economic process is developed for treating extremely low grade nickeliferous iron ores.

Despite the projected increase in production of nickel from Cuba and other sources, the Sudbury district of Canada will, as heretofore, continue for many years to be the predominant producing area.

Following the fulfillment of the National Stockpile objective the world capacity to produce nickel will exceed peacetime needs. However, the nickel industry's policies of aggressive research and development of new uses will, it is believed, result in finding markets for much of the expanded production. Nevertheless, development of additional markets for nickel has been retarded by the imposition of severe restrictions on use during and subsequent to the Korean conflict and on emphasis in public announcements on the scarcity of nickel. Moreover, the scarcity of nickel has resulted in a tendency to develop substitutes and to conserve the metal by downgrading.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

SIZE OF DOMESTIC NICKEL INDUSTRY

Very little domestic nickel ore had been mined, smelted, or refined in the United States before 1955. However, 2 plants—1 for nickel metal and 1 for ferronickel—were scheduled to begin production in the United States in 1955. At Huntington, W. Va., roasted and sintered matte from Canada is refined to Monel metal by the International Nickel Co., Inc. A few companies produce nickel salts.

Near Lancaster Gap, Pa., a lens of nickel-bearing sulfide at the Gap mine was worked for nickel from 1853 to 1893, when the mine was abandoned. Total production was about 2,000 tons. Before 1921 and in 1944 and 1945 small quantities of nickel were produced from the complex ores at Fredericktown, Mo. In 1941 matte containing 35 short tons of nickel was produced from ore mined near Gold Hill, Colo.; and in 1941–44 concentrates containing about 20 tons of nickel were recovered as a byproduct of talc production at Johnson, Vt. In 1953 and 1954 a small quantity of nickel contained in the cobalt ore of the Blackbird mine in Idaho was recovered. Otherwise, primary nickel has been produced domestically as a byproduct of copper refining.

Table 1 shows the production of nickel in the United States from 1910 through 1953.

TABLE 1.—*Production of nickel as byproduct of copper refining in the United States, 1910–53*

Year	Short tons, nickel content	Year	Short tons, nickel content
1910–19 (average)	[1] 498	1946	352
1920–29 (average)	[1] 329	1947	646
1930–39 (average)	246	1948	883
1940	554	1949	790
1941	[1] 660	1950	913
1942	[1] 612	1951	756
1943	[1] 642	1952	633
1944	[1] 988	1953	[1] 602
1945	[1] 1, 155		

[1] Includes nickel produced from ore and as byproduct of talc.

TABLE 2.—*Production of refined nickel salts* [1] *in the United States, 1947–53*

Year	Short tons, nickel content
1947	2, 126
1948	2, 038
1949	2, 309
1950	1, 926
1951	1, 722
1952	1, 668
1953	1, 837

[1] Excludes refined nickel salts produced as byproduct of copper refining.

GEOGRAPHIC DISTRIBUTION OF NICKEL INDUSTRY

At present the nickel industry is centered chiefly in the Sudbury district of Ontario, Canada, which for many years has supplied about 90 percent of the world (outside the U. S. S. R.) nickel ore. The International Nickel Co. of Canada, Ltd., operates mines in the Sudbury district, smelters at Copper Cliff and Coniston, Ontario, and refineries at Port Colborne, Ontario, and Clydach, Wales. The Falconbridge Nickel Mines, Ltd., operates mines in the Sudbury district, a smelter at Falconbridge, Ontario, and a refinery at Kristiansand, Norway.

Most of the remainder of the world (outside the U. S. S. R.) supply of nickel ore comes from Cuba and New Caledonia. The Cuban ore is mined in Oriente Province and is converted to oxide at a plant at Nicaro. The largest and most recent nickel-producing centers in New Caledonia are the Thio Group at Thio on the east coast and the Pin-Pin concession at Moindah, 130 miles north of Noumea. Most of the ore is converted to matte, ferronickel, and fonte; the matte is exported to Le Havre, France, for refining.

RESERVES

The following information on nickel reserves in the United States was prepared by the Bureau of Mines and Geological Survey and is quoted from a report on mineral position of the United States, published in the hearings before a subcommittee of the Committee on Public Lands, United States Senate, 80th Congress, first session, 1947:

The higher grade sulfide ores, except those at Fredericktown, Mo., occur in small bodies in widely separated parts of the country. Most of them are small and expensive to explore and too few to warrant establishment of a smelting industry. The larger bodies contain scarcely more than a few hundred tons of nickel in ore whose average grade is 1 to 3 percent. The potentially more important sulfide deposits are those at Yakobi Island, Alaska, and at Mouat, Mont., in which the larger and better grade bodies have an average nickel content of about 0.4 percent. Many of these deposits contain various amounts of copper, platinum, and cobalt, which enhance their value slightly.

The better grade silicate ores, as represented particularly by the deposits at Riddle, Oreg., contain 1 to 2 percent nickel and are low in

TABLE 3.—*Estimated nickel reserves in the United States as of January 1944*

[Short tons]

	Indicated [1]		Inferred		Total [2]	
	Ore	Nickel content [2]	Ore	Nickel content [2]	Ore	Nickel content
Sulfide ores:						
Alaska	11, 000, 000	40, 000	11, 000, 000	30, 000	22, 000, 000	70, 000
States	6, 500, 000	35, 000	7, 500, 000	35, 000	14, 000, 000	70, 000
Silicate and nickeliferous iron ores: States	15, 000, 000 to 20, 000, 000	175, 000	25, 000, 000 to 30, 000, 000	235, 000	40, 000, 000 to 45, 000, 000	410, 000
Total [2]	35, 000, 000	250, 000	45, 000, 000	300, 000	80, 000, 000	550, 000

[1] Includes measured ore.
[2] Round numbers.

iron. Although the potential tonnage of the nickeliferous iron ores of Washington containing nickel is large, the grade is low—about 0.5 percent nickel or less—and commercial methods of treatment have not been developed.

Utilization of each of the various types of ore involves special problems, and the various grades do not represent the relative worth of the ore. The Bureau of Mines has experimented with the silicate and iron ores and believes that virtually all the nickel can be recovered in a nickel-iron alloy.

With incentive for development and further exploration for these known types of deposits, it may be expected that considerably larger reserves than have been indicated in table 3 will be revealed. Not less than 10 times the currently estimated reserves is reasonably expectable from present knowledge of the distribution of deposits and the thoroughness with which they have been explored. In Alaska, particularly, only a few of the geologically favorable areas have been prospected for nickel. Some bodies known to contain nickel have not been sampled systematically.

Since the estimates in table 3 were prepared, nickel-copper deposits have been discovered in gabbro in Minnesota and Wisconsin. Exploration by private interests and inspections by the Bureau of Mines and Minnesota Geological Survey have indicated the possibility of enormous reserves of low-grade nickel-copper ore along the western contact of the Duluth gabbro in the Superior National Forest, Lake County, Minn.

The largest reserves of developed sulfide ores of nickel in the world are those of the Sudbury district of Canada. As of December 31, 1953, the proved ore reserves of the International Nickel Co. of Canada, Ltd., were 261,500,000 short tons containing 7,817,000 tons of nickel-copper, and the developed and indicated reserves of Falconbridge Nickel Mines, Ltd., were 34,571,000 tons containing 544,000 tons of nickel. At the 1953 rate of production, the reserves of these 2 companies were sufficient for 19 and 30 years, respectively.

Probably the second largest reserves of sulfide ores of nickel are in the Lynn Lake area of northern Manitoba, where Sherritt Gordon Mines, Ltd., and other companies have holdings. The small amount of exploration done by Sherritt Gordon indicated ore reserves of 14,055,000 short tons containing 172,000 tons of nickel.

Two nickel discoveries were reported made near Kluane Lake, Yukon Territory, in 1952. The Hudson Bay Mining & Smelting Co., Ltd., has staked and holds under option to purchase the Wellgreen property, which is favorably located close to good highways leading to tidewater. It covers an area roughly 12 miles long and 3 miles wide, comprising 538 mineral claims, which, on the whole, is generally mineralized and in particular contains a nickel-copper deposit with precious metals. Prospectors Airways Co., Ltd., was reported to have staked 28 claims in the area.

The largest reserves of nickel silicate ores in free-world countries outside the United States are in Brazil, Indonesia, New Caledonia, and Venezuela, but at present only the deposits in New Caledonia are being exploited. Figures on tonnage and grade of the reserves in New Caledonia have not been published; but according to La Société le Nickel, which owns the major deposits, the reserves are adequate to maintain an output of 12,000 metric tons of metallic nickel annually for 80 years.

In Brazil approximately 50 nickel silicate deposits occur in an area of 14 square miles north of São José do Tocantins, State of Goiás. Conservative estimates of reserves indicate that the district as a whole may contain as much as 16 million tons of ore averaging 1 to 3 percent nickel. About 1,400,000 tons of ore is available that has an average tenor of 4.5 percent nickel, with a possible yield from the district of as much as 5 million tons averaging 4 percent nickel. Small deposits of nickel silicate ore have been mined and smelted at Livramento in south Minas Geraes. There is a probable reserve of about one-quarter million tons and a possible reserve of 2 million tons having about 1.5-percent nickel content. Deposits at Ipanema in the same region contain about 200,000 tons of 1- to 2-percent nickel and a possible 2 million tons of the same grade.

The Celebes contains deposits of nickel silicate ore that have been known since 1909, but it has been only since 1938 that a small output of ore containing 2 to 3 percent nickel has been realized. Two principal regions having reserves of nickel ore are known: (1) The Mekongga (Bingkoko) Bay region and (2) the Lakes region, both lying on the eastern side of the Gulf of Bone in southeastern Celebes. In the vicinity of Pomalea, Kolaka, Pandamerang Island, and Maniang Island, geological conditions indicate a probable reserve of at least 2 million tons of 3-percent nickel and a possible reserve exceeding 10 million tons having a grade of 1 to 3 percent nickel. High-grade ore of 4-percent nickel content or better occurs irregularly in the order of magnitude of a few hundred thousand tons. In the Lakes region there are favorable prospects in the districts of Soroaka and Balang. In part, these lie in rough terrain, unfavorable for development. Reserves of a million tons or more, with somewhat less than 2 percent average nickel content, are known, but reserves of 3-percent grade do not exceed 300,000 tons.

In Venezuela large, low-grade nickel silicate deposits have been explored at Loma de Hierro. A portion of the ground underlain by a body of serpentine 10 to 12 miles long and more than half a mile wide has been sampled by the International Nickel Co. of Canada, Ltd., and a number of separate ore bodies have been found. Data on details of ore reserves have not been published, but press notes have indicated that reserves exceed 30 million tons of about 1¾ percent nickel. The topographic situation of the ore-bearing blankets is very favorable for mechanized mining from open pits, from which the ore could be transferred to river site by aerial tramway.

The nickeliferous iron ores are the world's largest potential source of nickel. Immense deposits, measurable in hundreds of millions of tons are known in Cuba, Philippines, Celebes, and parts of Borneo. Smaller and lower grade deposits are found in Japan, Madagascar, Greece, Puerto Rico, and elsewhere. However, the average low content of nickel in the deposits, commonly a little above or below 1 percent, combined with the finely divided distribution of the nickel in an iron-rich material, has made it relatively costly to obtain a product of high nickel content.

The deposits of nickeliferous iron ores owned by the Freeport Sulphur Co. in the vicinity of Levisa Bay, Cuba, are the only ores from which a high-grade nickel product (nickel oxide) has been produced on a commercial scale. The Nicaro plant is owned by the Cuban Nickel Co., a United States Government corporation. Extensive sampling and experimental work begun in 1940 outlined deposits that contained about 31 million tons of lateritic ore having a nickel content of 1.45 percent.

The nickeliferous iron deposits (containing 1 percent or more nickel) of Cuba are estimated to total about 356 million short tons of ore containing about 4,646,000 tons of nickel. Reserves of ore containing less than 1 percent nickel are estimated to total 1,653,000,000 tons containing about 13,000,000 tons of nickel.

Despite the fact that there are enormous potential reserves of nickel throughout the world, economic ore deposits are exceedingly rare. A notable exception is the Sudbury district of Canada, where the deposits are rich in nickel and copper and also carry important values in platinum metals, gold, silver, cobalt, selenium, and tellurium. Elsewhere, many of the reserves are less rich in nickel, and most are lacking in important values in byproduct metals; consequently, operation of many of the deposits would not be economic, and for others processes for treating the ores have not been developed.

Figure 1 shows the distribution of the known nickel resources of the world as of January 1950.

TECHNOLOGY

The ores worked primarily as sources of nickel fall into three classes: (1) Sulfides, (2) silicates, and (3) nickeliferous iron ores. Pentlandite, which contains varying percentages of nickel and iron [(NiFe)S] is believed to be the source of nearly all the nickel produced from the pyrrhotite-chalcopyrite ores of the Sudbury area, Ontario.

As previously stated, in addition to nickel and copper, the ores of the Sudbury district contain gold, silver, platinum-group metals, cobalt, and other metals, which are recovered as byproducts. The silicate ore of New Caledonia and the lateritic ore of Cuba contain small quantities of cobalt, which is not recovered as a separate product. The Cuban

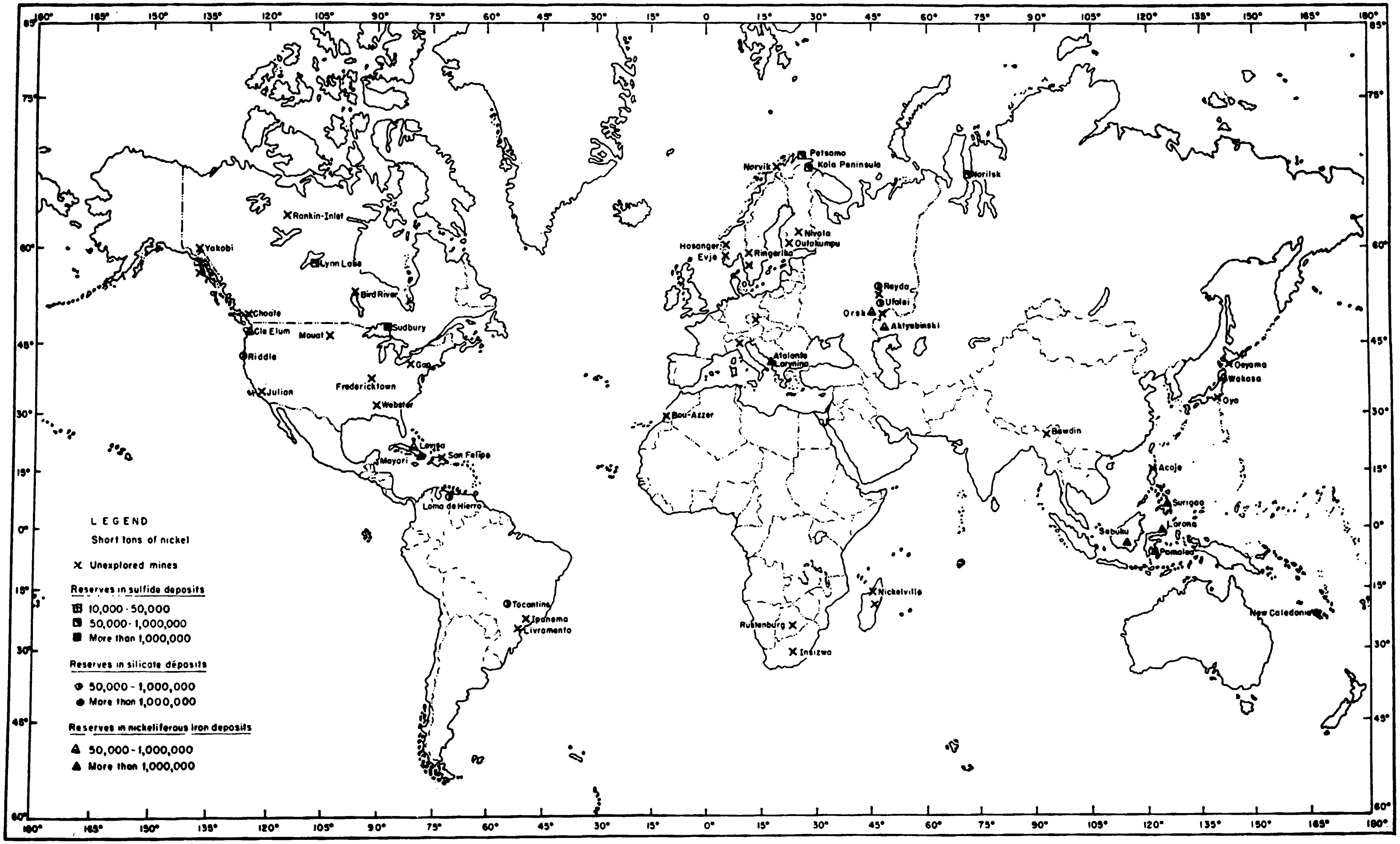

FIGURE 1.—Distribution of Known Nickel Resources of the World as of January 1950.

ore also contains iron and chrome, which are not recovered.

Nickel ore is mined by both open-pit and underground methods. In the Sudbury district, Ontario, open-pit quarrying is done with electrically operated shovels, and haulage from pits is by trucks. However, as a result of the exhaustion of ore that can be mined by open-pit methods, the International Nickel Co. of Canada, Ltd., is converting to mining exclusively from underground. Various stoping methods are used. The mining methods of International Nickel Co. of Canada, Ltd., have been described in much detail in the Canadian Mining Journal (*3*).[2] In Cuba and New Caledonia mining is also by open-pit methods.

The milling, smelting, and refining of nickel are complex and intricate operations. The operations of the International Nickel Co. of Canada, Ltd., have been described in much detail (*2*, *3*). The ammonia-leaching process used by Nicaro Nickel Co. to produce nickel oxide from Cuban lateritic ore has also been described (*1*, *4*).

GRADES OF NICKEL

The markets for primary nickel are served largely by electrolytic cathodes, which are cut into convenient sizes and contain 99.5 percent nickel plus cobalt. Commercially pure nickel is also supplied in the form of shot, ingot, pellets, and powder. The second largest quantity of nickel marketed is in the form of oxide and oxide sinter, which contain 75 to 90 percent nickel. Other special forms of nickel comprise "F" shot or ingot containing about 92 percent nickel; nickel-chromium-iron ingot containing about 45 percent nickel; nickel-copper-chromium ingot containing about 56 percent nickel; copper-nickel shot or ingot containing 50 percent nickel; cupronickel shot or ingot containing about 30 percent nickel; and Nisiloy, a nickel-silicon alloy, containing about 58 percent nickel. A relatively small quantity of nickel is sold in the form of salts containing 15 to 50 percent nickel. A substantial quantity of nickel is marketed in the form of Monel metal, which contains about two-thirds nickel and one-third copper, as well as in the form of Inconel, rolled nickel, and nickel anodes.

USES

The uses of nickel are numerous. Alloyed with iron and many nonferrous metals or employed as metallic nickel, it meets the peculiar physical, chemical, and esthetic qualities required in thousands of uses. Its ability to impart great strength and corrosion resistance to alloys is its outstanding characteristic. Other uses depend on the magnetic qualities found in alloys containing certain percentages of nickel. Indicative of the versatility of nickel is the fact that over 3,000 alloys in current use contain a fraction to something less than 100 percent nickel.

SUBSTITUTES AND ALTERNATES

In view of the urgent need for conserving nickel, cobalt, tungsten, manganese, and molybdenum because of acute world shortages during the Korean conflict, a Joint Subcommitee on Utilization was formed by the International Materials Conference to study the possibilities of saving these elements. Concerning nickel, the Subcommittee reported as follows (*11*):

> Many opportunities for adding to the availability of nickel for essential purposes by tightening up on scrap salvage and by substitution have been noted by the Subcommittee. Nickel could be saved in relatively large amounts by the increased use of economy steels, such as triple alloy (nickel-chromium-molybdenum), boron-containing steels, or other substitute steels. The greatest proportionate savings could probably be made in case-hardening steels where relatively high nickel content continues to be used in some countries. There are also important opportunities for the down-grading of high nickel alloy compositions, where lower grade non-ferrous alloys and stainless steels may often be substituted. In the case of nickel-containing electrical resistance alloys, downgrading would require technical consideration of the individual cases. In this field, more extended use might also be made of the iron-base, iron-chromium-aluminum alloys, once the technical problems of manufacture and use are fully mastered. Much of the nickel-containing stainless and heat-resisting steel production could similarly be downgraded, even going as far, in many instances, as substitution of nickel-free ferritic for nickel-containing austenitic steels. Such substitutions, however, would not of course be achieved without some sacrifice of properties. The 70 : 30 cupronickel composition, which is long established as a material for condenser tubes, is giving way in Canada and the United States to a copper alloy with 10 percent nickel and 1 percent iron (except where brazing is employed) and in Europe to aluminum-brass. As should be clear from what was said earlier, nickel may be saved in nonvital uses of both soft and hard magnetic materials. Nickel could also be saved (but to a limited extent) by reducing its employment in aluminum-base piston alloys. The recent development of spheroidal-graphite cast-iron (known in the United States as "ductile iron") foreshadows savings in nickel, since it can often be used alternatively with alloy cast irons and steels.
>
> Finally there are numerous applications of nickel and its alloys where the property requirements, especially corrosion-resistance, concern only the surface of the product. Here, there is room for extension of the use of nickel and nickel alloy-clad products. Specifically, heavy-gauge nickel and stainless clad steel could find wider use in the chemical industry and finer gauges be used in electronic applications, while nickel-coated steel is being used to replace pure nickel in coinage. In Appendix I of this report, reference is made to the wide applicability of a new type of electroplated finish in which a deposit of a nickel-tin compound is laid down. Composite materials have much to recommend them where metal economy is concerned. The Subcommittee,

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

TABLE 4.—*Trend in world mine production (exclusive of U. S. S. R.) of nickel for various years 1926–53*

[Short tons of nickel content]

Year	Canada	Cuba	New Caledonia	United States	Union of South Africa	Other countries	Total
1926	32, 850		4, 189	323			37, 400
1931	32, 833		4, 256	373		2, 300	39, 800
1936	84, 870		10, 141	107		5, 700	100, 800
1941	141, 129		11, 459	660	640	9, 700	163, 600
1943	144, 009	2, 679	8, 128	642	378	16, 000	172, 000
1946	96, 062	12, 391	3, 200	352	548	1, 100	113, 700
1947	118, 627	2, 219	3, 100	646	583	595	125, 800
1948	131, 740		2, 700	883	505		135, 800
1949	128, 690		3, 800	790	625	8	133, 900
1950	123, 659		4, 685	913	929		130, 200
1951	137, 903		7, 400	756	1, 254	94	147, 400
1952	140, 558	8, 924	11, 600	633	1, 444	680	163, 800
1953	143, 965	13, 844	18, 700	602	1, 890	605	179, 600

however, is of the opinion that any proposals for extension of the use in these ways of nickel and the other elements under consideration should be given very careful scrutiny from the point of view of the alloy value, if any, of the material as scrap when its serviceability comes to an end, before being accepted as economy measures.

The chromium-manganese-nickel stainless steels, with nickel up to 5 percent, appear to offer opportunities for nickel conservation as a substitute for the chromium-nickel stainless steels. Some grades have already been produced in quantity and others are currently under development.

A new permanent-magnet alloy—bismanol—has been developed by the Naval Research Laboratory. In the magnet, bismuth, manganese, and powdered iron are substituted for cobalt and nickel.

A new class of ferrite permanent-magnet material has been developed. The material is being manufactured in Europe under the name of Ferroxdure and is to be produced in the United States under the names of Magnadur and Indox. This material is an oxidic ceramic of the approximate composition $BaFe_{12}O_{19}$.

The application of ceramic coatings to J47 jet-engine parts through a method that allows substitution of Type 321 stainless steel containing 8 percent nickel, instead of an alloy with a 76-percent nickel content, was expected to result in substantial saving of nickel by the Solar Aircraft Co.

PRODUCTION

World (exclusive of U. S. S. R.) production of nickel has grown from about 10,000 short tons in 1900 to 180,000 tons in 1953. Despite the fact that nickel has been produced in several countries, since 1905 Canada has supplied the bulk of the total.

The trend in production of nickel in the world since 1926 is shown in table 4.

SECONDARY RECOVERY

As shown in table 5, a small quantity of nickel is recovered in the United States from nonferrous scrap.

TABLE 5.—*Secondary nickel recovered in the United States, 1940–53*

Year	Short tons	Year	Short tons
1940	4, 152	1947	9, 541
1941	5, 315	1948	8, 850
1942	4, 142	1949	5, 680
1943	6, 917	1950	8, 795
1944	4, 321	1951	8, 602
1945	6, 483	1952	7, 479
1946	8, 248	1953	8, 352

FOREIGN TRADE[3]

Being the largest consumer of nickel in the world and dependent on foreign countries for almost all of its supplies, the United States is the largest importer. Table 6 shows the spectacular growth in imports since 1910.

TABLE 6.—*Nickel imported into the United States, 1910–53*

[Thousand pounds]

Year:	*Nickel content (estimated)*
1910–14 (average)	38, 213
1915–19 (average)	62, 932
1920–24 (average)	29, 144
1925–29 (average)	52, 239
1930–34 (average)	37, 080
1935–39 (average)	85, 634
1940–44 (average)	218, 000
1945	214, 866
1946	185, 000
1947	161, 435
1948	192, 800
1949	182, 146
1950	182, 694
1951	186, 380
1952	217, 700
1953	237, 474

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[3] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

TABLE 7.—*Nickel products exported from the United States, 1948–53, by classes*

[Pounds]

Class	1948	1949	1950	1951	1952	1953
Ore, concentrates and matte	1, 500	52, 000	12, 826	35, 578	------------	------------
Alloys and scrap containing nickel (including Monel metal)	11, 652, 796	5, 568, 949	5, 675, 191	7, 984, 503	11, 648, 169	22, 846, 005
Metal in ingots, bars, sheets, etc	2, 705, 777	1, 610, 329	676, 169	386, 310	1, 966, 621	6, 578, 740
Nickel-chrome electric resistance wire	747, 082	686, 270	428, 885	393, 599	267, 473	356, 222
Nickel-silver crude, scrap, or bars, rods, etc	1, 260, 330	1, 024, 613	496, 598	443, 175	(1)	(1)

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[1] Beginning Jan. 1, 1952, not separately classified.

More than 90 percent of the nickel imported into the United States comes from Canada. Small quantities are received from Norway, United Kingdom, and France. The nickel imported from Norway, and most of that from the United Kingdom is produced from ore mined in Canada. During World War II New Caledonia supplied 79,877,000 pounds (gross weight) of nickel matte, which was refined at Huntington, W. Va. Cuba supplied 81,862,000 pounds (gross weight) of nickel oxide during 1944–47 and 56,149,000 pounds in 1952–53.

Since January 1, 1948, the rate of duty on refined nickel imported into the United States has been 1¼ cents a pound. Nickel ore, matte, and oxide enter the United States duty free.

Exports of nickel from the United States comprise largely products manufactured from imported raw materials.

CONSUMPTION

During and preceding World War I the principal use of nickel was in nickel steel, which was employed almost exclusively as armorplate and in various types of military and naval ordnance. From 1900 to 1912 world consumption ranged from about 10,000 short tons to 30,000. During 1916, 1917, and 1918 consumption averaged about 50,000 tons annually. With cessation of the manufacture of war material following the end of hostilities, the Washington Disarmament Agreement of 1921 (which limited the navies of the world), and the world depression, the demand for nickel largely evaporated. As a result, world consumption of nickel probably did not exceed 10,000 tons in 1921. The First World War, however, disclosed the possibilities of many new applications for nickel. In consequence, the producers entered upon a vigorous campaign with the object of developing a greater and more diversified demand for nickel. Technical staffs were created, laboratories for research were established, and a constant field service was given to uses of nickel and nickel-copper products. It was soon recognized that proper production facilities were needed to support the development of markets for Monel, a silvery white, high-strength, corrosion-resistant alloy that had been developed in 1905. Accordingly, in 1922 plant facilities for refining raw materials and their conversion to finished forms were completed at Huntington, W. Va. The results of these efforts are evidenced by the fact that world consumption of nickel had reached about 35,000 tons in 1925, after which it gained momentum until the depression that began in 1931.

Research to find new uses for nickel has been and still is pursued with great energy and pronounced success. Ferrous alloys include structural nickel steels, stainless steel, nickel cast iron, nickel wrought iron, and many other alloys indispensable to modern industry. Nickel plating has expanded, as well as the use of pure nickel and nickel alloys for coinage. Nickel-copper alloys, including Monel metal, have found wide application. During World War II world consumption of nickel reached an alltime high of about 184,000 short tons in 1943. Although world consumption of nickel declined substantially following the end of hostilities in 1945, it has remained at a high level.

The imposition of severe restrictions on the uses of nickel during and after the Korean conflict because of the shortage of nickel, plus the emphasis in many public announcements on the scarcity of nickel, has resulted in many consumers designing away from it. Accordingly, the nickel industry has been retarded in developing markets in the face of the surplus of nickel that will occur when the stockpile objective is completed.

Consumption of nickel in the United States from 1949 to 1953 is shown in table 8.

The combined consumption of nickel in the free-world countries in 1950 was 54 percent less than in the United States. The largest foreign user of nickel is the United Kingdom, which consumed about 18,000 tons in 1950 compared with 100,000 tons in the United States.

TABLE 8.—*Nickel (exclusive of scrap) consumed in the United States, 1949–53, by uses*

[Pounds of nickel]

Use	1949	1950	1951	1952	1953
Ferrous:					
Stainless steels	23, 817, 187	41, 822, 486	43, 584, 274	54, 685, 711	44, 548, 548
Other steels	26, 948, 418	35, 554, 167	32, 850, 461	35, 956, 787	37, 918, 877
Cast iron	6, 792, 472	9, 761, 622	7, 430, 972	7, 276, 976	8, 428, 079
Nonferrous [1]	39, 643, 277	58, 818, 365	53, 903, 848	64, 035, 398	61, 614, 181
High-temperature and electrical-resistance alloys	8, 107, 918	11, 407, 174	14, 815, 616	16, 040, 189	16, 442, 108
Electroplating:					
Anodes	25, 920, 038	32, 357, 201	10, 820, 921	12, 278, 313	26, 547, 805
Solutions [2]	2, 399, 369	2, 453, 419	909, 883	968, 885	1, 943, 572
Catalysts	2, 367, 000	2, 400, 000	2, 768, 905	2, 920, 062	2, 870, 455
Ceramics	536, 940	786, 838	498, 091	398, 067	501, 427
Magnets	2, 734, 533	1, 946, 971	1, 291, 856	1, 190, 624	1, 595, 439
Other		2, 669, 734	4, 491, 483	7, 043, 737	8, 950, 972
Total	139, 267, 152	199, 977, 977	173, 366, 310	202, 794, 749	211, 361, 463

[1] Comprises copper-nickel alloys, nickel-silver, brass, bronze, beryllium alloys, magnesium and aluminum alloys, Monel, Inconel, and malleable nickel.

[2] Figures estimated to represent about 70 percent of total.

PRICES

From 1929 to November 25, 1946, the base spot price for nickel in the United States was 35 cents a pound, but large consumers were supplied on contract until January 1939 at 32 cents and thereafter at 31½ cents a pound. On November 25, 1946, the price for 99.9-percent electrolytic nickel was advanced to 35 cents; that price included the United States duty of 2½ cents a pound. The price of electrolytic nickel since 1947 is shown in table 9.

TABLE 9.—*Price quoted by International Nickel Co., Inc., for electrolytic nickel f. o. b. Port Colborne, Ontario, 1947–54*

[Carlots]

Year:	*Cents per pound* [1]
1947	35
1948:	
Jan. 1—July 21	33¾
July 22—Dec. 31	40
1949	40
1950:	
Jan. 1—May 31	40
June 1—Dec. 12	48
Dec. 13—Dec. 31	50½
1951:	
Jan. 1—May 31	50½
June 1—Dec. 31	56½
1952	56½
1953:	
Jan. 1—Jan. 13	56½
Jan. 14—Dec. 31	60
1954:	
Jan. 1—Nov. 23	60
Nov. 24—Dec. 31	64½

[1] Prices for 1942–47 include duty of 2½ cents a pound and for 1948–54, 1¼ cents.

DEFENSE PROGRAM

The status of nickel, insofar as supply and demand are concerned, following the outbreak of hostilities in Korea, coincided somewhat with the pattern of World War II. Nickel was least abundant of the major ferrous alloying elements during World War II and one of the first metals placed on a restricted-use basis. As a result of hostilities in Korea, which led to an upward surge in nickel requirements, an acute shortage developed. Accordingly, both the International Nickel Co. of Canada, Ltd., and Falconbridge Nickel Mines, Ltd., established voluntary programs for equitable distribution of nickel for civilian use during the latter part of 1950. However, effective January 1, 1951, consumption and procurement of nickel were subject to NPA Order M–14, and effective August 15 nickel was placed under complete allocation under Order M–80. The use of nickel in a large number of items was prohibited by NPA. Order M–80 was revoked, effective November 1, 1953.

To achieve an orderly and equitable distribution of available supplies, allocation of nickel metal, oxide, and salts to the free-world countries in the fourth quarter of 1951 was announced on October 11 by the International Materials Conference. Nickel salts were excluded after 1951. Allocation was discontinued after the third quarter of 1953.

To increase the supply of nickel, the nickel facilities at Nicaro, Cuba, were rehabilitated under a contract with the United States Government; an agreement was entered into between Defense Minerals Administration and National Lead Co. to build a nickel-cobalt-copper separation plant at Fredericktown, Mo., and with Hanna Nickel Smelting Co. to build a smelter to treat the ore from Riddle, Oreg. Government assistance was given to several foreign producers.

RESEARCH

The nickel industry maintains modern research and development laboratories for developing new uses for nickel and nickel alloys and for developing new processes for smelting and refining nickel. These research and development programs have contributed largely to the growth of the nickel industry.

The Chemical Construction Corp. has announced a new process for extracting nickel, cobalt, copper, and other important metals from ores by chemical rather than the usual smelting and refining methods. The process consists of pressure leaching of unroasted sulfide ore concentrates, an almost simultaneous oxidation step, and then direct reduction of copper, nickel, and cobalt as pure metallic powders. Leaching can be done with either ammonia or acid, depending on the ore concentrate being treated. The process will be employed by Sherritt Gordon Mines, Ltd., which will use ammonia leaching, and National Lead Co., which will leach with sulfuric acid. The process has been described in some detail (*14, 15*).

The Hanna Development Co., at its laboratory at Hibbing, Minn., developed two alternative chemical processes for recovering the nickel in the silicate ore from Riddle, Oreg. Subsequently, the Bureau of Mines, in cooperation with the company, conducted pyrometallurgical tests at Albany, Oreg., to develop an alternate low-cost electrosmelting process to yield ferronickel. Three grades of ferronickel, averaging 10, 25, and 45 percent nickel, were produced from 140 tons of dried ore in 2 lots averaging 1.7 and 1.5 percent nickel.

SIGNIFICANCE OF CUBAN LATERITES

The Cuban laterite offered the best opportunity to augment the world supply of nickel during World War II, and in 1942 the United States Government began building a plant at Nicaro, Oriente, for extracting the nickel. Production was begun in late 1943. The investment in this plant is over 40 million dollars. The plant was closed March 31, 1947, and rehabilitated in early 1952. It has a capacity of about 30 million pounds of nickel annually from 1.3 million tons of ore.

Ore for the Nicaro plant has been mined from deposits in the Levisa Bay area, which are owned by Freeport Sulphur Co. Between the laterite and the underlying serpentine country rock is a layer of decomposed serpentine containing about 1½ times as much nickel and cobalt as the laterite. Both the laterite and decomposed serpentine are mined and treated at the Nicaro plant, operation of which is managed on a fee basis by the Nickel Processing Corp., a private concern. The plant and townsite are owned by the Cuban Nickel Company, a United States Government corporation.

In February 1952 the Defense Materials Procurement Agency requested the Bureau of Mines to conduct an investigation of the laterite ore reserves to ascertain if enough ore was available in claims acquired and denounced by the Cuban Nickel Company to double the present capacity of the Nicaro plant. Accordingly, the Bureau began drilling on the claims, which are in the Levisa area, on September 15, 1952; by the end of 1953 nearly 35 million tons of ore with an average grade of about 1.39 percent nickel and 0.10 percent cobalt had been indicated. In January 1954 General Services Administration announced that $43 million had been allotted to enlarge the Nicaro plant 75 percent and that $1 million had been earmarked as a research fund.

Thus far the Nicaro plant has been limited to the recovery of nickel and some cobalt, but not as separate products. Consequently, virtually all of the iron and chromium, most of the cobalt, and 20 to 25 percent of the nickel have not been recovered. The iron cannot be utilized because of its contained nickel, chromium, and cobalt. The gross value of all salable minerals in the ore, provided they can be recovered economically, is relatively high. Deposits with equal value in the base metals or gold and silver in the United States would be considered bonanzas. Profitable exploitation of the Cuban ores will depend upon the economic recovery of most of or all the valuable constituents. Along this line, samples of the laterite and serpentine, as well as tailings from the Nicaro plant, have been shipped to several stations of the Bureau of Mines for metallurgical research.

The laterite deposits in the Mayari district contain the largest known nickel reserve in the world. The deposits are about 20 miles from the coast. The ore contains, on a dry basis, slightly less than 1 percent nickel, about 49 percent iron, 3 percent chrome, and a little cobalt. The Bethlehem Steel Co. has made persistent effort to utilize the deposits as a source of iron ore. However, as the nickel and chromium remain with the iron, the sale and use of iron and steel made from the ore have been strictly limited. The company has developed a nitric acid process for recovering the valuable constituents in the ore.

The deposits in the Moa Bay area are next in importance to the Mayari deposits. The Moa Bay deposits are about 65 miles east of the Levisa Bay deposits and only a few miles from tidewater. The Moa Bay ore contains more nickel and less iron than the Mayari ore. Discovery of a large deposit of nickel ore as a result of extensive exploration recently completed in the Moa Bay area was announced by Freeport Sulphur Co. The deposit is reported to contain at least 40 million tons of ore averaging about 1.35 percent nickel and 0.14 percent cobalt. The company proposes to use a sulfuric acid leaching process to recover the nickel and cobalt.

Competition of Cuban nickel with Canadian is handicapped by the fact that Canadian ore contains many commercial byproducts that the Cuban ore does not have, such as copper, platinum metals, gold, silver, selenium, and tellurium, which are simultaneously recovered with the nickel. Although the Cuban ore also contains cobalt, iron, and chromium, thus far these minerals have not been recovered because of lack of a suitable metallurgical process. Compared with Canadian ore, the laterite has an advantage so far as mining costs are concerned but a disadvantage in metallurgical costs with the leaching process now used for treating the ore.

At present it is reported that nickel is being produced at the Nicaro plant at a cost, including plant depreciation, below the current market. Whether or not Cuban nickel can compete in a peacetime economy would seem to depend on economic recovery of the associated minerals in the ore or on development of an improved and lower cost method for extracting nickel than is now used at the Nicaro plant.

OUTLOOK

World production of nickel for many years will, as in the past, continue to come chiefly from Canada, where expansion programs have been under way. In July 1951 the International Nickel Co. of Canada, Ltd., increased its annual rate of production of refined nickel 6,000 short tons, and beginning 1954 an annual output of 12,000 tons of refined nickel will be derived from nickel-bearing pyrrhotite. Much mine development and construction work are in progress by Falconbridge Nickel Mines, Ltd., which completed a third blast furnace at its smelter in October 1953. The smelter and refinery of Falconbridge are now geared to handle 20,000 tons of nickel annually. However, further substantial increases in capacity at both the smelter and refinery are planned. Production of small outputs of nickel were begun at the East Rim and Milnet mines in 1952 and at Nickel Offsets in 1953. First production of nickel metal by Sherritt Gordon Mines, Ltd., at its new chemical-metallurgical plant at Fort Saskatchewan, Alberta, was made in July 1954. Production of 8,500 tons annually is anticipated.

The nickel-cobalt-copper separation plant of National Lead Co. at Fredericktown, Mo., was completed during the first quarter of 1954. Beginning in 1955, production of 900 tons of nickel annually is planned. The smelter of Hanna Nickel Smelting Co., which will treat ore from the Riddle deposit in Oregon, began production January 4, 1955. Ultimate production probably will be at an annual rate of about 8,000 tons of nickel. The capacity of the Nicaro, Cuba, plant is being increased 75 percent.

As a result of these expansions, it is estimated that countries outside the U. S. S. R. will have capacity to produce about 215,000 tons of nickel annually; the 1953 output was 179,600 tons. This increased production will assure the availability of needed nickel in an emergency.

Following the expansion program the capacity to produce nickel will exceed peacetime needs. However, many new uses and applications for nickel being developed for the defense program will, as was the case following World Wars I and II, be converted to peacetime use. Moreover, the nickel industry's policies of aggressive research and development of new markets, which brought about the great expansion of peacetime uses following World Wars I and II, will further enlarge the sphere of nickel for commercial use. However, because of severe restrictions on uses of nickel and emphasis in public announcements on its scarcity, development of additional markets has been retarded. It is believed that the pre-Korean peacetime uptrend in consumption of nickel will project itself into the postemergency period and that within 10 to 15 years markets will be found for most of the expanded output of the current defense program.

Largely as the result of the scarcity of nickel during and after the Korean conflict, there has been a tendency to develop substitutes for nickel. However, many of the substitutes have not proved as satisfactory as nickel; consequently, with assurance of adequate supplies in the foreseeable future, many customers will return to it. Likewise, a plentiful supply will permit the electroplating industry to use the right amount of nickel for coatings thick enough to provide satisfactory performance of plated items, and it will also make it unnecessary for other users to conserve nickel by downgrading.

PROBLEMS

The foremost problem confronting the nickel industry is the great disparity between peace and war requirements. An industry geared to peace requirements is entirely inadequate to meet war needs. On the other hand, an industry expanded to supply war demand finds itself with an enormous burden of excess capacity when peace is reestablished.

Thus far metallurgists have been unsuccessful in developing an economic and feasible process for recovering all the valuable minerals in the Cuban nickel-bearing ores or in developing a low-cost process that permits profitable recovery of nickel alone. At present, recovery at the Nicaro plant is limited to nickel. Consequently, the tailings from the plant are rich in iron, but the presence of unrecovered nickel, cobalt, and chromium prevents use of the tailings as an iron ore. A problem is to recover all the valuable minerals, thus permitting wide flexibility in production to meet changing markets. Moreover, the present method of producing nickel carbonate, which precedes by one step the outturn of nickel oxide, adds appreciably to the cost. Production of metal direct from carbonate would result in lower operating costs.

The nickeliferous iron ores (oxides) and the nickel silicate ores are reported to furnish the largest reserve of nickel, but at present the capital investment in facilities and recovery costs is too high for profitable commercial production. Methods for reducing production costs at these deposits would expand greatly the commercial reserves of these ores.

An unknown but significant quantity of nickel contained in scrap is being wasted because of the high cost of recovery.

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NITROGEN COMPOUNDS

By

E. Robert Ruhlman[1]

NEARLY 400,000,000 tons of nitrogen annually is "fixed" by lightning during electrical storms. From this large supply, rain and snow deposit about 6 pounds of nitrogen per acre annually. The total quantity of fixed nitrogen thus precipitated per acre per year, however, is very small compared to that removed by crops. Modern science, using the same raw material (air), has created world independence from natural mineral sources.

Summary

Nitrogen in the atmosphere and in coal deposits assures ample reserves until eternity. The supply of nitrogen compounds depends primarily on adequate processing-plant capacity.

The main uses of nitrogen compounds are in fertilizers and explosives. Increasing quantities are being used in plastics and synthetic fibers.

The big problem of the nitrogen industry is to maintain a healthy economic position to supply the domestic needs of agriculture and the chemical industry and yet be able to meet the expanded requirements in periods of emergency.

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[1] Commodity-industry analyst. Bureau of Mines.

BACKGROUND

Since the early 1900's, when nitrogen fixation was first developed, the production of nitrogen has become predominantly a manufacturing industry. Before this the nitrate deposits of Chile were virtually the only commercial source of nitrogen in the world.

The three main sources of nitrogen today are (1) the atmosphere, (2) coal, and (3) natural nitrate deposits.

ATMOSPHERIC NITROGEN

Nitrogen gas has limited commercial value. It usually is combined or "fixed" with other elements. Many methods have been developed to "fix" nitrogen since the first commercial plant, using the arc process, was operated in Norway about 1900. By the arc process, the nitrogen and oxygen of the air were combined by an electric arc to form nitrogen oxides, which were absorbed in water to form nitric acid. The cyanamide process, developed about 1900, makes calcium cyanamide from coke, limestone, and nitrogen. The Haber-Bosch process was developed in Germany between 1905 and 1913 for producing synthetic ammonia by combining nitrogen and hydrogen in the presence of a catalyst. Research in every major country was stimulated by the success of the Haber-Bosch process, and a number of variations were developed to produce ammonia from nitrogen and hydrogen.

DOMESTIC INDUSTRY

The first successful domestic atmospheric nitrogen plant was built at Syracuse, N. Y., by the Atmospheric Nitrogen Corp. in 1921, using a modification of the Haber-Bosch process. This was followed by other plants which soon were contributing large quantities of nitrogen compounds to supply the growing demand. To insure adequate supplies of nitrogen compounds during World War II, the Government constructed 10 synthetic ammonia plants with a total rated capacity of 863,000 short tons of nitrogen equivalent per year. The plants operated during the latter part of the war, following which all except the Tennessee Valley Authority plant at Muscle Shoals, Ala., were offered for sale or lease to private industry.

The nitrogen industry in the United States was expanded rapidly between 1951 and 1954. Construction of additional facilities was in progress in 1954 as a result of an increase in the expansion goal to 3.5 million tons a year of nitrogen by the Office of Defense Mobilization. The present industry of about 30 plants throughout the United States, with plants under construction and proposed plants, is shown in table 1.

TABLE 1.—*United States nitrogen plants (excluding byproduct plants) in November 1954*

Company and location of plant	Operating	Under construction	Proposed
Allied Chemical & Dye Corp.:			
Hopewell, Va.	X		X
La Platte, Nebr.	X		X
South Point, Ohio	X		X
American Cyanamid Co.: Avondale, La.	X		
Ammonia Chemical Co.: Oleander, Calif.			X
Atlantic Refining Co.: Point Breeze, Pa.	X		
Atlas Powder Co.: Atlas, Mo.			X
Brea Chemicals, Inc.: Brea, Calif.	X		
Columbia River Chemical, Inc.: Pasco, Wash.			X
Columbia Southern Chemical Co.: Natrium, W. Va.		X	
Commercial Solvents Corp.: Sterlington, La.	X		
Cooperative Farm Chemicals Corp.: Lawrence, Kans.	X		
E. I. du Pont de Nemours & Co.:			
Belle, W. Va.	X		
Niagara Falls, N. Y.	X		
Deere & Co.: Pryor, Okla.	X		
Dow Chemical Co.:			
Freeport, Tex.	X	X	
Midland, Mich.	X		
Pittsburg, Calif.	X		
Grace Chemical Co.: Memphis, Tenn.		X	
Gulf Chemical Co.: Pensacola, Fla.			X
Hercules Powder Co.:			
Hercules, Calif.	X		
Louisiana, Mo.	X		
Hooker Electrochemical Co.: Tacoma, Wash.	X		
Ketona Chemical Co.: Birmingham, Ala.			X
Lion Oil Co.:			
El Dorado, Ark.	X		
Luling, La.	X		
Lange Bros.: St. Louis, Mo.			X
Mississippi Chemical Corp.: Yazoo City, Miss.	X	X	
Mississippi River Fuel Corp.: Crystal City, Mo.		X	
National Distillers Products Corp.: Tuscola, Ill.	X		
Northern Chemical Industries: Searspoint, Maine			X
Olin-Mathieson Chemical Corp.:			
Lake Charles, La.	X		
Morgantown, W. Va.	X		
Niagara Falls, N. Y.	X		
Pennsylvania Salt Manufacturing Co.: Wyandotte, Mich.	X		
Phillips Chemical Co.:			
Etter, Tex.	X		
Pasadena, Tex.	X		
Salt Lake City Chemical Co.: Salt Lake City, Utah			X
San Jacinto Chemical Corp.: Houston, Tex.	X		
Shell Chemical Co.:			
Pittsburg, Calif.	X		
Ventura, Calif.	X		
Spencer Chemical Co.:			
Henderson, Ky.	X		
Pittsburg, Kans.	X		
Vicksburg, Miss.	X		
Standard Oil Co. (Ohio): Lima, Ohio			X
Standard Oil Co. (Indiana): Hammond, Ind.			X
Sun Oil Co.: Marcus Hook, Pa.			X
Tennessee Valley Authority: Wilson Dam, Ala.	X		
United Chemical Co.: Timpson, Tex.			X
U. S. Steel Corp.:			
Geneva, Utah		X	
Gary, Ind.			X
Utah Chemical Co.: Mount Pleasant, Utah			X
Westvaco Chlor-Alkali Division, Food Machinery-Chemical Corp.: Charleston, W. Va.		X	
Total annual rated capacity—tons of equivalent nitrogen	2, 680, 000	265, 000	791, 000

BYPRODUCT NITROGEN

The recovery of nitrogen as a byproduct of the coke industry was begun about 1900 and rapidly became a large contributor to the world nitrogen supply. The increased coke requirements of an expanding iron and steel industry made available large quantities of byproduct nitrogen.

In the distillation of coal, the nitrogen is distributed among the coke, tar, and gaseous components and recovered only from the gases. In the gases, the nitrogen is present as ammonia (NH_3) and recovered either as aqua ammonia (NH_4OH) or ammonium sulfate [$(NH_4)_2SO_4$], depending upon whether water or sulfuric acid is used as an absorbent.

Byproduct coke ovens have largely replaced beehive and other types of ovens that do not recover the byproducts of coke. Nitrogen compounds are among the most profitable byproducts in coke plants and are recovered by about 95 percent of all coke plants in the United States.

DOMESTIC PRODUCTION

The production of anhydrous ammonia and other nitrogen compounds is shown in table 2.

IMPORTS AND EXPORTS[2]

Imports into and exports from the United States of major nitrogen compounds are listed in table 3.

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[2] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, records of the U. S. Department of Commerce.

TABLE 2.—*Principal nitrogen compounds produced in the United States, 1943–53*

[Short tons]

Commodity	1943	1944	1945	1946	1947	1948	1949	1950	1951	1952	1953
Ammonia (NH_3): Synthetic plants: Anhydrous ammonia	543,804	543,651	548,655	725,537	1,114,000	1,089,786	1,294,057	1,565,569	1,777,074	2,052,114	2,287,785
Byproduct coking plants (NH_3 content):											
Aqua ammonia	34,106	31,665	27,607	24,991	25,718	24,753	22,750	23,387	24,878	22,060	24,846
Ammonium sulfate	190,317	204,561	191,073	160,938	202,360	207,671	189,202	207,754	224,566	200,603	236,533
Subtotal	224,423	236,226	218,680	185,929	228,078	232,424	211,952	231,141	249,444	222,663	261,379
Grand total	768,227	779,877	767,335	911,466	1,342,078	1,322,210	1,506,009	1,796,710	2,026,518	2,274,777	2,549,164
Principal ammonium compounds:											
Ammonium sulfate:											
Synthetic plants	65,000	88,000	88,863	156,653	195,848	264,476	846,195	1,137,721	622,084	812,795	576,232
Byproduct coking plants	761,270	818,244	764,293	643,752	809,440	830,683	756,807	831,016	898,263	802,412	946,133
Total	826,270	906,244	853,156	800,405	1,005,288	1,095,159	1,603,002	1,968,737	1,520,347	1,615,207	1,522,365
Ammonium nitrate, basis solution, 100 percent NH_4NO_3	(1)	(1)	421,487	724,899	1,086,869	988,342	1,018,706	1,213,911	1,346,443	1,467,341	1,558,457

Source: Bureau of Mines Minerals Yearbooks.

[1] Not available.

TABLE 3.—*Major nitrogen compounds imported for consumption into and exported from the United States, 1948–53*

[Short tons]

(U. S. Department of Commerce)

	1948	1949	1950	1951	1952	1953
Imports:						
Industrial chemicals:						
Ammonium nitrate	209	1	----------	4	----------	----------
Fertilizer materials:						
Ammonium nitrate mixtures:						
Containing less than 20 percent nitrogen	250	2, 290	1, 523	361	624	8, 294
Containing 20 percent or more nitrogen	100, 314	136, 405	221, 299	342, 757	467, 166	755, 087
Ammonium phosphates	108, 228	126, 274	107, 695	134, 962	133, 288	166, 497
Ammonium sulfate	105, 887	105, 498	144, 732	216, 106	238, 063	523, 858
Calcium cyanamide	116, 504	115, 885	97, 725	68, 231	96, 195	82, 218
Calcium nitrate	(1)	38, 611	44, 331	55, 743	39, 466	67, 794
Nitrogenous materials, n. s. p. f	5, 304	4, 829	23, 830	26, 023	22, 067	17, 104
Potassium nitrate, crude	(1)	1	20	3, 367	12, 738	15, 941
Potassium-sodium nitrate	----------	6, 802	20, 409	8, 655	16, 460	12, 516
Sodium nitrate	709, 573	675, 543	618, 018	737, 324	675, 329	568, 873
Exports:						
Industrial chemicals:						
Anhydrous ammonia	3, 407	3, 477	10, 202	5, 907	15, 431	15, 119
Ammonium nitrate	5, 087	17, 004	3, 336	5, 049	5, 709	6, 013
Fertilizer materials:						
Ammonium nitrate	(2)	470, 443	94, 169	1, 255	3, 833	2, 172
Ammonium sulfate	136, 648	660, 733	819, 285	134, 100	121, 587	39, 440
Nitrogenous chemical materials, n. e. s.	701, 450	23, 510	41, 363	63, 768	48, 109	46, 485
Sodium nitrate	17, 100	3, 714	32, 862	43, 669	9, 441	24, 209

[1] Included in nitrogenous materials, not specially provided for.
[2] Included in nitrogenous chemical materials, not elsewhere specified.

PRICES

The prices of the major nitrogen compounds are shown in table 4.

CHILEAN NITRATE DEPOSITS

Production of nitrates from the Atacama Desert of South America was begun in the 18th century, but it was not until after the wars between Chile and Peru and Chile and Bolivia (1879–82), when Chile acquired control of all the nitrate lands, that appreciable activity began. Chile, lacking funds necessary to develop the nitrate industry, encouraged exploitation by foreigners, and large sums of English and German capital were invested in this industry. By the latter part of the 1880's, the Chilean nitrate industry obtained a world monopoly on the production of natural nitrates. The industry suffered drastic curtailment during World War I and never recovered the markets lost to the synthetic and byproduct nitrogen industries.

The nitrate-bearing ores of Chile range in composition from 5 to 30 percent sodium nitrate. They occur as slightly dipping deposits from a few inches to several feet thick overlain by an overburden of loose sand and gravel or consolidated conglomerate.

TABLE 4.—*Prices of major nitrogen compounds, Jan. 1, 1954, per short ton* [1]

Commodity:	*Price*
Chilean nitrate, port, warehouse, bulk	$53. 00
Sodium nitrate, synthetic, domestic, c. l. works, crude, bulk	47. 00
Ammonium sulfate, coke ovens, bulk	43. 50
Cyanamide, fertilizer-mixing grade, 20.6% N, granular, Niagara Falls, Ont., bagged	55. 00
Ammonium nitrate, fertilizer grade:	
Canadian, eastern, 33.5% N, c. l., shipping point, bags	77. 50
Western, domestic, works, bags	68. 00–70. 00
Anhydrous ammonia, fertilizer, tanks, works	85. 00–88. 00
Ammonium-nitrate-dolomite compound, 20.5% N, Hopewell, Va., bags	51. 00

[1] Quotations from Oil, Paint and Drug Reporter, Dec. 28, 1953.

The ore is mined by both hand and mechanical mining. Open-pit mining usually is employed, but some underground mining is done when the overburden exceeds 30 feet in thickness.

There are two main methods of extracting the nitrate, the Shanks process and the Guggenheim process. The Shanks process is not used for treating ore containing less than 15 percent sodium nitrate. Only about half of the nitrate is recovered. Development of the

Guggenheim process in 1925 permitted the mining of ore containing as little as 8 percent sodium nitrate, with nearly 100-percent recovery.

STATUS OF CHILEAN NITRATE INDUSTRY

With the development of synthetic processes, many chemical products became competitors of Chilean nitrate; however, natural nitrates contain minor quantities of magnesium, iodine, boron, calcium, potassium, lithium, and strontium, which contribute slightly to the minor element supply of the soil. Despite available cheaper synthetic nitrogen, sales of Chilean nitrate have been maintained at a relatively high level, although the industry has been hampered by strikes in recent years.

Anglo Lautaro Nitrate Co. and the Cia. Salitera de Tarapaca y Antofagasta produce about 90 percent of the total output. Both companies have modernized and expanded their facilities. Several plants, idle for many years, were again producing in 1954.

Production of nitrates totaled 1,853,655 short tons in 1951, 1,585,341 in 1952, and 1,439,057 in 1953, compared with more than 3½ million short tons in 1929, and less than 500,000 short tons in 1933.

Table 5 gives the distribution of exports of Chilean nitrate by major countries of destination, 1952–53.

TABLE 5.—*Exports of Chilean nitrate by major countries of destination, 1952–53*

[In percent]

Country of destination	1952	1953
Egypt	17	12
Spain	6	15
France	5	8
United States	48	44
South America and Central American countries	8	9
Other countries	16	12
Total	100	100
Short tons	1, 455, 679	1, 361, 170

WORLD PRODUCTION AND CONSUMPTION

The rapid expansion of the nitrogen industry has not been limited to the United States. The latest figures (1952–53) show that world production is equivalent to nearly 5 million tons of nitrogen. Table 6 gives world production and consumption, by countries.

RESERVES

The atmosphere contains unlimited reserves of nitrogen. The production of ammonia and other nitrogen compounds requires hydrogen and electrical energy. Hydrogen can be obtained from natural gas, coal, or coke, petroleum, wood, and electrolysis of water or as a byproduct of an electrolytic process.

The quantity of nitrogen associated with coal ranges from less than 1 percent to 3 percent. With the large domestic reserves of coal available, the quantity of nitrogen available from this source is substantial.

USES

Nitrogen compounds are essential to the fertilizer and explosives industries and have many important applications in the chemical industry. Ammonia is the major nitrogen product, and from it many other compounds and salts are made for use in the fertilizer and chemical industries.

The nitrogenous material used for fertilizers can either be applied directly or as mixed fertilizer containing two or more plant-food elements. The principal compounds now in use, in the order of importance, are ammonium nitrate and ammonium nitrate-limestone mixtures, sodium nitrate, ammonium sulfate, anhydrous and aqua ammonia, calcium cyanamide, calcium nitrate, ammonium sulfate-nitrate, and urea. The production and use of more concentrated forms are increasing, reflecting a trend toward higher analysis fertilizers.

Industrial uses of nitrogen compounds are many and varied. Ammonia is the most important industrial nitrogen compound. It is used directly and serves as the basic chemical for the production of many other compounds. Ammonia is employed in the manufacture of wood pulp, plastics, and synthetic fibers and in refrigeration, metal treatment, water purification, and oil refining. Nitric acid is one of the more valuable chemical reagents and also is a basic chemical in the manufacture of explosives and dyes, and as a solvent.

Many other compounds are important to both the fertilizer and chemical industries. Urea is being used in increasing quantities as a fertilizer and as a feed supplement for ruminant animals and new industrial applications are being found for it. New facilities for the manufacture of urea are being built, to supply this increased demand.

REGULATIONS

In normal times the domestic nitrogen industry operates without controls. In wartime, or periods when the supply is extremely short, the United States Government, the Food and Agriculture Organization of the United Nations, and others regulate imports and exports to and from the member countries. During emergency periods, the United States Government also controls the domestic use of nitrogen compounds.

TABLE 6.—*World production and consumption of fertilizer nitrogen compounds, fiscal years ended June 30, 1951–53, by principal countries*

[Metric tons of contained nitrogen]

(United Nations Food and Agriculture Organization)

Country	Production			Consumption		
	1950–51	1951–52 [1]	1952–53 [2]	1950–51	1951–52 [1]	1952–53 [2]
Austria	74, 900	94, 750	100, 700	22, 542	25, 000	25, 000
Belgium	173, 357	214, 269	215, 000	78, 000	77, 500	78, 000
Canada	149, 208	149, 208	161, 208	32, 659	32, 659	34, 500
Chile	242, 583	234, 660	234, 660	8, 369	9, 000	9, 000
Czechoslovakia	30, 000	30, 300	30, 300	40, 000	40, 000	40, 000
Denmark	---------	---------	---------	70, 000	73, 000	75, 000
Egypt	---------	27, 900	31, 000	42, 533	130, 118	143, 400
France	259, 030	285, 000	305, 600	262, 100	280, 000	315, 000
Germany:						
Federal Republic	464, 677	500, 000	520, 000	361, 562	380, 000	400, 000
Soviet Zone	205, 000	205, 000	213, 000	184, 000	191, 000	196, 000
Greece	---------	---------	---------	22, 000	35, 000	40, 000
India	8, 417	37, 998	71, 120	46, 650	62, 998	108, 120
Italy	177, 301	186, 000	225, 000	156, 500	158, 000	170, 000
Japan	414, 595	456, 770	480, 000	442, 000	442, 000	442, 000
Korea, South	---------	306	1, 122	14, 598	48, 189	80, 010
Netherlands	189, 053	226, 500	245, 000	165, 978	160, 000	165, 000
Norway	160, 747	159, 404	164, 795	30, 699	32, 000	33, 000
Peru	35, 440	36, 000	36, 000	37, 680	39, 630	---------
Poland	65, 000	65, 000	65, 000	75, 000	75, 000	75, 000
Portugal [3]	---------	---------	---------	31, 870	33, 000	34, 000
Spain	6, 600	7, 000	7, 000	56, 600	60, 000	60, 000
Sweden	25, 426	16, 028	24, 659	67, 999	72, 542	80, 314
Taiwan (Formosa)	6, 112	13, 849	14, 320	61, 279	76, 215	---------
United Kingdom	275, 000	278, 900	286, 000	218, 800	175, 000	220, 000
United States [3]	996, 000	1, 099, 000	1, 202, 000	1, 166, 000	1, 275, 000	1, 379, 000
World total [4]	4, 011, 103	4, 379, 654	4, 705, 864	3, 930, 054	4, 268, 353	4, 639, 255

[1] Preliminary figures.
[2] Preliminary estimates.
[3] Figures for consumption include overseas territories.
[4] Exclusive of U. S. S. R.; includes quantities for minor producing and consuming countries not listed above.

OUTLOOK

Completion of plants now under construction and proposed will meet the expansion goal of 3.5 million tons of nitrogen a year determined by the Office of Defense Mobilization. Some industry sources predict that there will be excess capacity for the next few years, but agricultural and industrial demand probably will continue to increase. The long-range outlook is for continued growth of the nitrogen industry.

PROBLEMS

Estimation of future requirements is hampered by meager data, and it is a problem to determine the level of production facilities needed to achieve maximum wartime production and at the same time enable the industry to operate profitably during periods of minimum market demand. During past emergencies it was necessary to divert nitrogen-compound supplies from normal markets to explosives manufacture.

Research is essential to find improved methods of producing high-analysis nitrogen materials in more stable and more readily available form.

Other problems include selection of plant sites that will serve the market most economically and yet be situated advantageously with regard to availability of power, raw materials, and transportation; and how to cope with fluctuating demand resulting from varying seasonal requirements, the weather, and farm income.

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OIL SHALE

By

Simon Klosky [1]

OIL SHALE in Colorado, Utah, and Wyoming represents an untapped fuel reserve that appears destined to assume an important place in the liquid-fuel economy of the Nation. This deposit, called Green River oil shale or Colorado oil shale, is the only one known in the United States that is expected to have commercial importance in the foreseeable future, although there are numerous others.

Summary

Oil shale is a sedimentary rock containing solid organic matter that yields oil upon heating to destructive distillation temperatures but is only slightly soluble in organic solvents. At about 800° F. this solid organic material decomposes, yielding oil vapors, gas, and a carbonaceous residue that remains on the inorganic material. Condensation of the oil vapors produces a thick crude oil, which in many respects resembles crude petroleum and which on suitable refining yields gasoline and other liquid fuels, along with numerous byproducts.

Industrial use of oil shale originated in Scotland and France about the middle of the 19th century. In the United States a few years later, shale oil was used to a limited extent with coal oil for heating and lighting. Since then there has been sporadic interest in oil shale in this country, but the current sustained effort to develop the technology of an oil-shale liquid-fuel industry has been the most important by far.

The only oil-shale deposit in the United States thought of as being exploitable, as far as can be anticipated, is the Green River formation of Colorado, Utah, and Wyoming. This formation resulted from deposition of organic and inorganic matter on the bottom of a fresh-water lake and represents a tremendous reserve of oil that can be refined to produce gasoline and other liquid fuels. The reserve in Colorado alone is estimated at 500 billion barrels. Generally speaking, the character of the Green River formation, the properties of the shale, and the refining characteristics of the oil are different from those of the oil-shale deposits in other parts of the world and require mining, retorting, and refining methods especially adapted to these properties and characteristics.

The Bureau of Mines has made some noteworthy technological advancements in developing low-cost mining techniques especially adapted to the Green River oil shale in an experimental mine that uses a room-and-pillar mining system. Low-cost drilling, blasting, loading, and scaling equipment and techniques have been developed. However, intensive studies are also being made on the adaptation of other mining methods to assure an efficient low-cost system of mining.

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[1] Assistant chief, Branch of Oil Shale, Division of Petroleum, Bureau of Mines.

The most important work by the Bureau on retorting of oil shale to extract the oil has been the gas-combustion process. Before that, experimental work was done on other processes to evaluate them; but, upon obtaining successful results using a small pilot plant of the gas-combustion-retort type, work on other processes was discontinued in favor of this one.

When oil shale is heated to 800° F. and higher, the solid organic material, called kerogen, decomposes, yielding oil, gas, and a carbonaceous residue. The gas thus formed is used as the source of process heat in the gas-combustion retort. Virtually all heat transfer takes place between the shale and gases within the retort, so that no heating or cooling equipment is required outside the retort and no process or cooling water is required.

Experimental operation of a 100-barrel-a-day thermal-cracking unit, followed by chemical treating of the distillate fractions of the cracked material, has demonstrated that regular gasoline, diesel fuel, and various grades of burner fuels can be produced satisfactorily by this method, although other methods of refining the oil can be coupled with or used in lieu of this one. The thought in testing thermal methods first was that some of the existing industrial refining facilities might be used at first and that it might be preferable to use these and perhaps initiate additions to them as compared with building all-new facilities. Such techniques as mild hydrogenation and catalytic reforming could be applied to thermally cracked distillate.

In addition to the aforementioned products shale oil can be a source of jet-engine fuel, high-octane gasoline, and numerous byproducts.

A number of foreign countries have oil-shale operations involving a wide variety of mining and processing methods. However, the production in these countries is relatively small.

The consensus among authorities on petroleum supply and demand—both in industry and in Government—is that in the not far-distant future the domestic supply of petroleum in the United States probably will reach a peak and then begin to taper gradually, whereas the anticipated national requirement for liquid fuels will continue to mount. Inasmuch as the United States is already a net importer of oil, more and more dependence on oil imports or domestic sources of liquid fuels other than petroleum or both is obvious. Although the Bureau of Mines makes no attempt to predict when an oil-shale industry will start or how fast it will grow, the outlook for such an industry appears favorable. From an economic viewpoint, estimates of the cost of producing shale oil indicate that if shale gasoline were sold on the west coast at the current price of petroleum gasoline the manufacturer would obtain a return on his investment. This return may not be great enough to attract capital immediately, but the influencing factors can very well increase the attractiveness before long if advancements are made in oil-shale technology and the trend toward greater exploration and production costs in the oil industry continues.

BACKGROUND

Liquid fuels are being produced commercially from oil shales in a number of foreign countries (*12*) [2] but at the time of this writing there is no oil-shale industry in the United States, although the largest known reserve in the world is here. Because the various bodies of oil shale throughout the world were formed during different geologic periods, each deposit differs from the others. This, with different economic climates and variations in other factors, has occasioned the development of numerous processes for extracting the oil and refining it. Table 1 shows some of the differences, as well as similarities, among the principal European oil shales.

From the table it is readily apparent that the European oil shales vary widely in richness and composition. The richness varies from 4.8 percent oil for the shale from Severac, France, to 29.8 percent oil for the shale from Estonia, which is only exceeded in richness by the Australian (not shown).

Even the inorganic part of the shale varies considerably from a low carbon dioxide (and high silica) content for Swedish shale up to a high carbon dioxide (and low silica) content for the Estonian shale. By way of comparison, Colorado oil shale averages 12 percent oil and 16 percent carbon dioxide.

TABLE 1.—*Comparative analyses of European oil shale* [1]

Country	Scotland [2]	France				Sweden		Spain: Puertollano	Germany: Württemberg	Estonia
Sample		Autun A	Autun D	Severac	St. Hilaire	Kvarntorp I	Kvarntorp II			
Fischer assay:										
Oil .... percent	8.2	9.7	5.4	4.8	9.5	5.7	7.6	17.6	6.0	29.8
Water .... do	2.2	3.2	2.0	1.3	1.4	2.0	2.0	1.8	1.4	3.2
Spent shale .... do	86.6	84.0	91.3	92.2	85.9	8.72	85.5	78.4	89.9	61.5
Gas and loss .... do	3.0	3.1	1.3	1.7	3.2	5.1	4.9	----------	2.7	5.5
Specific gravity 60°/60° F	0.881	0.902	0.892	0.935	0.910	0.982	0.954	0.901	----------	----------
Analysis of shale:										
Ash (uncor.) .... percent	76.4	69.8	78.7	66.9	62.7	70.7	69.0	62.8	67.8	39.1
Mineral CO₂ .... do	----------	1.9	7.2	20.5	8.4	0.1	0.8	2.3	13.7	16.5
Organic carbon .... do	12.33	18.79	8.18	7.41	22.25	18.84	19.0	26.01	10.27	34.77
Total hydrogen .... do	1.95	2.46	1.30	1.01	2.36	2.02	2.16	3.38	1.29	4.30
Total nitrogen .... do	0.46	0.58	0.26	0.15	0.54	0.35	0.42	0.55	0.18	0.34
Total sulfur .... do	0.73	1.22	0.30	2.38	2.32	5.44	6.43	1.68	2.88	1.85
Properties of shale:										
Specific gravity .... solid	2.218	2.028	2.304	2.419	2.011	2.090	2.101	1.795	----------	----------
Heating value .... B.t.u./lb	2,540	3,810	1,620	1,450	4,330	3,870	4,050	5,380	2,542	6,979
Coking tendency	None	None	None	None	None	None	None	None	----------	----------
Ash fusion point .... ° F	2,300	2,175	2,100	2,300	2,175	2,150	2,150	2,320	----------	----------
Analysis of ash:										
SiO₂ .... percent	55.71	55.13	57.27	34.91	47.03	62.42	57.53	56.63	43.24	25.13
Al₂O₃ .... do	25.07	27.56	17.10	13.06	18.26	17.64	19.45	27.65	11.84	6.27
Fe₂O₃ .... do	9.89	9.27	6.35	5.27	7.64	10.71	13.06	9.12	10.30	5.91
CaO .... do	2.63	1.66	6.93	36.73	17.83	1.19	2.29	2.63	14.75	48.14
MgO .... do	3.08	1.89	5.96	2.26	3.53	1.70	2.12	2.17	2.51	2.35
SO₃ .... do	0.89	0.29	0.93	5.90	0.39	0.93	1.57	0.51	8.38	9.39
Undetermined (mostly alkalis) .... do	1.71	4.20	5.46	1.87	5.32	5.41	3.98	1.29	8.98	2.81

[1] Compiled by A. S. Houghton, Bureau of Mines, Rifle, Colo.

[2] Analyses made by Bureau of Mines, Laramie, Wyo.

HISTORICAL SKETCH

According to Gavin (*10*), 53 companies were distilling coal and shale, mostly in the Eastern States, in 1860; however, the advent of the Drake petroleum well in 1859 and the volume of petroleum that soon followed ended this early industry.

Owing to the rather high price of petroleum after the first World War, there was rather widespread interest in oil shale in the 1920's and pilot plants were erected, including two by the Bureau of Mines (*16*). However, large new discoveries of petroleum in the United States caused a decrease in the price of crude petroleum, and oil-shale activity was forced to halt.

Beginning as far back as 1850 a promising industry started in Scotland which has continued uninterruptedly ever since. Although industrial developments were begun earlier in France, they were soon terminated and were not revived until much later. In the early

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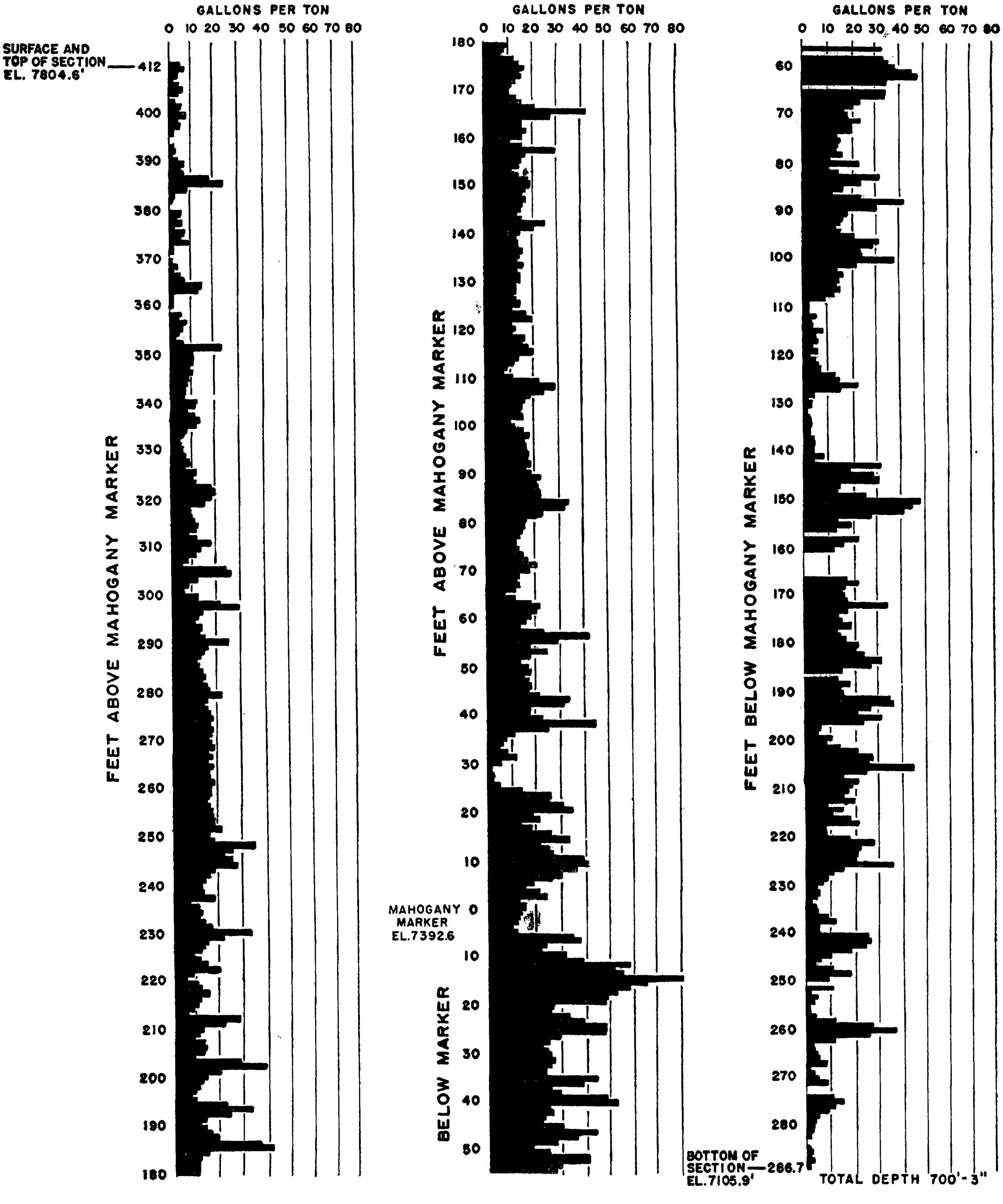
[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.



TOTAL LENGTH OF SECTION-698.7 FEET
316 FT. OF PLUS-15-GALLON SHALE..............74 FT. OF PLUS-30-GALLON SHALE

SELECTED SECTIONS

LENGTH	FROM	TO	GALLONS PER TON	TONS PER SQUARE MILE	BARRELS OF OIL PER SQUARE MILE
92 FT.	+25 FT.	-67 FT.	30.97	172,060,000	126,874,000
133	+25	-108	27.11	255,382,000	164,843,000
155	+47	-108	25.00	300,641,000	178,953,000
391	+163	-228	19.25	795,942,000	364,807,000
697.7	+411	-286.7	15.79	1,463,134,000	550,069,000

FIGURE 1.—Hole D, Drilled by the Bureau of Mines on Naval Oil Shale Reserve No. 1, Garfield County, Colo.

Oil yields of oil-shale beds in the Evacuation Creek and Parachute Creek members of the Green River formation on the north side of the middle fork of Parachute Creek (SW¼SW¼ sec. 11, T. 5S., R. 95 W., 6th P. M.) as in dicated by the Fischer method at the Bureau of Mines Laramie, Wyo., station.

1900's Australia was mining oil shale and exporting it to other countries for conversion to oil and gas.

Still later in the 1920's Estonia, Spain, and Japan built oil-shale plants, but it was only in the later 1930's that South Africa and Sweden joined in this movement.

MINERALOGY OF GREEN RIVER OIL SHALE

It is rather paradoxical to call the material of the Green River formation oil shale, as it contains no oil as such and is not a shale. The organic matter is a solid material rather than a liquid oil, and the rock is a laminated marlstone rather than a true shale. Bradley (*3*) describes Green River oil shale as "a fine-grained sedimentary rock containing organic matter which was derived chiefly from aquatic organisms or waxy spores and pollen grains, which is only slightly soluble in ordinary petroleum solvents, and of which a large proportion is distillable into artificial petroleum."

In the Rocky Mountain States—Colorado, Utah, and Wyoming—oil shale occurs in the form of thick, sedimentary beds virtually undisturbed and outcrops on the faces of the cliffs and ravines, where the rest of the formation has been eroded away by the Colorado River and its tributaries.

The richness of these deposits is indicated by the core analysis shown in figure 1 of a typical diamond-drill core taken in the vicinity of the Bureau of Mines station at Rifle, Colo.

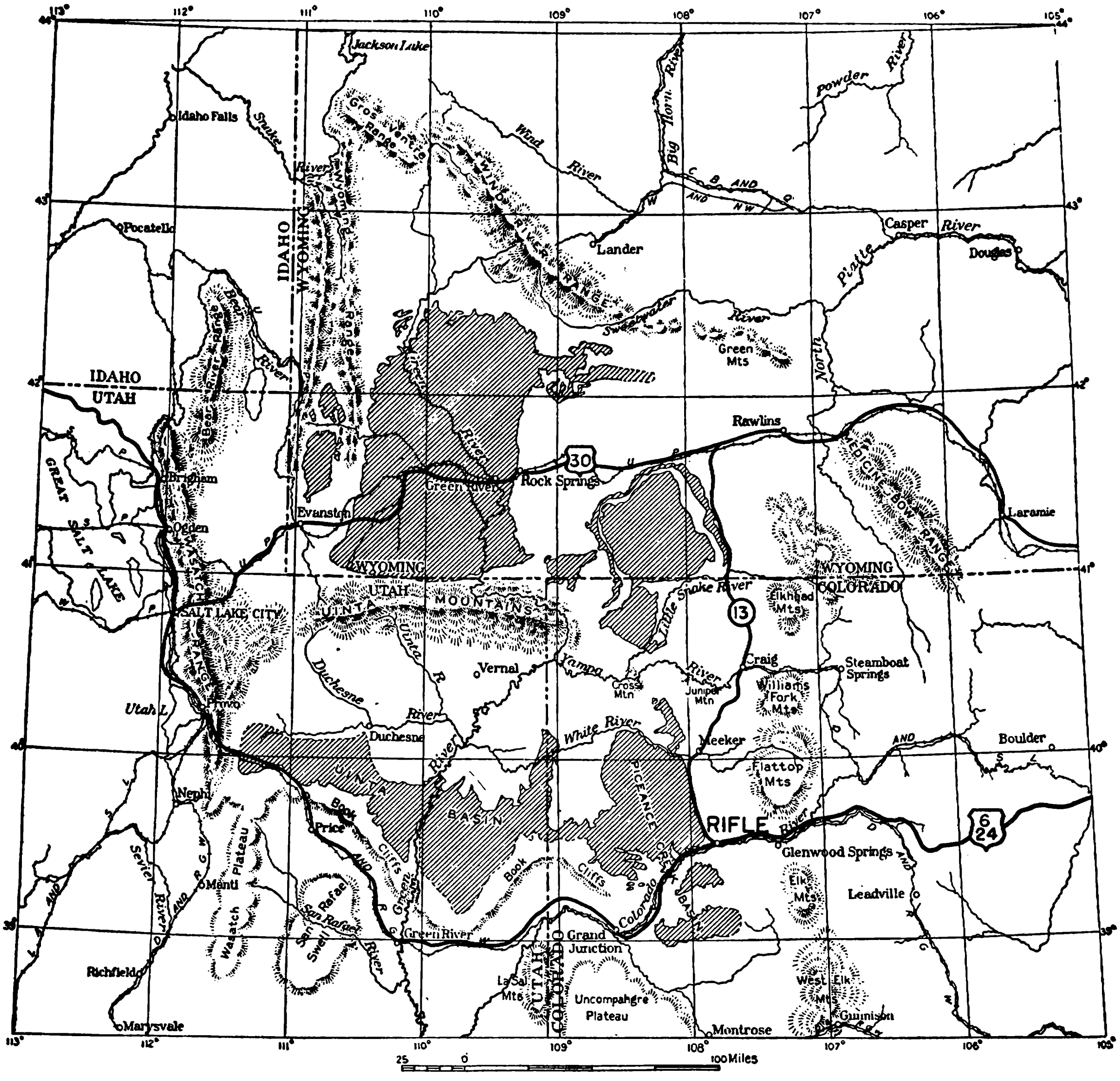


FIGURE 2.—Oil-Shale Reserves in Colorado, Utah, and Wyoming.

COLORADO SHALE-OIL RESERVE

Although not enough data have been assembled to estimate the total reserve in Colorado, Utah, and Wyoming (see fig. 2), analyses of cores obtained by diamond drilling have shown that at least 1,000 square miles in the 2,500-square-mile oil-shale area in northwest Colorado is good minable shale. From this 1,000-square-mile area it is estimated that at least 300 billion barrels of oil can be recovered. Using Belser's (*2*) "in-place" estimate of nearly 500 billion barrels, allowing for a 33-percent loss in mining and crushing, and assuming a recovery of 90 percent of the Fischer-assay value of the shale gives the round figure of 300 billion barrels of recoverable oil.

The 300-billion-barrel recoverable reserve in Colorado is in a continuous series of beds approximately 500 feet thick, which averages 15 gallons to the ton. Within those strata is a measure approximately 100 feet thick that averages 25 gallons to the ton.

As is usually the case in mining operations, it is anticipated that the richer material will be mined first, since that offers the greatest profit incentive.

It is estimated that about 100 billion barrels of shale oil can be recovered from the Mahogany ledge alone. This figure is derived from Belser's (*2*) "in-place" estimate of 75 billion barrels, making reasonable allowances for mining and processing losses.

In an earlier survey of the oil potential of the United States, Winchester (*37*) estimated oil reserves of other deposits in the United States along with that of the Green River formation.

DEVELOPMENT OF OIL-SHALE TECHNOLOGY IN THE UNITED STATES

The Bureau's current research and development program began as a result of passage of the Synthetic Liquid Fuels Act by the Congress in 1944 and embodies both basic research and engineering-development studies. The former, including development of analytical methods, studies of the structure and composition of oil shale and shale oil, and laboratory- and bench-scale process-development investigations, is carried on at the Petroleum and Oil-Shale Experiment Station, Laramie, Wyo.; the latter, including an experimental mine as well as pilot- and engineering-scale experimental retorting plants (and before July 1, 1954, an experimental refinery), is conducted at the Oil-Shale Experiment Station, Rifle, Colo.

Many oil companies, chemical companies, universities, and other research-minded organizations and individuals have made research and small-scale development studies relating to oil shale, most of them in cooperation with the Bureau.

Processes under development appear to have good possibilities for commercial application, and it is anticipated that further development of some engineering problems associated with these processes will result in obtaining the necessary experience and information for designing and operating industry-scale plants.

PROPERTIES AND COMPOSITION OF GREEN RIVER OIL SHALE AND SHALE OIL

The properties of Green River oil shale and shale oil were compiled by Boyd Guthrie. His results were published in 1938 as a Bureau of Mines bulletin (*11*). The origin of oil shale and the properties and methods of testing shale oil are extensively covered in this bulletin, which includes 85 footnote references to oil-shale papers published up to 1938.

The properties of the Green River shales in the various strata of the Colorado deposit have been investigated more recently by Stanfield and his coworkers (*31*). The properties of six selected shales are given in table 2.

The properties of oil shale and shale oil have been thoroughly studied by Thorne and his coworkers at the Bureau's experiment station at Laramie, Wyo., during the past 10 years (*35*). These studies included an improved method of analyzing shale oil (*32*) and a very scholarly paper on the composition of shale oils (*34*). From this paper table 3 has been taken, which shows the effect of method of retorting on the composition of shale oils produced from Colorado shale.

Studies at Laramie also have included work on the isolation, composition, and constitution of kerogen and more detailed studies of the components of shale oil have been started (*24*).

Some progress has been made on identification of the nonhydrocarbon constituents in shale oil, although this is a difficult and time-consuming task. Details of this work are reported in the Annual Reports for 1949, 1950, and 1952 (*24*, *25*, *27*). Research also is being conducted to determine the constitution of kerogen; and success is being achieved, especially by oxidation with permanganate (*26*), by hydrogenolysis (*27*), and more recently by solvation (*29*).

Studies on the thermal decomposition of kerogen were carried out early in the program (*25*), and more recently conversion of shale to oil by thermal solution (*13*) and high-temperature retorting (*28*) was investigated.

Some of the physical properties of oil shale have also been studied, such as the specific heat (*23*), the heat of retorting (*25*), and the relation between specific gravity of oil shale and its oil content (*29*).

TABLE 2.—*Physical and chemical properties of 6 selected Colorado oil shales*

Oil-shale sample number[1]	9	8	7	2	1	10
Fischer-assay yield:						
Oil, weight-percent	4.0	10.4	13.8	21.9	23.6	28.7
Water, do	0.5	1.4	1.5	1.2	1.1	1.5
Spent shale, do	94.4	85.7	82.1	72.3	70.4	63.6
Gas, do	1.1	2.0	2.2	3.9	4.2	4.6
Loss, do	--------	0.5	0.4	0.7	0.7	1.6
Oil from Fischer assay:						
Specific gravity, 60°/60° F	0.925	0.930	0.911	0.918	0.919	0.918
Viscosity, 100° F	20.71	23.72	18.19	17.10	17.12	17.28
Gross heating value	18,510	18,330	18,680	18,580	18,510	18,440
Pour point, ° F	80	75	85	80	80	75
Sulfur, percent	0.67	0.62	0.46	0.62	0.71	0.72
Spent shale:						
Organic residue	2.9	4.2	4.8	9.9	8.9	11.4
Gross heating value	80	250	330	1,160	1,090	1,250
Coking tendency	None	None	Slight	Moderate	Heavy	Heavy
Coke fracture	--------	--------	3	20	90	180
Mineral CO_2	15.9	23.2	21.7	14.4	14.9	15.3
Sulfur	0.61	0.51	0.65	2.02	1.92	2.44
Shale dried at 221° F.:						
Gross heating value	1,020	2,340	3,080	5,510	6,010	7,000
Specific heat (77°–400° F.)	0.244	0.267	0.280	0.309	0.316	0.334
Sulfur, percent	0.62	0.56	0.73	1.96	1.99	1.86
Nitrogen, do	0.28	0.54	0.44	0.58	0.66	0.81
Mineral CO_2, do	16.7	20.0	18.9	10.9	9.9	10.0
Water-soluble, do	0.17	0.29	0.24	0.63	0.60	0.79
Benzene-soluble, do	0.7	1.3	1.3	2.6	2.2	2.1
Ash, do	76.7	62.2	60.3	54.7	53.6	46.6
Fusion point, ° F	2,275	2,275	2,225	2,050	2,050	2,225

[1] Arranged according to increasing richness of the shale.

TABLE 3.—*Properties of Colorado shale oils*

Retorting method	Crude oil (weight-percent)		Distlllate, boiling up to 315° c. (volume-percent)		
	Sulfur	Nitrogen	Saturates	Olefins	Aromatics
N.T.U	0.74	1.78	36	36	28
Pumpherston	.77	1.57	30	38	32
Gas flow	.61	2.10	27	43	30
Royster	.67	1.97	27	41	32
Union Oil Co	.71	1.89	--------	--------	--------
Parry	.87	1.81	26	46	28
Gas combustion	.69	2.13	27	44	29
1,200° F. (high-temperature)	.82	2.46	10	36	54
1,500° F. (high-temperature)	.76	3.08	1	1	98

MINING

As a part of its oil-shale research and development program, the Bureau of Mines has operated an experimental mine (*9*) in the Mahogany ledge near Rifle, Colo.

In addition to supplying material for retorting and refining studies, an important part of the mining work is selection and development of methods, procedures, and equipment for mining oil shale on a large-scale commercial basis.

The physical characteristics of the Green River oil-shale deposit—flat, horizontal, formations that can be mined without artificial support—are unusually favorable to low-cost mining.

The cliffs, in which the oil-shale deposits are visible, rise to an elevation of over 8,000 feet—2,000 feet above the retorts. The original mining plan was to develop the mine on three levels. An advance heading 27 feet high was to be followed by two 23-foot-high benches. An upper and a middle adit were driven to open up the two top levels. Pillars 60 feet square were left 60 feet apart in staggered rows for support.

Cost studies were made of mining test runs after the top heading and middle level were well advanced; they indicated that a 2-level system with a 39-foot advance heading and a 34-foot bench would be more economical and would also eliminate the need for blending, as shale from either level would be of the same grade. A 39-foot advance heading was started later by taking up 12 feet of bottom in the 27-foot headings.

A new, lower adit was driven to develop the 34-foot bench and open a section of the mine to the full 73 feet. Figure 3 is an elevation of the two-level system.

The large underground rooms permit a highly mechanized operation to be utilized. Many types of specialized equipment were designed and developed for use therein.

A mobile air compressor was built to supply compressed air and water for operating a multiple-drill-heading jumbo. This electrically operated unit supplies 1,540 cubic feet of air a minute to the drills through short hoselines, eliminating costly pipelines and much line loss.

The drill jumbo consists of two movable platforms on a framework on a diesel truck. Each platform mounts two long-feed percussion drills. Two miners drill a 96-hole round 15 feet deep in the 60-foot-wide by 39-foot-high heading in an 8-hour shift.

Several types of mobile units have been designed and built for loading blast holes at height and for scaling (barring-down) operations. The most successful is a self-leveling platform on a hydraulic-powered boom, which is mounted on the bed of a diesel truck. Two men working on this platform position themselves as needed by foot-operated, remote-control switches and scale down a face 60 feet wide by 39 feet high in 1 shift.

The broken rock is loaded into diesel dump trucks with a 3-yard electric shovel. Bulldozer tractors are used for pushing up and for cleanup work.

The cost of drilling blastholes obviously represents a large percentage of the overall mining expense. Accordingly, much of the research has covered drilling.

A test drill unit built especially for drilling experimental benching rounds has been used to develop bits that will withstand the service required of them and has shown conclusively that rotary is considerably superior to percussion drilling for vertical blastholes.

Recently a test machine for drilling horizontal holes of heading rounds has been built, and from the few tests that have been made to date it is believed that continued research and development work along this line will result in rotary drilling altogether for commercial oil-shale mines.

In 1953 a rock fall occurred near the lower portal in an area that had been exposed to weathering for over 5 years. The slab that fell was about 20 inches thick. From an analysis of all the conditions that contributed to the fall, it was concluded that the time factor was very important; that is, after a few years there appeared to be a tendency for certain layers to part where the bonding between them was not very strong; hence it was concluded that roof bolting would be desirable. A series of pull tests was made with a wedge-type roof bolt, and satisfactory results were obtained. A 1-inch-diameter iron bolt was adopted, which, installed under a torque of 300 foot-pounds, failed at 24,000 pounds per square inch tension. Approximately 4,200 roof bolts were installed in the experimental mine.

On February 28, 1955, another roof fall occurred. This time the parting of the rock that fell from the layers immediately above was a little above the upper extremity of the roof bolts, which are 6 feet long.

At the time of this writing the full significance of this rock fall has not been delineated; however, for some time rather extensive effort has been devoted to studies of alternate mining methods and on modifications of the present room-and-pillar system, using some of the low-cost drilling, blasting, loading, and scaling techniques that have been developed at Rifle.

RETORTING

Although at present there is no commercial production of shale oil in the United States, considerable effort has been directed toward developing an oil-shale retorting process adapted to the type and location of oil shale, as well as the economic conditions prevailing in this country. Taking these factors into account dictates three major requirements that must be met by a retorting process if it is to be successful:

1. United States petroleum demands are measured in fantastically large figures—during 1944, 5 million barrels per day, in 1953 almost 8 million, and in 1975 one forecast is 14 million. Shale-oil production, to have significance as a supply of liquid fuels, also must be on a large scale, perhaps 50,000 barrels per day for a single plant. To avoid excessive multiplicity, with attendant high capital and operating costs, individual processing units must handle

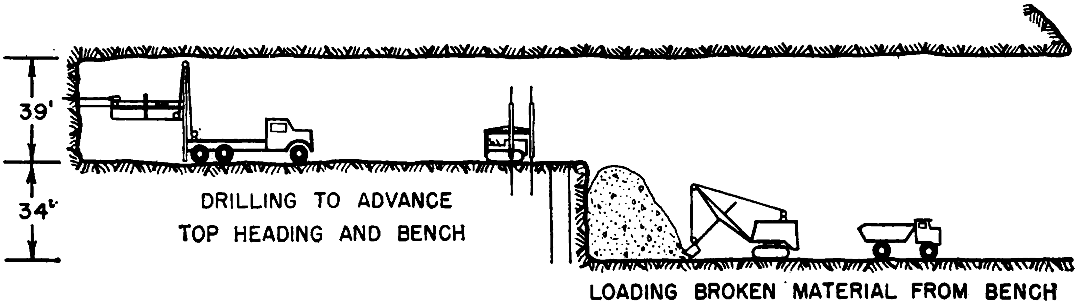


FIGURE 3.—Two-Level Mining Operation.

large tonnages of shale; therefore, a requirement for the successful retort is a high throughput and operability in units of large individual capacity.

2. When kerogen, the organic matter in oil shale, is pyrolyzed, three products are formed—an oil, a gas, and a carbonaceous residue. Almost 30 percent of the heating value of the shale is contained in the gas and the carbonaceous residue. In order not to use a part of the shale oil as a fuel, thereby decreasing the already small (10–12 weight-percent) yield of oil, it is axiomatic that a successful oil-shale process derive its energy from the gas, the carbonaceous residue, or a combination of the two.

3. Colorado's principal oil-shale deposits are in a semiarid region, where the only water source is the Colorado River. Irrigation is practiced extensively, and the total water supply during periods of low river flow is allocated for this and other purposes. Although water could be made available for use in oil-shale processing by storage during periods of high stream flow, it would be most expensive; therefore, a third axiom is to use little or no process water in shale-oil production.

The Bureau of Mines is developing a process that meets these requirements. It is known as the gas-combustion (7) process. Work is in progress to solve engineering problems associated with scaleup of the process, but performance of gas-combustion pilot plants at the oil-shale experiment station near Rifle, Colo., has been very encouraging.

In the gas-combustion retort shale flows downward by gravity countercurrent to a rising stream of gas, which is burned in a central zone of the retort. Combustion, retorting, condensation of the oil, and recovery of sensible heat from the spent shale are all accomplished in a single vessel. Both the ash and the product stream are cold as they leave the retort; thus, no external cooling is required, and conventional, low-cost equipment can be used to remove the ash as well as to recover the oil.

Figure 4 is a flow diagram of the process. About midway in its downward passage through the retort, the shale bed is heated to the retorting temperature by hot combustion gases generated by burning recycle gas, air being injected as indicated in the diagram. Spent shale leaves the retort after passing through a gas seal, and this flow of shale is controlled by the rotor at the bottom of the retort. Cold product gas is introduced at the bottom of the retort and flows upward through the heat-exchange zone, where it is preheated by the hot spent shale, which in turn is cooled. The preheated gas stream flows through a mixing chamber, where it is

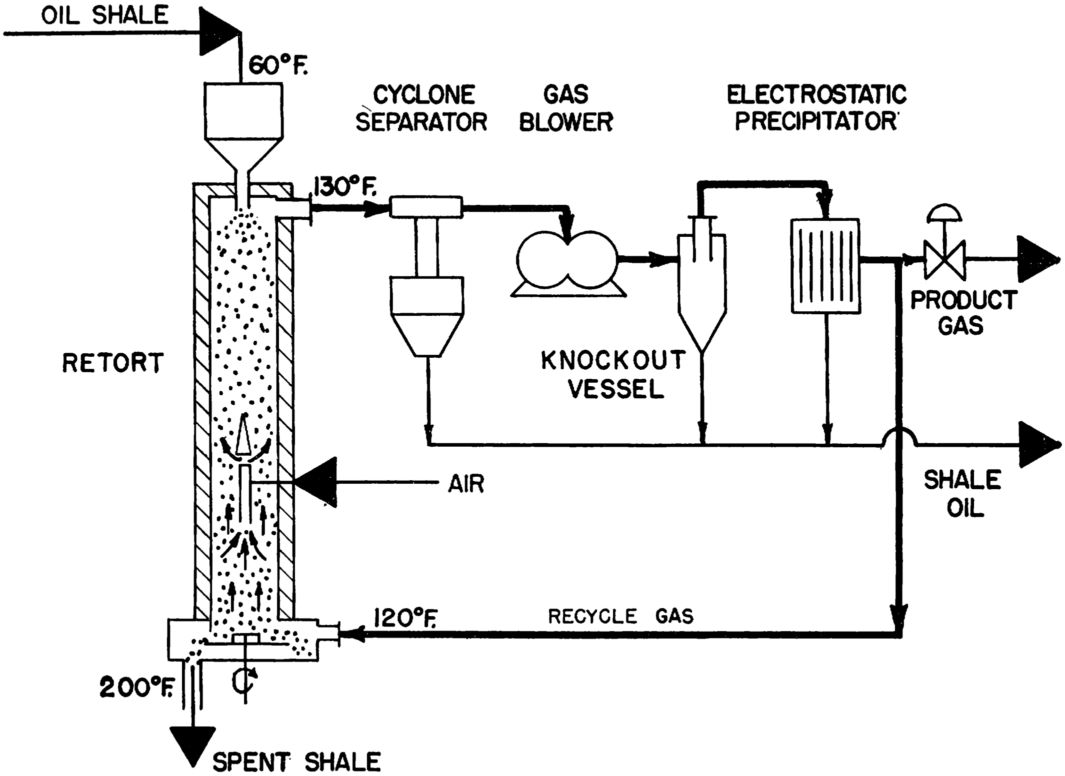


FIGURE 4.—Gas-Combustion Oil-Shale Retorting Process.

mixed with air and then into the combustion zone; here the combustible constituents in the gas mixture and some of the organic residue remaining on the retorted shale are burned. The hot combustion gases formed pass upward into the retorting zone, heating the shale and causing conversion of the kerogen. Continuing upward, the gases sweep the oil vapors and shale gas generated in the retorting zone through the product-cooling zone, where the

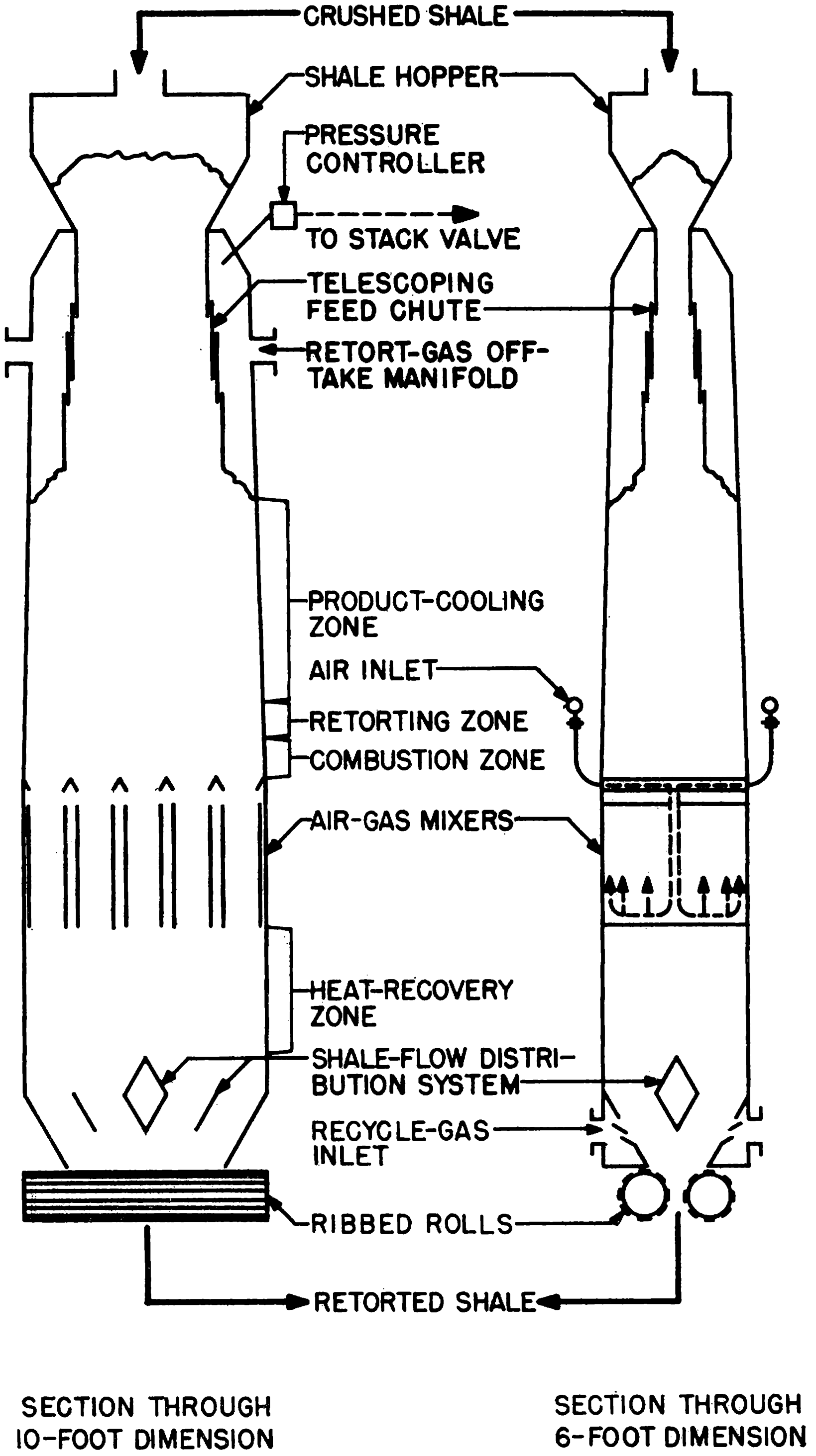


FIGURE 5.—Retort Vessel. Gas-Combustion Retort 3.

oil vapors are condensed by the cold, incoming raw shale. As the vapors are condensed, minute droplets of oil are formed, which remain in the gas stream, for the most part, forming an oil mist (*29*). Excessive refluxing of product oil in the top portion of the shale bed does not occur because the oil mist is stable, and the mist particles are not agglomerated to an appreciable extent upon passing through the shale bed.

The relatively cool stream from the retort then passes through separators for mechanically removing the oil droplets from the gas stream, for example, cyclones, wet-type centrifugal separators, and electrostatic precipitators. After removal of the oil, part of the gas stream is recycled to the retort, and part of it is vented. Figure 5 shows a drawing representative of the 150-ton-a-day experimental engineering-scale plant for developing the process and the equipment.

Perhaps the most important feature of the gas-combustion process—one that makes its successful operation possible—is control of conditions to form a stable shale-oil mist which will remove liquid oil from the top of the retort at a low temperature. In the retorting zone the oil is liberated as a vapor. As this vapor moves upward through the descending cold shale, it is cooled to a temperature below the dew point of the oil. If the proper conditions of heat transfer are established, the gas can become supersaturated with respect to the oil, the oil condensing as minute droplets directly in the gas stream rather than as a film on the shale. The oil mist then will be carried out of the retort by the cold gas without any significant loss by impingement on the shale particles. Relatively simple devices, such as cyclones, electrostatic precipitators, or even high-speed gas blowers, will efficiently remove the oil from the gas stream.

Oil from the gas-combustion process has characteristics similar to those of most other Green River shale oils. It falls into the classification of a heavy oil (about 20° A. P. I.) and has a pour point of about 90° F. Sulfur content is moderate (0.7 weight-percent); but nitrogen, present as organic nitrogen compounds, is very high—over 2 percent by weight. Low-boiling constituents are almost absent, as evidenced by the high initial boiling point. A further characteristic of shale oil of this type is a high percentage of unsaturated compounds.

Before the gas-combustion process was developed, the Bureau experimented with several other methods of retorting (*8*). Furthermore, conception of the gas-combustion process itself came about as an evolutionary development during experiments with various modifications of a basic countercurrent gravity-flow retort.

Some oil companies and other investigators also have experimented on a small scale with processes for retorting oil shale. Among the best known processes that have been studied using bench and pilot-plant models are the following: N. T. U. (*23*, *24*), gas-flow (*8*, *23*, *24*, *25*, *26*, *27*), Royster (*36*), dual-flow (*8*), countercurrent (*8*, *24*), thermal-solution (*13*), entrained-solids (*26*, *27*, *28*), Standard Oil Development Co. (fluidized solids) (*8*, *24*), and Union Oil Co. of California (*8*).

REFINING

Shale-oil-refining research and development by the Bureau of Mines and industrial laboratories in the United States have been devoted to adaptation of modern petroleum-refining techniques and to development of new processes more suitable to shale oil. The characteristics of shale oil include differences between this material and petroleum that limit the refining methods that may be applied. For example, catalytic cracking—a common beneficiation process practiced by the petroleum industry—is not applicable to shale oil, as the activity of the common cracking catalysts is inhibited by the nitrogen compounds. Also, some catalytic desulfurization and re-forming processes are not as successful on shale oil as on petroleum. On the other hand, *hydrogenation*, even at low pressures (400 to 1,500 p. s. i.), reduces the sulfur and nitrogen content of shale-oil fractions and stabilizes the more reactive unsaturated components to reduce their gum-and-color-forming tendencies. The nitrogen and sulfur are removed as ammonia and hydrogen sulfide, both of which may be valuable byproducts. Another method of reducing nitrogen and sulfur is to extract compounds containing these by treatment with sulfuric acid and caustic soda. Oxygen-containing compounds also are extracted. Losses of the product are greater than by hydrogenation; but the process is less expensive, and the extracted components may have byproduct value.

The various types of thermal processing common to the petroleum industry have been applied experimentally to Colorado shale oil under a wide variety of conditions on both in laboratories and in the 200-barrel-a-day experimental refinery of the Bureau of Mines at Rifle, Colo. (see fig. 6). Viscosity breaking has produced a crude oil suitable for pipeline transportation. Recycle cracking of the crude oil has produced over 50-percent gasoline-boiling-range material. Distilling the crude oil to coke and a lower boiling distillate than the original crude may be advantageous to subsequent refining. Catalytic or thermal reforming of the gasoline fraction improves its octane rating,

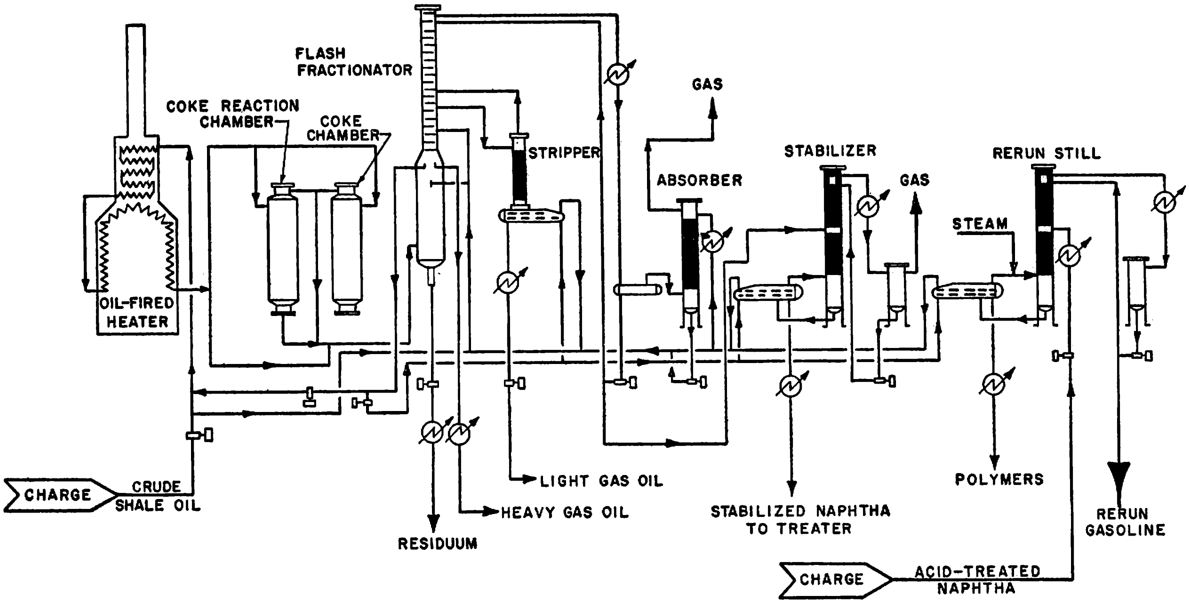


FIGURE 6.—Shale-Oil Cracking Plant.

and high-temperature or vapor-phase cracking of the crude oil or its high-boiling fractions produces highly aromatic products. The highly aromatic oil produced by vapor-phase cracking or by high-temperature retorting may be refined readily by hydrogenation into Premium-Grade gasoline or aromatic chemicals. Figure 7 shows one flow scheme for thermal refining and chemical treating that could be used industrially.

Overall shale-oil refining procedures made up of various combinations of the above mentioned unit operations have been evaluated. The National Petroleum Council in 1951 proposed a procedure whereby crude shale oil would be coked, the resulting gasoline catalytically re-formed, the higher boiling distillate hydrogenated to remove sulfur and nitrogen, a portion of the hydrogenated oil catalytically cracked to gasoline, and the remainder used as diesel fuel. By this procedure about equal quantities of Regular- and Premium-Grade gasoline could be made available. Several alternate plans also have been studied, one of which provides for viscosity-breaking the crude oil near the retorting plant in Colorado, pipelining the product to Los Angeles, and refining it there by conventional recycle cracking, catalytic re-forming, and acid treating, the primary products in that case would be Regular-Grade gasoline and heavy fuel oil.

In 1951 the Rifle, Colo., station of the Bureau of Mines made an exhaustive estimate of the cost of producing gasoline from Colorado shale by technological processes for mining, retorting, and refining, available at that time (*26*). The two methods of refining just described were assumed for this study. With hydrogenation the investment and operating costs are higher, but the value of the products is greater.

Table 4 shows a synopsis of the capital requirements, operating costs, and the resulting costs of the shale gasoline.

TABLE 4.—*Summary of estimated gasoline costs*

	Option 1 (with thermal refining)	Option 2 (with mild hydrogenation)
Capacity of plant____bbl. per day__	[1] 250,000	[1] 250,000
Crude-oil production____	$350,100,000	$350,100,000
Transportation____	134,800,000	72,200,000
Refining____	260,200,000	565,600,000
Other capital____	[2] 127,050,000	197,050,000
Total capital requirement____	$872,150,000	$1,184,950,000
Daily operating expense [3]____	$353,542	$424,677
Total value of sales____	$701,870	$982,495
Value of gasoline per gallon____	$0.12	[4] $0.12-$0.13
Return after taxes [5]____percent__	8.4	11.2

[1] This is crude oil before refining.
[2] Includes operating capital, royalties and start-up expense.
[3] Excluding depreciation.
[4] Part of this gasoline is premium.
[5] On half equity capital after deducting depreciation.

FOREIGN OIL-SHALE TECHNOLOGY

In general, the foreign oil-shale industries have not fared so very well, and most of them have been kept alive through subsidy. Mining and processing methods vary widely from one country to another. Owing to the fact that the United States uses such astronomical quantities of liquid fuels the tendency is to think in terms of large volumes with respect

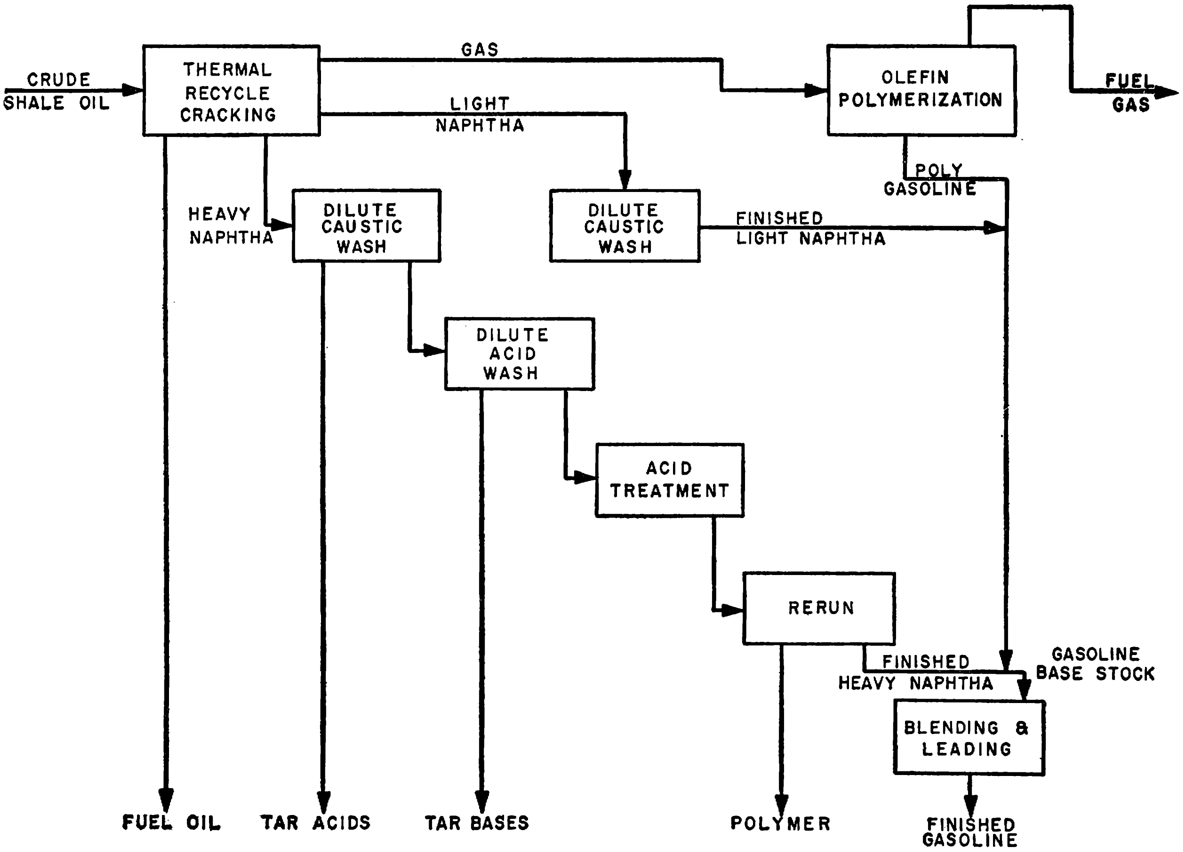


FIGURE 7.—Shale-Oil Gasoline by Thermal Cracking and Chemical Treatment.

to them; the quantities produced from oil shales in foreign lands seem small indeed by comparison. Table 5 shows the tonnages of oil shale mined abroad during the middle 1930's (*27*).

TABLE 5.—*World production of oil shale, 1933–37*

[Metric tons]

(Minerals Yearbook, 1938, p. 896)

Country	1933	1934	1935	1936	1937
Australia	3, 456	3, 532	37	----------	(1)
China	2, 683, 440	2, 105, 765	3, 436, 647	(1)	(1)
Estonia	499, 969	588, 958	604, 288	766, 410	1, 121, 860
France [2]	91, 000	102, 340	88, 473	(1)	(1)
Germany	553	869	722	874	(1)
Italy	918	749	1, 118	889	(1)
Spain	60, 448	37, 783	(1)	(1)	(1)
U. S. S. R	201, 600	206, 400	417, 000	700, 000	(1)
Scotland	1, 419, 410	1, 423, 257	1, 430, 976	1, 432, 036	(1)
Yugoslavia	----------	479	260	137	(1)

[1] Data not available.
[2] Includes some boghead coal.

A study by the writer (*14*) reviews over 2,000 patents relating to the treating of oil shale and its products granted to various inventors; however, relatively few of these matured into operating processes or plants.

The more successful European processes are described by Guthrie and the writer (*12*); of these, the Scottish, Swedish, French, German, Estonian, Manchurian, South African, and Russian are the most important.

In Scotland (*12*) oil shale is mined chiefly from underground deposits, but in recent years the supply to retorting plants has been supplemented by shale from open pits. Although several underground methods have been used, the room-and-pillar technique is favored. A vertical retort called the Pumpherston has been used in this industry for many years and from time to time has undergone improvements. In the latest model, known as the Westwood retort, the heat comes partly from the product gas and partly from the carbonaceous matter on the retorted shale. The primary products obtained by refining the oil from the retort are motor fuel, diesel fuel, solvent naphthas, waxes, gas, tar, coke, and resins.

In Sweden (*12*) the shale lies near the surface and is mined entirely by open-pit methods after the overburden has been removed by a large dragline. Material is processed in three different kinds of retorts: Bergh, Rockesholm, and the Industrial. Heat for the Bergh, which seems to be the most successful, is obtained by burning fixed carbon on the retorted shale in specially designed apparatus. Crude shale oil from the retorts is refined to produce Premium-Grade motor fuel, Regular-Grade motor fuel, kerosine, gas, fuel oil, sulfur, and ammonium sulfate.

France (*12*) has widely distributed oil-shale deposits, none of which are very extensive, but three of them have been worked actively. These are at St. Hillaire, Autun, and Severac-le-Chateau. The only one still active is that at Autun. At Autun mining operations are similar to the underground operations in Scotland. Pumpherston retorts are used at Autun, and at St. Hillaire externally heated shale gas flows horizontally across a downward-moving shale bed in the retort. This process is named for its developing company—Grande Paroisse. In this system, producer gas from a Dutch oven is the source of process heat. At Severac a modified Grande Paroisse retort is also used. Refining is fairly simple, and the products are gasoline, fuel oil, coke, and gas. At Laverne a pilot-plant retort known as the Marecaux has been tested.

Although shale deposits in Germany (*12*) are widely distributed throughout the country, the best known deposits are in Württemburg; the one near Balingen has been in operation up to now. Mining is accomplished by the open-pit technique after a small amount of overburden has been stripped off. Two processing plants have been built—one at Fromern and the other at Dotternhausen—and each employs a different retorting process, the two in use being the Schweitzer and the Meier-Grollman. The latter is sometimes known as the Otto, since that company supplied the retorts. The Schweitzer is similar to the NTU, and the Otto is similar to the Grande Paroisse, except that steam instead of gas is used as the heat-carrying medium.

One of the most successful oil-shale industries in Europe was the Estonian (*12*)—at least until recent years. At present little information on the industry is available. Raw material comes from a very rich deposit of an oil shale, usually called kukersite, bordering the Gulf of Finland. Kukersite was mined by both underground and surface quarry methods. Besides the Pintsch retort, which was installed in the State-owned plant and duplicated by the Japanese in Manchuria, the most successful retort was a tunnel kiln built by a German shale-oil company. In the tunnel kiln, gas, oil, or raw shale was burned and the heat transferred by heat exchangers to a circulating gas stream that passed through shale held by cars that were pulled through the tunnels. The crude shale oil was cracked thermally and treated with caustic soda and sulfuric acid. Principal products were 65- to 70-octane motor fuel, diesel fuel, and impregnating oil containing phenol, which was used for wood preserving.

In Manchuria the oil-shale beds are overlain with coal. Before recent postwar years there was a fairly important industry there. However, with the changed political situation, it is understood that the plants have been dismantled. After the coal was stripped off, the oil shale was mined by open-pit methods. After undergoing several changes the Pintsch retort used in Estonia was adapted for the Manchurian industries and is sometimes referred to as the Fushun retort, as the plant was at Fushun. The oil was refined to give gasoline, kerosine, heavy fuel oil, waxes, and coke.

In South Africa (*33*) an oil-shale industry has been thriving for a number of years without any Government subsidy; however, it is favored by the fact that freight rates on petroleum and petroleum products from seaports to the interior are very high. The shale, or torbanite as it is called, is taken from the ground by a room-and-pillar system of mining and processed in Salermo retorts. Another process called the Davidson rotary retort is used to a smaller extent. The oil is refined to produce gasoline, asphalts, kerosine, gas oil (used for cutback solvent in road-oil blends), gas, and coke. Shale from a room-and-pillar mine in Spain (*12*) is processed and the resultant oil refined. A new plant is designed to produce largely gasoline, lubricants, and waxes. An older plant produces gasoline, diesel fuel, light fuel oil, paraffin, and lubricants.

Although Russia (*15*) has acquired both the Estonian and Manchurian oil-shale industries, before the Second World War the U. S. S. R. had achieved a modest oil-shale industry of its own. A modification of the Pintsch retort had been developed and a large plant erected near Gdov.

Oil-shale operations in Australia (*18*) were discontinued a few years ago owing largely to the high cost of mining oil shale at Glen Davis. Before this the Government had subsidized an oil-shale industry for a number of years. The oil shale (torbanite) was extracted by room-and-pillar mining methods; and Fell retorts, which are quite similar to the Pumpherston, recovered the oil. Other retorting processes were used to a limited extent. The oil was refined, but the only product that was marketed was gasoline, as the coke, gas, and residual fuel were used in the plant.

In Brazil (*17*) there are rather extensive oil-shale deposits, and considerable effort is being expended at present to develop process and engineering technology on a pilot-plant scale to serve as a basis for an oil-shale industry. In the past the Brazilians have tested several retorting and refining processes. Currently the Brazilian Government is engaged in a more intensive and wider scope research and development program. The Bureau of Mines has been cooperating by training some of its technical personnel in the scientific and engineering knowledge that has been accumulated in the United States and in the methods and approaches being followed to further the development of oil-shale technology.

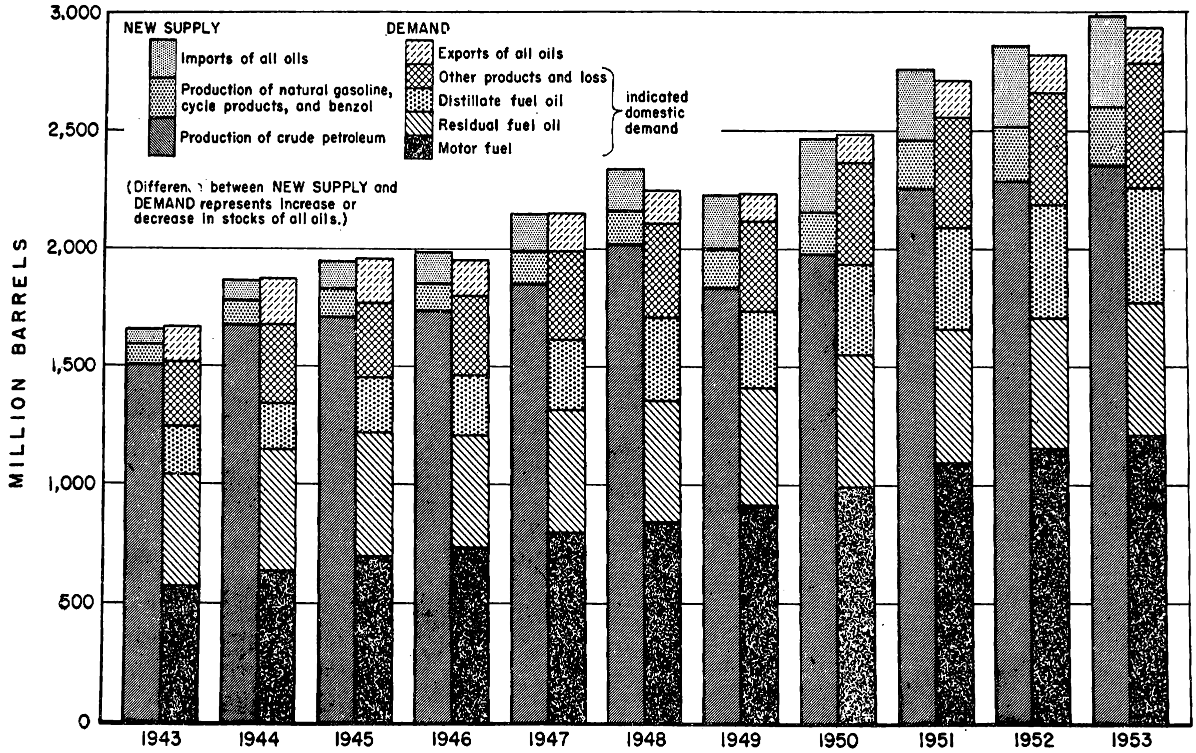


FIGURE 8.—Supply of and Demand for Crude Petroleum and Its Products, 1943–53.

OUTLOOK

The outstanding industrial growth of the United States is due largely to efficient utilization of abundant domestic energy resources. During its early history the country's economy relied mainly on solid fuels—wood and coal—for its supply of energy. As the Nation grew, wood gave way to coal, and by 1925 coal supplied about three-quarters of the energy needs. After oil was discovered its use as fuel increased steadily and then more rapidly when gasoline began to be used for the internal-combustion engine. More recently, liquid fuels have expanded into domestic and industrial heating markets.

Total energy consumption in the United States has increased two-thirds during the past 25 years, and virtually all of this expansion in energy has come from petroleum and natural gas. These convenient liquid and gaseous fuels supply over 60 percent of domestic energy needs today. In 1952, for the first time in the history of the United States, oil alone supplied more energy than did coal. Petroleum is now being consumed at the rate of almost 3 billion barrels a year. The magnitude of this demand may be seen in figure 8.

The tremendous growth of the automobile industry, the evolution of air travel, the increased use of mechanical equipment on the farm, dieselization of railroads, the development of mechanized military forces, and the expanding use of oil for home heating are examples of the shift to liquid fuels.

Several estimates of future needs are shown below. These forecasts are based on detailed analyses of many economic indexes, including such trends as total output of goods and services, population, per capita use, and agricultural needs, as well as other forces that determine liquid-fuel requirements.

TABLE 6.—*Predictions of United States domestic demand for liquid fuels*

[Million barrels a day]

Year	J. W. Boatwright [1]	E. Ayres [2]	PMPC [3]	Average increase over 1950, percent
1950	6.5	6.5	6.5	----------
1955	8.3	8.6	8.1	28
1960	9.18	9.8	9.4	45
1965	9.99	10.4	10.8	55
1970	----------	----------	12.2	88
1975	----------	----------	13.7	110

[1] Boatwright, J. W. (Standard Oil Co. of Indiana), paper presented at 32d meeting of API, Refining Div., November 1952.

[2] Ayres, Eugene (Gulf Research & Development Co.), Sci. American, December 1949.

[3] President's Materials Policy Commission, Resources for Freedom: Vol. 1, p. 108.

At present about 87 percent of United States liquid fuel is obtained from domestic crude oil, and excess productive capacity is available. The remaining 13 percent comes from foreign oil, most of which (85 percent) is imported from countries in the Western Hemisphere, principally Venezuela. If the gap between domestic demand and domestic supply continues to widen, further imports of petroleum can be procured, and supplementary oil can be obtained from oil shale.

Reflecting the views of some fuel technologists, Boatwright states that total United States liquid fuel demands at least through the next 15-year period could be met with domestic petroleum supplies if necessary. Ayres (*1*), who has been studying liquid-fuels availability for years, states that his studies indicate a significant exploitation of oil shale before 1960 and some production of oil from coal shortly after 1960, even with a moderate increase in imports (see fig. 9).

In any event, when synthetic fuels enter the economy of the United States in significant quantities, a very large industrial development will be needed. In terms of capital investment the United States oil industry is one of the largest in the Nation. At the end of 1950 oil-industry gross assets amounted to 33 billion and the industry's annual volume of business totaled 15 billion dollars. Furthermore, in recent years assets of the oil industry have been increasing at the average rate of 3 to 4 billion dollars annually. If by 1975 synthetic fuels supply 10 percent of anticipated liquid-fuels demand, an industry having a capacity of about 1.3 million barrels a day, employing thousands of workers, and doing a business of almost 3 billion dollars annually (at present fuel prices) will be needed.

The cost of making liquid fuels from oil shale is now a little higher than the cost of petroleum products (*27*), but the cost differential between

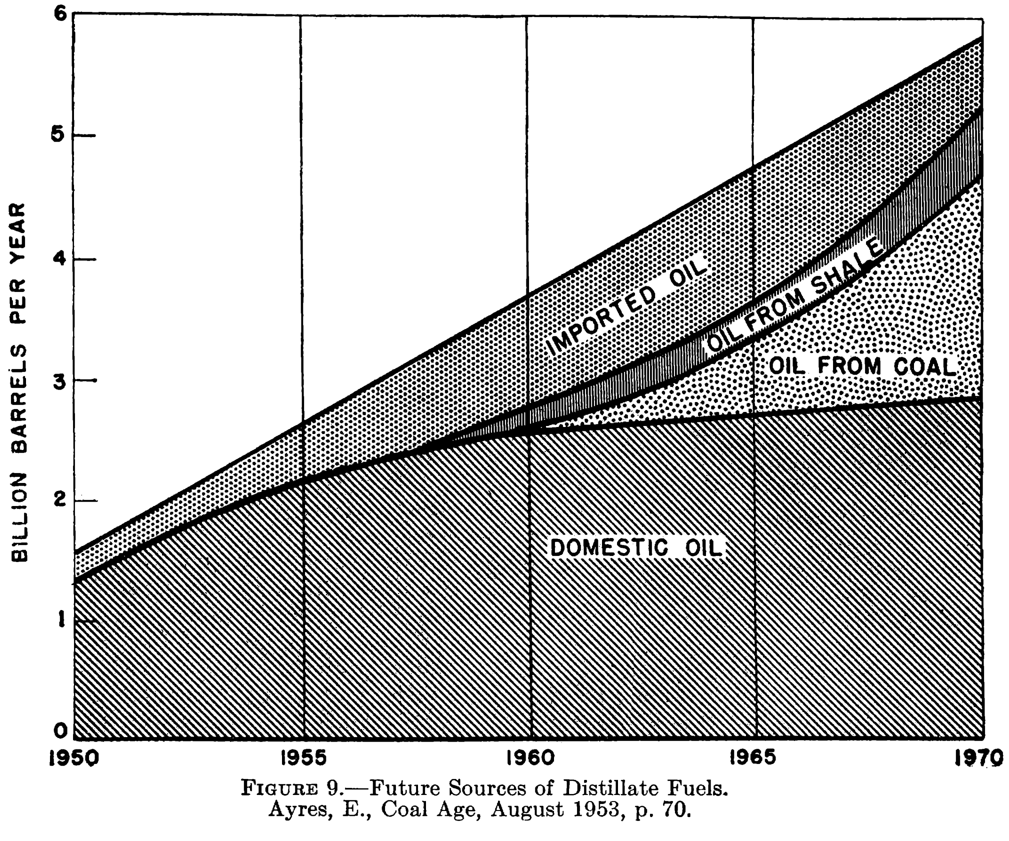


FIGURE 9.—Future Sources of Distillate Fuels.
Ayres, E., Coal Age, August 1953, p. 70.

synthetic and petroleum fuels is not static, and there are indications that it will shift in favor of synthetic fuels. One of these indications is that the cost of petroleum exploration is rising. According to B. Jennings, president of Socony-Vacuum Oil Co., Inc., it cost his company about twice as much to drill a well of given depth in 1951 as it did in 1941. Although inflation accounts for much of the increase, it is only a partial explanation. Deeper wells in more inaccessible areas also increased the cost of finding new sources of oil. In 1942 the average drilling depth was about 3,200 feet; 10 years later—in 1952—it was 4,100 feet—an increase of 28 percent. In addition to the increase in total cost of drilling with increasing depth, unit costs per foot drilled also increase with depth, as shown in table 7.

TABLE 7.—*Typical well costs*

(Based on 1951 drillings by Magnolia Petroleum Co.)

Depth, feet:	*Cost*
5,000	$50,000
7,500	110,000
10,000	200,000
12,500	305,000
15,000	490,000
17,500	700,000

Of the total expenditures by the oil industry, 60 percent or more are for production (which includes exploration, equipment, and natural-gas facilities owned by oil companies). Expenditures by 30 oil companies during 1934–50 are summarized in table 8.

TABLE 8.—*Expenditures for property, plant, and equipment, by 30 oil companies, 1934–50* [1]

Department	Million dollars			Distribution (percent)
	United States	Foreign countries	Combined	
Production	9,935	1,572	11,507	59.2
Transportation	1,882	235	2,117	10.9
Refining	3,119	369	3,488	18.0
Marketing	1,790	242	2,032	10.5
Others	275	6	281	1.4
Total	17,001	2,424	19,425	100.0

[1] Coqueron, F. S., and Pogue, J. E., Capital Formation in the Petroleum Industry: Chase National Bank, Petroleum Dept., February 1952, 39 pp.

In contrast to rising costs for finding new sources of petroleum, the real costs of producing synthetic liquid fuels (which would involve little exploration) are expected to remain constant or to decrease somewhat as improved methods of extracting and refining the synthetic products are developed. Thus, economic factors rather than crude-oil shortages could set the time for the commercial production of synthetic fuels.

PROBLEMS

The only known deposits of oil shale in the United States that appear to have commercial interest are far from any large market for gasoline and other liquid fuels. Colorado, and in fact all the States within a radius of several hundred miles of Colorado, are sparsely populated and could not consume the output of a large oil-shale industry. Consequently, it will be necessary to transport most of the oil produced in Colorado to distant markets. Undoubtedly the first major market for shale-oil products will be the west coast. Since the most economical method of transporting this oil appears to be by pipeline, it will be necessary to build a pipeline, and to make transportation of this oil profitable the pipeline will have to be large. An indicated economic size exceeds 100,000 barrels a day. When the size of the mine, retorting plant, and refining capacity necessary to handle this much oil is imagined it can easily be seen that the total investment for these facilities will exceed 400 million dollars. Naturally no group of investors is willing to risk this amount of venture capital until they are reasonably certain that the technology of oil-shale mining and treatment is well worked out. Consequently, they must be sure that a safe and economical mining method has been developed and that all engineering problems associated with scaling-up the retorting process are solved before any serious attempt is made to design, build, and operate a complete industrial plant.

Because the oil-shale area is sparsely settled and is not industrialized, it will be necessary to build new housing and other community facilities to accommodate the necessary increase in population. This is not thought to pose a major problem but may add to the time required to get into production after an industrial project has been initiated.

After the recognized problems are solved insofar as production in a large pilot plant is concerned, a prototype retorting plant employing a unit retort, near the economical size of a single unit of commercial multiple-unit plant of a projected industry, would disclose any engineering problems that should be solved and serve as a tool for solving them before industry-scale plants are designed.

Although methods of making a fairly wide range of fuels from shale oil at a reasonable cost have been demonstrated, there is room for much research and development of refining of the crude material; however, present knowledge probably would enable refineries to make marketable products from it; and once the technique of providing crude shale oil in quantities and doing it economically is available it seems logical to expect improvements in refining to follow.

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PERLITE

By

Oliver S. North[1]

EXPANDED perlite has been spoken of as the champion of the lightweight aggregates and from the standpoint of rapid growth in recent years comes close to living up to that title.

Summary

Perlite is a rock of volcanic origin containing silica and chemically bound water and perhaps gases and other liquids that, when rapidly heated to a suitable temperature in its softening range, undergoes great expansion because of volatilization within the softened mass of the gases and/or liquids. This expansion ranges from 400 to 2000 percent or more, depending on the inherent nature of the crude rock and controlled furnacing factors.

From a virtual unknown in the commercial world in 1946, perlite had become by 1953 an annual $10 million industry. Mines are now producing in several Western States, and there are processing plants in nearly two-thirds of the States.

Although the major quantity uses of expanded perlite are in construction—plaster, concrete, and loose-fill insulation—many specialized uses have been found for these glassy, inert, white particles.

The technology of expanding perlite has improved considerably in the past 5 years, and members of the industry are constantly working to obtain greater fuel efficiency and a product that has carefully controlled qualities of particle size, hardness, color, etc.

There is need for further study of the efficiency of various types of furnaces, the relationship between the physical and chemical properties of different perlites and their processing and end-product characteristics, and the use of lightweight perlite concretes. There also is need for dissemination of comprehensive technical, statistical, and economic information on this commodity.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

Virtually unknown on a commercial basis before World War II, production of expanded perlite in 1953 was 40 times greater than in 1946. Bureau of Mines statistics indicating the remarkable growth of this industry are presented in table 1.

By 1953 perlite mines were active in 7 Western States; deposits in Nevada and New Mexico furnished 80 percent of the total requirements of crude rock.

In 1953 there were perlite-processing plants in 30 States. The greatest quantity of expanded perlite was produced and sold by firms in California; large quantities were also produced in Pennsylvania, Illinois, New York, New Jersey, and Ohio.

COMPOSITION, PROPERTIES, AND ORIGIN

The term "perlite" originally was defined petrographically as a member of the obsidian family; as an acid volcanic glass, being an essentially amorphous aluminum silicate showing perlitic structure; or as the last product in the acid magma series. As commonly applied by geologists, the perlitic, or onionskin, structure—not the mineralogic nature—was made the sole identifying characteristic. Almost all rocks then known to have perlitic structure were associated with exclusively acidic volcanics.

It has long been known that most volcanic glasses, whether basic or acidic, will tend to expand in some degree when subjected to high heat. Usually this expansion—for pumice, scoria, pitchstone, and the like—is not too marked; for example, only 10 to 200 percent. On the other hand, because of its physical structure and its content of combined water and perhaps gases and other fluids also, perlite will expand from 4 to 20 or more times its original volume.

Perlite is a rock, not a mineral, and therefore is variable in chemical composition. Composition of separate deposits differs, and there is variation even within the same deposit. The chemical composition of a given perlite is very nearly the same, except for the combined-water content, as that of the volcanic rock with which it is associated genetically.

The mode of origin of perlite is debatable. Theories differ to some extent, and no single explanation may be correct for all deposits. Formation of perlite as a surface extrusion has been questioned, and most geologists who have studied the matter lean either to near-surface intrusion, which would permit rapid cooling under moderate pressure, or to hydrothermal alteration of other materials, such as rhyolite or pumice. It may be said that the formation of perlite is due to a combination of such factors as intrusion, pressure, temperature, and alteration, but there is no general agreement as to the exact process.

In hand specimens perlite is seldom easy to identify. It has wide color and textural range; the perlitic structure may not be visible; its specific gravity, although somewhat lower than for most volcanic rocks, is not distinctive; its luster, usually but not invariably "pearly,"

TABLE 1.—*Crude and expanded perlite sold or used by producers in the United States, 1946–53*

Year	Crude perlite				Expanded perlite	
	Sold		Used at own plant to make expanded material		Sold	
	Short tons	Value	Short tons	Value	Short tons	Value
1946	1,500	$8,300	3,100	$16,300	2,600	$92,500
1947	550	3,000	9,900	55,000	7,700	271,000
1948	4,400	29,000	17,700	105,000	18,600	742,000
1949	27,300	193,000	43,800	317,000	52,200	2,385,000
1950	59,802	411,205	41,734	237,957	86,962	4,741,383
1951	110,119	663,981	43,383	194,118	133,175	7,243,298
1952	135,070	873,054	29,775	129,866	154,563	7,997,731
1953	141,282	1,072,065	57,469	367,593	174,461	8,894,735

TABLE 2.—*Expanded perlite sold or used in the United States, by States or districts, 1952–53*

State or district	1952			1953		
	Short tons	Value		Short tons	Value	
		Total	Average per ton		Total	Average per ton
California	28, 419	$1, 202, 603	$42. 31	35, 342	$1, 601, 988	$45. 33
Illinois	14, 562	776, 728	53. 34	11, 127	712, 238	64. 01
Ohio	9, 881	667, 561	67. 56	10, 015	675, 207	67. 42
Pennsylvania	15, 441	938, 690	60. 79	13, 109	810, 965	61. 86
Texas	11, 691	627, 917	53. 71	(1)	(1)	(1)
Other Western States [2]	38, 309	1, 714, 067	44. 74	49, 253	2, 194, 613	44. 56
Other Eastern States [3]	36, 260	2, 070, 165	57. 09	55, 615	2, 899, 724	52. 14
Total	154, 563	7, 997, 731	51. 74	174, 461	8, 894, 735	50. 98

[1] Included under "Other Western States."

[2] Includes Arizona, Arkansas, Colorado, Iowa, Kansas, Louisiana, Minnesota, Missouri, Nebraska, Nevada, New Mexico, Oklahoma, Oregon, Texas (1953 only), and Utah.

[3] Includes Florida, Indiana, Maryland, Massachusetts, Michigan, New Jersey, New York, North Carolina, Tennessee, Virginia, and Wisconsin

approximates that of many siliceous rocks; and its rock associations, formerly considered exclusively rhyolitic, now are believed also to include some of the basic volcanics. Unless from an area previously known to contain perlites, where its general appearance may have become familiar, it can be identified with certainty only by testing for degree of expansion.

Perlite occurs widely in the Western States. None is known east of the Rocky Mountains.

MINING AND PROCESSING

Virtually all perlite is mined by simple open-pit methods. Factors requiring consideration are thickness, hardness, and disposal of overburden, topographic features, mode of transportation, available capital, and the like.

Milling is only a little more complex than the mining and consists of comminuting the perlite rock to the desired mesh-size specifications. Two general types of milling can be used, one being a wet and the other a dry process. All major mills now use dry methods. The wet method minimizes the dust hazard but necessitates a drying operation, in addition to requiring about 15 tons of water per ton of perlite. Water is often in short supply in the vicinity of perlite deposits.

After primary crushing in a jaw or impact crusher, the material is sent to a secondary unit, usually rolls. This is followed by a screen from which the oversize is reground and screened in closed circuit. The material is separated for binning or other disposal with vibrating screens fitted with wire of varying mesh sizes. Two or three screens ordinarily are used. Extreme fines either are sent to waste or stored with an eye to possible future demand for that fraction. The preferred mill combination is one that yields the correct percentages of the desired particle sizes with the least production of dust and waste fines, with the greatest economy of equipment cost and upkeep and the least requirements of labor and power.

Two major types of furnaces lead the field—the horizontal and the vertical. Each has many variations of detail, and there are advantages and disadvantages with each; no one furnace can be considered best for every type of crude perlite used and end product desired. Most of these furnaces are designed so that a current of air will carry out the expanded material and deliver it to a suitable cyclone dust-collecting system, where the desired size separation is effected. In some plants fines are collected in water, and the slurry is sent to waste; this method is particularly useful where gases vented to the atmosphere must be clean.

Perlite furnaces may utilize either gas or oil for fuel. Best results are obtained when conditions of treatment and processing are maintained constant.

In any furnace the size and characteristics of the particles of the finished product are influenced by the particle size of the feed material, the rate of feed, the temperature, and the retention time, in addition to the physical and chemical nature of the crude rock.

USES

About 80 percent of the output of expanded perlite in the United States is used as aggregate—replacing sand—in gypsum plaster. Advantages claimed for perlite plaster include light weight, good acoustic and thermal insulating properties, fireproofing qualities, resiliency, nailability and sawability, ease and rapidity of application, good bonding properties, etc.

As a concrete component, perlite has been used in bricks, blocks, beams, studs, prefabricated units (panels and sections), poured roof decks and walls, and poured floors. Of these, the most important uses are roof decks, prefabricated units, and floors. Its relatively low compressive strength has prevented it from being widely used in bricks, blocks, beams, or monolithic walls.

Perlite concrete is used for grouting oil wells, and loose perlite can be pumped into oil wells to seal off porous formations. Perlite is useful as loose-fill insulation in the voids in concrete-block walls and between wall studs. It is an effective insulator for steam pipes, either as loose fill packed into jackets or as a built-on layer of plaster or lean concrete. As a plaster, loose-fill material, slabs, or blocks, it can be used to insulate refrigerators, refrigerator cars, portable or walkin ice boxes, deep-freeze units, and frozen-fruit shipping boxes. Perlite has been used as a loose-fill medium for embedding hot-steel ingots during shipment and as a substitute for ordinary foundry sand to surround a pouring riser. In the last application the insulating properties of the perlite improve the casting by preventing excessively rapid cooling of the mold.

The use of perlite fines as fillers and extenders in rubber, cleansing compounds, paints, plastics, resins, etc., has been growing steadily, and certain plants now sell large portions of their outputs for some of those purposes. Perlite fines also are used in glazed building tile, sandwich boards, metal-surface plaster, porous supports for catalysts and chemicals in gaseous reactions, and granular carriers for pesticides, in refractory brick, as an abrasive, for agricultural purposes, and for filtering dry-cleaning fluids, alcoholic beverages, and various chemicals. It also is used in water-filtration beds.

Perlite meets competition in all of its fields of use. As a plaster aggregate, it competes in the same market with sand and exfoliated vermiculite. Competitive concrete aggregates include the heavy materials—sand, gravel, stone, air-cooled slag, etc.—and the so-called lightweight materials—expanded clay, expanded shale, expanded slag, exfoliated vermiculite, cinders, pumice, scoria, diatomaceous earth, sawdust, etc. Concrete also is made light in weight by entraining air or other gases rather than by using a mineral aggregate.

Mineral wool and exfoliated vermiculite are important competitors of perlite in loose-fill insulation, and other materials used as fillers include a wide variety of minerals, such as barite, chalk, clay, diatomaceous earth, feldspar, graphite, gypsum, mica, ocher, pumice, and talc, as well as byproduct rock dusts, industrial byproducts, organic materials and numerous manufactured fillers. Diatomaceous earth and sand are the principal competitors of perlite in filtration.

Reserves of perlite rock in the western section of the United States are known to be immense. A geological report prepared for the Union Pacific Railroad showed a proved tonnage of over 400 million tons in southern Nevada, and huge quantities also have been identified in New Mexico, Utah, Arizona, California, Colorado, and Oregon.

ECONOMICS

Although there have been inquiries into the possibility of importing crude perlite into the Eastern States from foreign countries, little if any is now imported. Because of prohibitive freight rates, only minor quantities of plaster or concrete aggregate or other perlite products are exported. Increasing quantities of crude perlite are being exported to processors in eastern and central Canadian cities.

Transportation costs are a large part of the delivered cost of crude rock—in fact, freight charges to eastern producers amount to more than the f. o. b. mine cost of the rock. Even higher freight charges for the bulkier expanded perlite tend to limit its area of distribution around a plant to a radius of from 50 to 200 miles in the north and eastern market areas and from 200 to 400 miles in the south central, west, and southern market areas. New expanders usually try to locate in the immediate vicinity of concentrated housing or commercial construction activity.

Prices for crude perlite rock, ground to customers' specifications, usually range between $6 and $12 per ton, f. o. b. mill. Plaster and concrete aggregate in 3- or 4-cubic foot paper bags sell for from 15 to 35 cents per cubic foot, f. o. b. plant. Extreme fines, when a market for them can be found, often sell for from 25 to 50 cents per cubic foot.

RESEARCH

The Bureau of Mines has studied the thermal characteristics of perlite, and the Bureau of Reclamation collaborated with the National Bureau of Standards in a study of the characteristics of many lightweight aggregates, including perlite. At least one Texas college has experimented on its use in oil wells. Many perlite expanders conduct tests and do experimental work with their product; however, most research is organized by the industry's trade association, the Perlite Institute, which releases the results to its members, building contractors, Government agencies, and interested individuals.

OUTLOOK

The use of perlite in plasters, concrete roof decks and floors, and precast slabs backed up with sheet metal has been growing steadily, and demand for these products continues to increase as more of the expanded material becomes available and the use technology becomes better known. Perlite plaster can be applied either by hand or machine. At least two manufacturers now sell "plaster guns," which apply plaster mechanically at remarkable speeds, and perlite has been found to be a very satisfactory aggregate. Another development contributing to the growing use of perlite is premixed perlite-gypsum plaster, which now is being manufactured and sold by several large gypsum producers.

Perlite has not yet shown any marked relationship to local and regional developments. In most instances it has been simply a matter of building the plant in an active area and then promoting use of the product on a quality basis. Of course, unusual housing-construction activity in an area improves the demand for perlite; and, with the current outlook for a continued high level of residential construction in the next few years, it is expected that demand for this product in the building industry will increase to some extent.

The uses of perlite are so new that it has not reached all of its potential market, either geographically or in variety of uses. The industry faces some serious technical and competitive problems, but despite these it is expected that consumption of this commodity will continue to grow during the next few years.

PROBLEMS

Problems confronting the perlite industry include development of new uses and markets for the material, improvement of processing and use practices, and the same economic problems of cost and distribution that generally face industries.

Volume of business will depend to some extent on successful search for new uses and the extension of currently minor uses. To date, perlite expanders have been successful in finding markets for most of the material that they could produce, except for the finer sizes, and demand for even that fraction has improved.

Many building codes in use today were written long before perlite, and often any other lightweight aggregate, was in use; and the specifications, especially as regard the required compressive strengths, are not in harmony with current construction concepts. An industry problem is to effect needed revisions in building codes to permit the use of perlite where it is serviceable.

Studies of the chemical and physical properties of various perlites are needed to correlate those characteristics with particular processing requirements and qualities in the finished product.

There is need for investigation of the efficiencies and advantages of the various furnace types.

High transportation cost from the areas in which perlites are known to occur to the major market areas is a serious commodity problem.

Compilation and distribution of more detailed and comprehensive technical, statistical and economic information are required to provide the industry and Government agencies with basic facts required for the most effective management and guidance of policy decisions.

Lightweight and semilightweight concretes, including perlite concretes, are being used in increasing quantities, and much study remains to be given to the production technologies and the uses and relative merits of the several aggregates, including perlite. Studies of the use of perlite concrete as a means of permitting the design of buildings with lighter steel frameworks than was formerly possible are needed.

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PETROLEUM AND NATURAL GAS

By
R. A. Cattell[1] and others

THE MAGNITUDE of the petroleum and natural-gas industry and its importance in the everyday life of the people of the United States are best illustrated by the facts that petroleum and natural gas supplied almost 67 percent of the energy furnished by mineral fuels and waterpower in 1954; the petroleum industry's estimated gross assets in 1954 were about $45 billion; the estimated number of employees of the industry totaled over 1.6 million in 1954; and an estimated $4 to $5 billion will be spent in 1955 on expansion of the various divisions of the industry.

The present major product of petroleum is gasoline, the primary use of which is in the operation of motor vehicles. During the first 50 years of the petroleum industry, the products were used primarily for illumination, and for the past 45 years they have been used for power. The entire transportation system, whether it be by land, air, or water, depends to a major extent on petroleum-fuel products such as gasoline, kerosine, residual fuel oil, diesel oil, and, of course, lubricants.

Natural gas, which for many years had only limited areal distribution from its source for use as an illuminant and as a source of heat, now is distributed through large-diameter pipelines to most parts of the United States for use in space heating and as an industrial fuel.

Summary

The first use of petroleum and natural gas in modern times was for light. Now petroleum and its products and natural gas and its products are used primarily for power and for heat. In the United States only about 6 percent of the annual production of petroleum is processed for other uses, such as lubricants, wax, coke, asphalt, road oil, and petrochemicals. Less than 4 percent of the marketed natural gas is used in the manufacture of carbon black; an undetermined but relatively small amount of the marketed gas is used in the petrochemical industry, although this is an industry that is growing rapidly.

Archeologists have found that asphalt was used by mankind as early as 4000 B. C. Many traces of the early uses of petroleum and natural gas have been found in various parts of the world, in addition to many historical references to their use.

The birth of the mammoth petroleum and natural-gas industry commonly is dated with completion of the E. H. Drake well at Titusville, Pa., on August 27, 1859. The resulting "oil boom" in the Pennsylvania area caused drilling for oil to spread to other areas throughout the United States, so that within 25 years oil was being produced in 7 States, and at present 29 States contribute to the Nation's supply.

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[1] Chief, Division of Petroleum, Bureau of Mines.

Accompanying the discovery of oil was the beginning of the refining, transportation, and marketing industries of today. Refining was in the best position to meet the demands of the new oil industry because of a background of experience in distilling "coal oil" from coal and oil from oil shale.

The petroleum industry is best described as being composed of four functional groups: (1) Exploration, drilling, and production, (2) transportation, (3) refining and processing, and (4) marketing. Numerous organizations operate in all 4 fields of activity, although by far the greatest number of organizations or individuals are active in only 1, 2, or possibly 3 of the branches. Although retail marketing of petroleum, natural gas, and their products includes the greatest number of single-unit businesses, the extent of the industry can be visualized from the fact that 44,600 business entities are engaged in the production, refining, transportation, and wholesale distribution of petroleum and its products.

The natural-gas industry functions similarly to the petroleum industry, except for the differences resulting from the nature and the end uses of the primary marketed product, natural gas. Natural-gas transmission and distribution, being of a public utility nature, are subject to Government regulations; however, the natural-gas industry, likewise, is made up of a large number of independent organizations that function in one or more of the categories comprising the natural-gas industry.

The technology of finding petroleum and natural gas, of producing it wisely so that a minimum of waste occurs, of processing it into marketable products, and of transporting first the oil and gas from the wells to the processing plants and then transporting the products to market is extremely complex. Because of this complexity, the industry consists of specialists who generally are capable of performing work in only a very restricted segment of the overall operation of the industry. The geologist who looks for new reserves of oil and gas may be a specialist in only one of the several geologic methods used by industry in locating favorable conditions for oil accumulations and has knowledge only of the rudiments of geophysics as applied to oil exploration. Likewise, a petroleum engineer may be a specialist in reservoir mechanics and know only general facts about other phases of this profession, a chemist may be a specialist only in catalytic processes, and so on throughout the entire industry. The huge body of knowledge that has been developed makes it no longer possible for one to be an expert in all phases of the industry as in the earlier days.

Petroleum and natural gas are nonreplaceable resources; moreover, there is an ultimate limitation to the amount of these resources available in the earth's crust. Although the present available sources are completely adequate, some authorities question whether the present rising demand for petroleum and natural gas or the sudden impact of an increased demand in case of emergency could be satisfied for an extended period if imported petroleum were eliminated. In order that the life of United States petroleum resources may be extended to a maximum, conservation must be practiced throughout all branches of the petroleum industry.

Proved reserves of hydrocarbon liquids, which included crude oil, condensate, and natural-gas liquids in the United States as of December 31, 1954, were estimated by the American Petroleum Institute and the American Gas Association to be approximately 34.8 billion barrels. Proved reserves of natural gas, estimated by the American Gas Association as of December 31, 1954, amounted to approximately 211.7 trillion cubic feet.

Precise estimates of the potential recoverable reserves of petroleum and natural gas in the world or in the United States cannot be made; however, several reputable engineers and geologists, using methods that are available, have made estimates of the ultimately recoverable reserves. For petroleum these estimates range from 150 to 200 billion barrels in the United States and including the Continental Shelf of the United States and Alaska; for the land areas of the earth and its Continental Shelves the estimates range from 1 to 1.2 trillion barrels. The estimated ultimate volume of natural gas recoverable in the United States exceeds 500 trillion cubic feet.

In the discovery, production, processing, and marketing of petroleum, natural gas, and their products, the overall industry continuously faces and solves problems whereby the public is receiving better and more products at lower cost, although the cost of finding and producing oil is increasing. Through research, geological conditions favorable for accumulation of petroleum are being found with geophysical equipment flown by plane, equipment has been developed for drilling wells over 20,000 feet into the earth's crust, natural gas has been made available through the use of large-diameter transmission pipelines from the gulf coast area for domestic consumption as far north as New Hampshire, liquefied petroleum gas is available for rural domestic use where natural gas is not available, dependence on imported rubber for automobile tires has almost been eliminated through synthetics derived from petroleum, and gasoline is increasingly efficient as an automotive fuel. Although these are a few of the more outstanding accomplishments, hundreds more are the direct result of research. By continued research continued benefits will result for the consumer.

BACKGROUND

USES OF PETROLEUM, NATURAL GAS, AND THEIR PRODUCTS

The two principal uses of petroleum are for the production of heat and of power. The statement applies to petroleum in its broad meaning, which includes crude-oil products, natural gas, natural gasoline, liquefied petroleum gas (LP-gas), crude-oil residues, and coke. The first use of petroleum probably was to furnish light. Written records 5,000 years old speak of torches made from pitch, and other old writings describe flaming natural-gas fissures. The first commercial use of natural gas in this country in 1821 was for lighting purposes, and the object of Drake's well in 1859 was to find large quantities of oil that would yield kerosine for lamps.

By 1900 the biggest demand for petroleum was for heat and power, and this demand has continued. A brief survey of the kinds of fuels derived from petroleum and used for heating purposes in and around any large city reveals a diversity of products. Most large cities now are supplied with natural gas that is used for all forms of heating, in the smallest bungalows and the largest buildings. Liquefied petroleum gas also is available in all large cities and in suburban areas for use in about the same manner as natural gas, beyond the ends of the gas mains. For these same uses heating units to burn gasoline also are available. Distillate fuel oils are widely used for all kinds of space heating, especially for homes and small commercial establishments, and the residual grades of fuel oil are used generally in the larger heating units and for generating steam in industrial plants. Petroleum coke can be used in large heating installations and, in the form of briquets, is an excellent fuel for picnic cooking and for fireplaces.

The wide diversity of use of petroleum also is evident in the fueling of internal-combustion engines. Almost every kind of transportation on land, sea, or in the air can be or is powered by petroleum products. In some foreign countries cars even have been run on natural gas. Liquefied petroleum gas is used to power many types of transportation, including passenger cars, buses, and trucks. Gasoline is available within a few miles of anyone in this country. Railroads use large quantities of diesel and residual fuels and could use petroleum coke. Airplanes, submarines, and ocean liners all use petroleum for fuel, and virtually every piece of mechanical equipment, from the smallest to the largest, depends on petroleum lubricants for efficient operation.

These applications of petroleum for heating and for engine-fuel use are evident to everyone. To get an idea of the many other ways in which petroleum affects our daily lives, we have only to look at the myriad uses of a specialty petroleum product that represents less than 0.05 percent of our total demand. This is white mineral oil, or liquid petrolatum (*45*).[2] It is the major constituent of our baby oils and is used widely as a human internal lubricant. It is used in hundreds of cosmetic products, in carbon paper, in pharmaceutical preparations, in throat sprays, and in penicillin processing. Petrolatum keeps candy from sticking to candy-kitchen pans and keeps eggs fresh. It impregnates many types of paper wrappings, including the familiar brown paper at meat counters, and helps to produce most bakery products. It assists in the production of electrical equipment, tobacco, rubber, plastics, textiles, and leather. And this single product has many more special applications, including new ones developed every day.

The petroleum industry depends on large volume demand for its economically successful operation, and this large volume demand has been responsible for the development of over 5,000 products. Six principal products fill 90 percent of the total demand. The remaining 10 percent goes to such products as wax, asphalt, specialty products, and chemicals. These products are a minor factor in overall petroleum demand but represent most significant portions of the total available supply of many important chemicals or commodities. An example is the production of benzene, a vital chemical in many industries. By the end of 1953 the petroleum industry produced benzene at a rate of 100 million gallons a year or about 25 percent of the total benzene supply (*12*), yet this represents only about 0.1 percent of the total refinery production of all products.

PETROLEUM-REFINERY PRODUCTS

Table 1 shows the refinery production for the major classes of products (*16*, *1954 ed.*). The

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

discussion following this table is based on reference (*16, 1954 ed.*) unless otherwise indicated.

TABLE 1.—*Output of petroleum products at refineries in the United States, 1954*

Product:	*Production, thousand barrels*
Gasoline	1, 232, 989
Kerosine	122, 305
Distillate fuel oil	542, 278
Residual fuel oil	416, 757
Jet fuel	46, 550
Lubricants	53, 243
Wax	5, 290
Coke	24, 284
Asphalt	74, 912
Road oil	7, 213
Still gas	102, 552
Liquefied gases	34, 169
Losses or gains and other	2, 545
Total output	2, 665, 087

GASOLINE

The major use for gasoline has been in spark-ignited internal-combustion engines, principally in passenger automobiles, trucks, and aircraft. Some stationary engines require gasoline, as do some watercraft engines, tractors, and construction-equipment engines. Small quantities of gasoline are used for heating and lighting in special gasoline-burning equipment. Of the total gasoline indicated in table 1, 9 percent was natural gasoline blended with other refinery stocks to produce finished motor fuel. An additional 36 million barrels of natural-gas liquids was blended into motor fuel outside refineries so that this source contributed 12 percent of the total motor-fuel supply. The total also includes 94,868,000 barrels of aviation gasoline for both civilian and military use. In 1954 over 57 million passenger cars and trucks were in operation in this country, and operation of aircraft by civilian airlines and private flyers continues to increase.

KEROSINE

About 70 percent of the kerosine is used for heating and cooking. About 3 percent of the total is consumed as tractor fuel, and the remainder is used as lamp fuel and in orchard heating, weed burning, dust control, jet fuel, and other minor uses.

DISTILLATE FUEL OIL

The total production of this product in 1954 was 542,278,000 barrels, of which over half was used for heating purposes. Railroads purchased about 77 million barrels (14 percent), primarily for diesel-engine fuel. The third largest demand (8 percent) was for use in smelters, mines, and the manufacturing industries. Gas and electric-power plants, watercraft (including diesel tankers), and the Armed Forces used smaller quantities.

RESIDUAL FUEL OIL

For residual fuels smelters, mines, and manufacturing industries were the largest consumers, using about 30 percent of the total domestic consumption of 522,099,000 barrels. Other demands included gas and electric power plants 14 percent, watercraft 21 percent, space heating 15 percent, and railroads, oil companies, and the Armed Forces lesser quantities.

LUBRICANTS

Lubricants composed only 2 percent of refinery output, yet these vital products are essential for the operation of our mechanized civilization. They are used for virtually everything mechanical, from the smallest to the largest machines. 53,243,000 barrels of lubricants was produced in 1954. About 52 percent of this quantity was sold as automobile-type oil and grease for use in passenger cars, trucks, tractors, motorboats, motorcycles, and airplanes. As an example of the different grades and types possible under this classification, the Navy alone lists 30 different types and grades for its use, exclusive of any oils for aircraft engines. Exports of all types of lubricants totaled 15,168,000 barrels. Production of all types of greases was about 7 percent of total lubricant production. Aircraft used about 2 percent of the total lubricant output. Fifteen percent of all oils and greases was utilized for nonlubricating purposes. Representative uses were as absorbents, in air filters, as antirust oils, as flotation oils, fogging oils, and fruit preservatives, in heat transfer, in inks, in paints and putty, in paper processing, in making detergents, for quenching and tempering metals, in sprays, in textile processing, in electric transformers, and as wood preservatives. The balance of the output was consumed in lubricating industrial, marine, and railroad equipment.

WAX

Wax is another important product from petroleum but still represents less than 0.2 percent (5,290,000 barrels) of the total oil processed. Paraffin wax is recovered from petroleum distillates and has a relatively low melting point. Microcrystalline wax is extracted from residuals and has a higher melting point. About 90 percent of the total wax produced is paraffin wax. Wax is another oil product in common use, as it is employed to waterproof paper containers, such as those used for milk, bread, and frozen foods, to make candles, and as an ingredient of most floor and furniture polishes. It also is used to keep powder dry and as an insulating material. About 80 percent of the wax produced in this country is required for paper treating. Wax also is used in matches, for coating fruit and vegetables, in cosmetics and printing inks, in leather and textile processing, in pharmaceuticals, in rubbermaking, and in many other minor applications (*2*).

COKE

Coke is another solid product from petroleum and results from the thermal cracking of heavier oils. The 1954 production was equivalent to 24,284,000 barrels. About 10 percent is used as fuel in the refineries themselves. Of the remainder, 13 percent was exported, and over 25 percent is consumed in aluminum production and the balance in abrasives and graphite electrodes and as fuel. Sometimes coke is mixed with the clay in brick to yield a better finished-brick color, through the reducing action of the burning coke. Coke also goes into dry-cell-battery carbons and carbon brushes for electric motors and is one of the fuels for smudgepots (*2*).

ASPHALT

Domestic production of asphalt in 1954 was 13,620,000 short tons, equivalent to 74,912,000 barrels of liquid. Asphalt is best known for its major use as a road-paving material, and about 68 percent of the total is consumed in all kinds of paving, such as public highways, roads on private property, sidewalks, automobile parking areas, and airfield runways. It is interesting to note that about 7.6 million tons was used on public highways compared to 7.4 million short tons of portland cement for the same purposes. An additional 2.4 million tons of cement was employed in constructing bridges, culverts, and other structures.

Large quantities of asphalt were consumed for roofing purposes in 1954, as in previous years, and amounted to 22 percent of the total. The remaining 10 percent of the total asphalt sales was consumed as a binder in briquetting, as a waterproofing material, as a component of dust-laying oils and paints, as a pipe coating, and for paper lamination (*2*, *22*).

ROAD OIL

Road oil, used for low-cost road surfacing, accounted for 7.2 million barrels of total output in 1954. This material has no other major use.

STILL GAS

Modern refinery operation results in the production of large volumes of gases. The principal use for these gases is in refineries as fuel. Often these gases are treated to recover some of the more valuable hydrocarbons before being burned, and this is one of the sources of petrochemical raw materials and liquefied petroleum gases. The still gas produced in 1954 was equivalent to 102,552,000 barrels of oil.

LIQUEFIED PETROLEUM GASES

Liquefied petroleum gases (commonly termed LP-gases) are produced not only in refineries but also in natural-gasoline and cycle plants. The latter two sources account for the larger part of the total recovered. The figure of 34,169,000 barrels (table 1) represents LP-gases produced in the refineries. This refinery product sometimes is referred to as liquefied refinery gas—LR-gas. Production outside refineries in 1953 was 111,735,000 barrels (*18, 1953 ed.*). The tremendous gain in the sales of these products in recent years can be realized from a comparison of 1953 consumption to 1940 and 1945. The 1953 consumption was about 16 times as much as in 1940 and almost 4 times the 1945 sales. The principal LP-gases are butane and propane. Propane represents 57 percent of total output, butane 14 percent, and mixtures of the 2 the remaining 29 percent. These fuels are employed very much as natural gas is used in homes where gas is not available. Domestic (household) and commercial use consumed 50 percent of the total, chemical industries 20 percent, synthetic rubber 8 percent, industrial uses 7 percent, gas enrichment 4.5 percent, and fuel in internal-combustion engines 10 percent. Use in tractors has been increasing rapidly each year, and most manufacturers now offer tractors equipped to burn these fuels. Use of LP-gases in trucks extends engine life and decreases maintenance costs. Many bus and some taxi companies are using LP-gases for fuel for the same reasons.

Principal farm uses for LP-gases are for heating, cooking, and as fuel for tractors and other power equipment. LP-gases are used additionally in chicken brooders, for poultry scalding, and for flame weeding. They are used for tobacco, alfalfa, and rice drying and as a fuel for irrigation pumps. Industrially LP-gases are used for heat-treating metals, including carburizing steel, preheating, and cutting and brazing. They can be used for many of the same applications as natural gas in industry and are used as standby or peak-period-demand fuel by gas utilities and industrial plants (*2*).

NATURAL GAS

In 1954 the total marketed production of natural gas was 8.667 trillion cubic feet, a 3-percent increase over 1953 production. The thermal value of this production represents well over half that for petroleum products consumed. Consumption of natural gas, classified according to its major uses, is shown in table 2. The following discussion is based on Bureau of Mines Quarterly Natural Gas Report 15 (*22*).

TABLE 2—*Use of natural gas in the United States, 1954*

Use:	*Quantity, million cubic feet*
Residential	1,937,300
Commercial	605,400
Field (drilling, pumping, etc.)	1,578,800
Carbon-black manufacture	247,000
Other industrial and utilities	4,048,200
Total consumption	8,416,700

An additional 66,500 million cubic feet was added to storage and 183,900 million cubic feet exported, used to increase the quantity of gas in pipelines, or lost in transmission.

RESIDENTIAL

Natural gas is consumed in residences in nearly all parts of the country, and only a few States are still outside the reach of the natural-gas lines. In 1954 about 25 million residential customers were served by natural gas, including reformed gas and mixed gas. The gas is used for heating, for cooking, for water heating, for laundry driers, for garbage disposal, and other household applications.

COMMERCIAL

As here indicated, the commercial uses of natural gas are primarily for space heating of stores, hotels, and other business establishments and for use in laundries and bakeries; these applications continue to grow. In 1954 the number of consumers was over 2 million, who used a total of 605,400 million cubic feet of gas.

FIELD USE

Large quantities of gas are used in oil and gas fields for powering pumping equipment, drilling equipment, and processing equipment. Natural gas was consumed as fuel for pumping and compressor stations on many oil and gas pipelines and was used at virtually all natural-gasoline plants.

CARBON-BLACK MANUFACTURE

Large quantities of gas are used in the manufacture of carbon black, an industrial commodity of great importance. Details of the industry are covered in the Carbon Black chapter of the Bureau of Mines Minerals Yearbook (*15, 1953 ed.*). Out of a total of 52 plants operating in 1953, 34 were in Texas. The remainder are in Louisiana, New Mexico, Kansas, Oklahoma, California, and Arkansas. The total output in 1953 was 1,610,437,000 pounds, of which about two-thirds was from natural gas and the balance from liquid hydrocarbons. Domestic consumption was 1,200,871,000 pounds. By far the largest part of this—over 94 percent of the total—was used in the manufacture of rubber tires and other rubber products. The manufacture of ink, including ink for magazine and newspaper printing, consumed almost 4 percent. Less than 1 percent went into paints and the remaining 1 percent into minor uses, usually those where a black color is required. Increasing amounts of carbon black are now being made from the heavier petroleum oils. Carbon black from petroleum oils has certain properties making it more desirable for some applications than carbon black from gas.

PETROLEUM-REFINERY FUEL

Many refineries near gas fields use natural gas for fuel purposes because it is the cheapest fuel available. About 7 percent of the total use is by refineries. Natural gas is used at compressor stations on many long-distance transmission lines for the same reason.

PORTLAND-CEMENT PLANTS

Natural gas is ideally suited for use as a fuel in portland-cement plants as a heating gas for passage through the inclined rotary kilns used in the manufacturing process. Natural gas likewise is used in other industries where hot gases at closely controlled temperatures are necessary.

OTHER INDUSTRIAL USES

Large quantities of natural gas are used by electric utility companies for the generation of electricity. The choice of a fuel in this particular application is based on economics, and natural gas cannot be used advantageously in many localities; however, the amount so used is significant, equaling about 10 percent of total use. Other natural-gas users are the mining industry, food-processing plants, the textile industry, the lumber industry, papermakers, manufacturers of glass and ceramics, smelting plants, and the fabricated-metals industries. In the glass and steel industries, as in many other uses, natural gas is used because of the inherent ease of temperature control associated with its use.

PETROCHEMICALS

Two recent newsworthy items in the petroleum industry are petrochemicals and platinum catalysts. Both are significant examples of petroleum technology, and developments in both fields have been marked by strong competition.

Petrochemicals originated in 1919–20, and some 7,800 short tons was produced in 1925. For 1953 the total production exceeded 13 million short tons, valued at over $3 billion, with a forecast of $4 billion output in 1955. Over 100 companies, with more than 200 plants, form the petrochemical industry. Over 50 of the companies are oil companies or their subsidiaries, and a dozen are joint ventures of oil and chemical companies. About 150 companies will be operating in this field soon.

On a weight basis, the 13 million tons of petrochemicals was only 3 percent of the total refinery output of products in 1953. It is estimated that 82 percent of all aliphatic chemicals was derived from petroleum, and petroleum sources accounted for 47 percent of the total aromatic chemicals produced.

Petrochemicals can be defined as chemical compounds made with a petroleum hydrocarbon as one of their basic components. Actually the general understanding goes beyond this to include pure hydrocarbons and other materials derived wholly or in part from petroleum but not generally classed as chemicals. Examples are ammonia made from natural gas and synthetic rubber—a mixture of hydrocarbon polymers. Carbon black, essentially pure carbon, is included in this category.

The major petrochemicals are ammonia, synthetic rubber, carbon black, ethylene, propylene, butylene, butadiene for rubber, acetylene, benzene, toluene, styrene, polyethylene, phenol, formaldehyde, acetaldehyde, methyl alcohol, ethyl alcohol, isopropyl alcohol, butyl alcohols, ethylene oxide, ethylene glycol, acrylonitrile, acetic acid, acetic anhydride, and acetone. The minor products are numbered by the thousands.

AMMONIA

Ammonia has been made from petroleum-derived raw materials since 1931, and 9 percent of the synthetic ammonia produced in 1940 came from this source. The total 1940 production was 474,000 short tons. In 1953 output was 2.4 million tons, and about 75 percent came from petroleum. Ammonia is produced by a variety of plants and is produced as a byproduct of coal carbonization; from synthesis gas made from coal; from synthesis gas derived from natural-gas and petroleum fractions; and from hydrogen produced in electrolytic cells and catalytic petroleum reformers.

The greatest increase in production for the past few years has been in new and converted plants using natural gas as the primary source of the hydrogen-containing synthesis gas. In the next few years new plants will be built to utilize the hydrogen available in refinery catalytic reforming units. Still other plants may be based on new developments in the coking industry. Ammonia production is expected to reach 3 million tons in 1955.

SYNTHETIC RUBBER

Synthetic rubber production in 1953 approached 1 million short tons. The principal ingredients are butadiene and styrene. In processing, about 3 parts of butadiene is mixed with 1 part of Styrene. About 75 percent of the butadiene required was produced from petroleum sources, and all of the styrene was based on petroleum benzene and ethylene. Rubber requirements took about 30 percent of the total styrene available. Carbon-black production has been discussed in preceding pages; carbon black is compounded with both natural and synthetic rubber, and each short ton of rubber is loaded with an average of about 675 pounds of carbon black. Natural-rubber consumption in the United States in 1953 was 41 percent of the total rubber use.

The Government rubber program during World War II involved operation of 15 polymerization plants with a total capacity of 860,000 short tons. It is expected that these Government-owned plants will be sold to private industry in 1955. This disposal will have a profound effect on the rubber industry, and a period of uncertainty in outlook may prevail for several years.

ETHYLENE

Ethylene is the most important petrochemical intermediate today, and total production (over 1 million short tons) was exceeded only by ammonia production. The bulk of ethylene production results from the cracking of ethane and propane, recovered in refinery and natural-gas operations. By 1955 output probably will reach 1.6 million short tons. A new plant, opened in 1953, has a capacity of 300 tons a day (about 10 percent of the total capacity of all plants).

Ethylene's major derivatives include ethylene oxide, ethyl alcohol, ethyl benzene, ethyl chloride, ethylene dichloride, ethylene dibromide, and polyethylene. Some of these are used directly, and most of them enter into further reactions to provide still other petrochemicals. Ethylene oxide primarily is an intermediate for such other products as ethylene glycol and acrylonitrile. Ethyl alcohol has many direct uses, but a substantial portion is utilized for making other petrochemicals, such as acetaldehyde.

ETHYLENE OXIDE

The total production of ethylene oxide in 1953 was 245,000 short tons, and capacity is expected to double by the end of 1955. The principal use of ethylene oxide was for the manufacture of ethylene glycol—the familiar "permanent" antifreeze. Ethylene glycol production in 1953 was 312,000 tons, the major share derived from ethylene oxide. Other uses of ethylene glycol were in the manufacture of synthetic fibers, cellophane, explosives, hydraulic fluids, resins, and adhesives.

Acrylonitrile is another major product of ethylene oxide, with 1953 production exceeding 28,000 tons. It is an important ingredient in the nitrile type of synthetic rubber, highly desirable for certain special uses. New production facilities for acrylonitrile use acetylene as the starting material. Acrylonitrile supplies a large market in the synthetic fiber field, including base materials for Orlon, Acrilan, and Dynel. Capacity for acrylonitrile will reach 100,000 tons by 1956.

ETHYL ALCOHOL

Ethyl alcohol, one of the oldest big-usage organic chemicals, long was a product of the fermentation of molasses and sugar-containing waste materials. In 1953 the total production was 237,000 tons, 72 percent from petroleum raw material, and in a few years this percentage may exceed 90. Ethyl alcohol is used directly as a solvent in many applications and an intermediate for acetaldehyde, for a number of esters and ethers, for vinegar, ethyl chloride, butadiene, styrene, and chloral.

ETHYL BENZENE

Ethyl benzene finds its major application as an intermediate in the production of styrene. Ethylene and benzene are combined to form ethyl benzene, and this in turn is dehydrogenated to yield styrene. About 400,000 tons was converted to styrene in 1953.

OTHER ETHYLENE DERIVATIVES

Other important ethylene derivatives are ethyl chloride, ethylene dichloride, and ethylene dibromide. All three are most important to the manufacture and use of tetraethyllead in gasoline to reduce or eliminate knock in gasoline engines. Ethyl chloride is an essential component in manufacturing tetraethyllead, and ethylene dichloride and dibromide are used together as a scavenging agent in tetraethyllead mixtures to prevent lead deposition in engines. Ethyl chloride also is an important source for ethyl cellulose. Total production of all 3 compounds probably exceeds 600,000 tons a year. Both ethyl chloride and ethylene dichloride were produced at the rate of about 260,000 tons each in 1953.

The only other major use for ethylene dichloride is for conversion to vinyl chloride. This process competes with vinyl chloride made from acetylene. About 50 percent of the yield is based on petroleum material. Vinyl chloride is the principal ingredient in the important vinyl-type resins. Polyvinyl resins are used for furniture coverings, curtains, luggage, raincoats, wire covering, and gaskets. About 200,000 tons of all types of vinyl resins, for which vinyl chloride is the major raw material, probably was produced in 1953.

POLYETHYLENE

The polymerization of ethylene yields polyethylene—the white, translucent plastic widely used in electrical insulation, as packaging material for foods, cosmetics, and pharmaceuticals, as liners for paper bags and containers, and for other purposes. Production in 1953 was of the order of 85,000 tons, and planned increases indicated 200,000-ton capacity by the end of 1955.

PROPYLENE

Propylene has grown more slowly than ethylene as an important petrochemical intermediate, and 1953 production was 600,000 short tons. The best-known propylene derivative—made entirely from petroleum raw materials—is isopropyl alcohol. Other derivatives include acetone, allyl chloride, acrolein, detergent compounds, polypropylene, octyl alcohol, glycerol, and cumene. Further processing involving propylene derivatives yields methyl methacrylate, basis for a thermoplastic resin known as an acrylate. Lucite and Plexiglas are two brand-name examples. In 1948 the first plant to produce glycerol from petroleum propylene came into production. Although synthetic glycerol is still only a small part of the total glycerol produced, it is a potential source for filling larger future demands for propylene.

ISOPROPYL ALCOHOL

Production of isopropyl alcohol in 1953 was 450,000 tons, with some increase expected in the next 2 years. Isopropyl alcohol is used directly as an antifreeze, a solvent, and rubbing alcohol. Its largest use is for conversion to acetone, another chemical intermediate of major interest.

ACETONE

The acetone supply picture underwent a change in 1953, when acetone became available in quantity as a byproduct from the synthesis of phenol, using cumene as a starting material. Up to this change, most acetone was produced from isopropyl alcohol, with a minor part supplied by oxidation of butane. The two major processes are competitive costwise, and the newer process will supplant the old, since acetone is a byproduct.

Acetone production in 1953 was 255,000 short tons, with a 1955 forecast for 300,000 tons. Acetone is used directly as a solvent, and its major solvent use is in the cellulose acetate industry. One pound of acetone is required to produce 8 pounds of product in that industry. Paint and lacquer mixtures used about 25,000 tons of acetone, and about 5 percent of acetone production served as a solvent for bottled acetylene. Acetone is an intermediate for other chemicals, principally diacetone alcohol, in turn an important intermediate and an excellent solvent as well as a component of brake fluids. Other special solvents, such as mesityl oxide and methyl isobutyl ketone, are derived from diacetone alcohol, and still other solvents are made from acetone. Ketene from acetone is a necessary compound in the synthesis of acetic anhydride.

ACETIC ANHYDRIDE

Acetic anhydride from acetone competes with three other methods of synthesis. The acetone route uses about 40,000 tons a year of acetone. Total anhydride production in 1953 was 400,000 tons. The largest outlet for acetic anhydride is in the production of cellulose acetate.

OTHER PROPYLENE DERIVATIVES

The production of synthetic glycerin (glycerol) from propylene is a significant petrochemical development. Propylene is combined with chlorine to yield allyl chloride. The allyl chloride is reacted further to yield other products, which in turn are reacted to give the desired glycerol. Glycerol is used in such products as alkyd resins, cosmetics, cellophane, tobacco, dynamite, pharmaceuticals, and toothpaste.

Allyl chloride also is the source of allethrin, substitute for pyrethrins, widely used in household insect sprays, grain sprays, and aerosol insecticides. Other products stemming from allyl chloride include important resin bases used as primers, baked finishes, floor varnishes. and can coatings.

Dodecyl benzene is a major factor in detergent manufacture and is made from propylene polymers and benzene. The total production in 1953 probably exceeded 100,000 tons, and over 75 percent of synthetic detergents is based on petroleum raw materials. Part of the increasing demand for phenol now is being met by phenol production from cumene, which in turn is being synthesized in part from propylene and benzene. Phenol demand in 1953 was 190,000 tons. Acrolein is another product based on propylene that has a bright future. Isobutyl and normal butyl alcohols are derived in part through propylene processes. Total butyl alcohol supply in 1953 was 195,000 tons, and virtually all new sources of supply were based on petroleum processing.

BUTYLENES

Butylenes generally are byproducts of refinery cracking and are used in liquid-fuel preparation as well as for petrochemical manufacture. The primary raw material is isobutylene, and the major product is butyl rubber (GR–1). Small amounts of high polymers are made for application as special gaskets and tank liners. Production of butylenes for all purposes in 1953 was 720,000 short tons.

BUTADIENE

Butadiene is the major component of synthetic rubber (GR–S), as previously indicated, and 1953 production was equivalent to 575,000 tons. The level of synthetic rubber production in the United States will hold or rise slowly in the next few years, and the product can be expected to improve in quality from year to year. It now is vastly superior to the first material made in World War II.

FORMALDEHYDE

Methyl alcohol and light hydrocarbons are the starting materials for formaldehyde production processes. More and more methyl alcohol is derived from petroleum sources. Formaldehyde and phenol were combined to produce the first large-scale durable plastic Bakelite. This type of product is known as a phenolic resin. Later came the combination of urea and formaldehyde to yield the urea-formaldehyde resins, to be followed by the melamine-formaldehyde resins. All three types are produced by the same basic process and constitute the major part of the molding plastics. Formaldehyde production in 1953 was over 220,000 tons.

ACETALDEHYDE

Acetaldehyde is the principal product of the conversion of ethyl alcohol. Acetaldehyde also is produced by partial oxidation of hydrocarbons, such as propane and butane. Methyl alcohol, when made in such a process, is accompanied by large quantities of acetaldehyde. Acetaldehyde primarily is an intermediate for the production of other products and finds its greatest utilization in textiles and acetate derivatives. About 400,000 tons was produced in 1953, and a slow growth in demand can be expected.

CARBON TETRACHLORIDE

Carbon tetrachloride was first produced from petroleum in 1943. A large percentage is produced from carbon disulfide.

Possibly 40 percent of the total production of 130,000 tons of carbon tetrachloride in 1953 came from petroleum sources. A gradual growth in demand is in prospect. The major use is for the manufacture of refrigerants of the Freon type. It is a versatile solvent and is used widely for degreasing metals and as a dry-cleaning solvent. It is an important grain fumigant and still is used in large volume in fire extinguishers.

AROMATIC HYDROCARBONS

Aromatic hydrocarbons have been known and used industrially for many years. They formed the basis for the coal-tar-dye industry years ago, and coal tar was the only source until World War I. At that time the supply of toluene for making TNT was inadequate, and petroleum was used as a source of supply to make up the deficit. After World War I coal-tar toluene was adequate for all demands and was cheaper than toluene from petroleum. World War II again made it necessary to produce more toluene from petroleum, both for TNT and aviation gasoline. Toward the end of World War II toluene from petroleum supplied 85 percent of the demand for TNT production. Since the war demand for aromatics for chemical manufacture has increased, and benzene, toluene, orthoxylene, paraxylene, xylene mixtures, cumene, ethyl benzene, naphthalene, and other aromatic mixtures are being supplied from petroleum. All the aromatics are used in large quantities as solvents.

BENZENE

Petroleum is a newcomer to the benzene-supply picture, with an experimental commercial production in 1947. In 1950, 2 oil companies began regular production, and by 1953 the oil industry supplied 24 percent of the total production of 950,000 short tons. By 1955 it is estimated that petroleum will supply 40 percent of a 1,200,000-ton demand. Petroleum distillates in the gasoline range can be reformed to produce high yields of benzene and higher boiling aromatics, utilizing the new platinum catalysts. Better methods of aromatic extraction and purification have made petroleum benzene almost competitive with coal-tar benzene. Benzene is used directly as a solvent in many applications and as an intermediate for styrene, dyes, other intermediates, and phenol.

TOLUENE

Petrochemical toluene already has outstripped coal-byproduct toluene production, and further increases for a number of years probably will come from petroleum raw materials. In 1953 petrochemical toluene represented 75 percent of the total of 550,000 tons produced. Toluene is used directly as a solvent and as a component of aviation gasolines. It is the principal ingredient in the manufacture of TNT explosives.

XYLENES

The petrochemical industry furnished a still larger share of the xylene production. This share was 83 percent of the total 1953 production of 450,000 tons. Orthoxylene is produced primarily as an intermediate of phthalic anhydride, an essential component of paints, resin plasticizers, and insecticides. The larger market for paraxylene is in terephthalic acid, a starting material for synthetic fibers, such as Dacron.

STYRENE

As indicated previously, styrene is made by dehydrogenation of ethyl benzene, manufactured from benzene and ethylene. Production in 1953 was about 400,000 tons, with about a third of this quantity required for GR–S synthetic rubber production. Another major use of styrene is for polystyrene production. Polystyrene is a thermoplastic resin (plastic) ideally suited for injection molding to give rigid articles such as toys, wall tile, and electrical insulation material. Its low price and the attractiveness of the finished products (transparent and colored) have made polystyrene a preferred product.

OTHER PETROCHEMICALS

It has been estimated that as many as 100,000 petrochemicals could be made available today at a price. The seven largest end uses now are for rubber, in the automotive and aviation industries, synthetic fibers, plastics, agriculture, surface coatings, and explosives. Over 45 petrochemicals have been developed since 1945 to the point where they are available in tank-car quantities.

Some of the petrochemicals not discussed above include cumene, naphthalene, acetylene, isopentane, isooctane, cyclohexane, and normal heptane among the pure hydrocarbons. Nylon is made in part from benzene, cyclohexane, and butadiene. A substituted propanediol is an excellent new insecticide. Diisobutyl carbinol and isooctyl alcohol are two new ones in the alcohol field. Nonyl phenol is an intermediate for synthetic detergents. Acrolein is the important component of the amino acid methionine. Diisobutyl ketone is a high-boiling lacquer solvent.

Chlordane, aldrin, dieldrin, sodium dichlorphenoxethyl sulfate (crag), and a chlorinated isopropyl carbonate are important new agricultural chemicals for weed and insect killing.

This discussion would be incomplete without some mention of sulfur from petroleum and natural gas. Several years ago a severe shortage of sulfur developed, and with characteristic zeal the petroleum industry began sulfur recovery in quantity. Some sulfur has been produced by the industry for a long time. By 1953 some 850,000 short tons of sulfur was recovered from natural gases and refinery gases in all parts of the country. Additional plants for such recovery are being constructed.

Everyone connected with petrochemicals is satisfied that demands will increase and new products will be developed at a rapid rate. Some products will be overproduced, and others will grow in demand faster than the supply is increased. These developments will be an important factor in the national economy, and petrochemicals will be items of concern to the petroleum industry from a price standpoint, even though it will be a long, long time before petrochemicals volumewise will represent even 10 percent of the total petroleum and natural-gas production.

In addition to the references previously cited in this section, others are listed in the appended bibliography.

BRIEF HISTORY OF PETROLEUM AND THE PETROLEUM INDUSTRY

The early history of the use of petroleum by mankind is buried in the depths of antiquity. David White, formerly of the Federal Geological Survey, has supplied an admirable summary of the early history of petroleum (*76*). Many statements in this historical sketch have been excerpted from his paper, and the following is quoted directly:

> The part played by petroleum in advancing civilization began some thousands of years, at least, before the Christian era. Accounts of the occurrence of and uses of oil or its solid residues appear in many ancient writings on natural history, travel, religion, and philosophy * * *. Most interesting compilations of these early accounts have been made at different times by various authors, and one of the most scholarly and comprehensive is that prepared by S. F. Peckham (*53*) and his collaborators * * *.
>
> In exhuming the buried cities of Ur, which antedated Babylon, archeologists have found that asphalt was used as far back as the structure below the "diluvial" layer, probably easily as early as 4000 B. C.

In the ancient world pitch or native asphalt was used as a binder for pavements and for waterproofing cisterns and grain silos and by the Egyptians in the process of mummification (about 3500 B. C.).

The oil and gas occurrences in the Caspian region, notably those at Baku and on the Apsheron Peninsula, were known to Marco Polo.

Petroleum was familiar to the aborigines of the Western Hemisphere long before Christopher Columbus; and bitumen was used by the Toltecs, who preceded the Aztec race in Mexico, to set mosaics.

It was found in excavating a shell mound at Emeryville, Calif., that asphalt had been used for fastening beads to various articles about 1,000 years ago.

PRODUCTION

Although oil now is extracted from the ground at the rate of billions of barrels a year and the annual production of natural gas is measured in trillions of cubic feet, the early supply was small. The first sources of crude oil were natural seepages and "oil springs." Later, wells were dug with hand tools, and the oil was lifted to the surface in buckets. Still later, shafts not unlike those used in coal mining were sunk, and the oil was brought up in containers on the backs of workers. Production of petroleum in great quantities from drilled wells is a comparatively recent development in the United States.

In the United States it is customary to date the beginning of the petroleum industry from the completion of a well drilled for petroleum at Titusville, Pa., on August 27, 1859, by E. L. Drake; however, Bowles (*11*) states that the petroleum industry really started in the Kingdom of Rumania in 1857, and this little country produced about 165 million barrels of crude petroleum up to the beginning of 1921 (1.4 billion barrels by the end of 1954). Marketed production of 1,977 barrels of crude petroleum in Rumania is reported for 1857 and 4,349 barrels in 1859, when the marketed production of crude petroleum in the United States was only 2,000 barrels. In 1860 the United States was well in the lead, with a marketed production of 500,000 barrels, and has produced the major portion of all crude petroleum marketed since that time. Canada became a producer of crude petroleum in 1862 with 11,775 barrels and Russia in 1863 with 40,816 barrels (*29*).

Thompson (*66*) states that before 1871 all Russian petroleum was obtained from hand-dug surface pits, the deepest of which rarely exceeded 50 feet, and that the first oil well was bored in the Balakhany Plateau in 1871. Bowles (*11*) notes that Russia led the world from 1898 to 1901, inclusive, with a production averaging 72 million barrels of crude petroleum a year.

The technique and some of the equipment used by E. L. Drake were developed in drilling for salt brines and apparently were devised largely by David and Joseph Ruffner, who in 1808 completed a well through rock which produced a concentrated brine. Drake has been credited with being one of the first to use casing in a drilled well.

Apparently the first flowing oil well in the United States was drilled for brine in 1818 or 1819 by David Beatty on the Big South Fork of the Cumberland River in Wayne County, Ky.

The product, called "The Devil's Tar" by Beatty, was allowed to flow into the Cumberland River and covered its surface for 35 miles and later caught fire, with disastrous results to adjoining property. The well was abandoned and almost forgotten for more than 30 years (*11*, *76*).

The first oil lease on record is said to have been made July 4, 1853, between J. D. Angier of Cherrytree Township, Venango County, Pa., and Brewer, Norton & Co. (*8*).

Statistics compiled by the Federal Geological Survey (*29*) indicate that crude petroleum was produced in the United States in Pennsylvania, New York, Ohio, West Virginia, and Kentucky from 1860 to 1875. Production was begun in California and Tennessee in 1876 and Colorado in 1887. In 1887 the recorded production of petroleum in the 8 producing States was 28,283,000 barrels, of which 20,281,000 barrels was produced in Pennsylvania.

REFINING

The first petroleum company was the Pennsylvania Rock Oil Co., incorporated December 30, 1854, in New York by G. H. Bissell and J. G. Eveleth, with a capital stock of $250,000 (*8*). This company was the forerunner of the Seneca Oil Co that hired E. L. Drake.

Apparently the first plant for distilling petroleum was built about 1855–57 by George S. Gilbert near Ventura Mission in California (*62*). Before the Drake well was drilled, illuminating oil and a few other products were distilled in the Eastern States from coal (hence "coal oil"), from albertite imported from Canada and from oil shale and cannel coal imported from England. After the Drake well and other oil wells began to produce petroleum, some of these coal and oil-shale plants were converted to distill and refine petroleum.

The first petroleum refinery in the Oil Creek region, about a mile from the Drake well (*5*), was built by William Barnsdall and W. H. Abbott in 1860.

TRANSPORTATION

The first oil pipeline to transport crude petroleum satisfactorily was completed in August 1865 by Samuel (or M. E.) Van Syckel of Titusville, Pa., and associates. The line extended from the Miller farm to Pithole City, was about 5 miles long, and had 2 pump stations. The sections of 2-inch pipe were joined by carefully fitted screw sockets, and the line carried 80 barrels of oil a day. The line demonstrated the practicability of transporting petroleum in this way (*5*, *8*, *25*).

In 1862 the Atlantic & Great Western Railroad was extended into the oil region of Pennsylvania. In 1866 the Allegheny Valley Railroad was opened from Oil City to Pittsburgh, and a number of narrow-gage lines were constructed as feeders. At first, wood barrels of crude petroleum were loaded on flat cars, but the leakage was so great that wood tank cars were built by Amos Densmore in 1865 comprised of 2 wood tanks, each holding about 2,000 gallons, on an ordinary platform car. About 1870 the ancestor of the present tank car appeared—the tank being one unit, horizontal and cylindrical, and fitted with a dome that allowed the oil to expand without damage to the tank. The capacity initially was about 90 barrels but sometime later was increased to 100 barrels (*5*, *8*).

The first oil tanker was the *Charles* of Antwerp, Belgium, which plied between the United States and Europe from 1869 to 1872. The ship had 59 separate iron tanks in the hold, with a total capacity of 714 tons (almost 5,000 barrels of crude oil). The first successful tank steamer is said to have been built by Wilhelm Riedeman of the German subsidiary of the Standard Oil Co. in 1880 (*61*). Towle (*25*) states that the first steamer that successfully carried petroleum in bulk was the *Glückauf*, which was built in England in 1886. Thompson (*66*) states that in 1879 the first tank steamer was constructed by Nobel Bros., the pioneering petroleum refiners in Russia.

Thompson states that astatki (residual fuel oil) was gradually introduced into the steamers on the Caspian Sea by Nobel Bros. It was discovered that astatki could be "pulverized" by means of a jet of steam and converted into a highly combustible state. In 1874 the Government of Russia adopted astatki for all the vessels of the Caspian fleet, and its use extended to the railways (*66*).

The foregoing brief sketch deals with the beginnings of the utilization of petroleum until about 1880. A much longer discourse would be needed merely to mention in brief outline other important first steps of the industry. Additional information is contained in the references cited in this account and in the primary source material mentioned in the references.

ORGANIZATION OF PETROLEUM INDUSTRY

Since the Drake discovery nearly a century ago, the petroleum industry of the United States has grown to be a complex and highly competitive business composed of many thousands of separate units, with independent corporate structures of almost every known form. The business activities of the petroleum industry can be divided into four functional groups which are widely recognized as (1) exploration, drilling, and production of crude petroleum, other hydrocarbon liquids, and natural gas; (2) transportation of crude petroleum and other hydrocarbon liquids, refined products, and natural gas; (3) refining crude petroleum and other hydrocarbon liquids, and processing natural gas into marketable products; and (4) distributing and marketing products to ultimate consumers or to processors who use petroleum products, such as asphalt, to make finished products, such as roofing (*5*).

The petroleum industry as a whole has business organizations that operate in all these four fields (frequently called "integrated" companies) and a much larger number of individuals and organizations whose business is restricted to only 1 or 2 of the 4 major activities or functions.

The Bureau of Mines estimates that over 18,000 distinct entities are engaged in exploration, drilling, and production of crude petroleum, other hydrocarbon liquids, and natural gas. These 18,000 entities include individual producers, partnerships, separate corporations, and organizations affiliated with wholly or partly integrated companies.

Recently the Oil Industry Information Committee of the American Petroleum Institute announced that a 3-year study revealed the existence of over 200,000 oil businesses in the United States. There are 44,602 engaged in production, refining, transportation, and wholesale distribution of petroleum and petroleum products. There are 188,253 service stations (the same number reported in the United States Department of Commerce survey), of which 177,923 are single-unit establishments and are classified as separate businesses. The various segments of the industry are: Producers who are employers and pay social security taxes, 12,010; pipeline companies, 54; tankship companies, 42; bargeline companies, 174; tank-car companies, 61; for-hire contract carriers (tank trucks), 1,063; petroleum refiners, 270; manufacturers of lubricants and greases not made in refineries, 243; petroleum bulk stations and terminals, 14,053; fuel-oil dealers, 11,132; and liquefied-petroleum (LP-)-gas dealers, 5,500. The last two figures are for dealers not included in the bulk-station and terminal classification. It is not possible to determine the exact number of dealers engaged in the fuel-oil and LP-gas businesses, but it is believed there are over 20,000 companies handling heating oil and 20,000 to 35,000 companies distributing LP-gas to ultimate consumers.

The facts presented in the preceding paragraphs illustrate the diversified ownership and competitive character of the petroleum industry of the United States. Competition is keen and has made for an industry capable of meeting the heavy demands of an expanding economy and security needs. Thirty-one of the larger units produce slightly less than 60 percent (*42*) of the Nation's crude oil. The largest unit produced about 386,509 barrels of crude oil a day in 1954, which at that time was only 6.09 percent of the daily production of the United States. The smallest producer of the group of 32 companies had a daily production of about 8,000 barrels or 0.13 percent of the domestic production. Two-fifths or over 2,400,000 barrels of crude oil a day was produced by over 18,000 small operators. Their production ranged from 8,000 barrels to less than 1 barrel a day and averaged less than 135 barrels a day per small operator. Refinery ownership is not as diversified as the production segment of the industry, but no single company dominates even this highly specialized manufacturing operation; in fact, the largest single company refines only 8.4 percent of the industry's throughput.

Corporate integration is of two kinds—horizontal and vertical. There is not much horizontal integration in the petroleum industry of the United States, in that companies will engage in an activity or activities throughout the United States, although a few large corporations have producing organizations from the Appalachian region to the Pacific coast, and the products of a few companies are sold in all 48 States. Some petroleum companies are almost completely integrated vertically in the sense that they engage in all activities connected with the petroleum industry, from financing their fund requirements to selling their products at retail. However, very few companies do all their own work. A company may have exploration specialists and yet employ independent organizations for exploration. Similarly, a company may drill wells with its own equipment and hired personnel and at the same time have other wells drilled by contractors who furnish all equipment, personnel, and supplies. These variations exist throughout the whole range of activities that comprise the petroleum industry.

ORGANIZATION OF NATURAL-GAS INDUSTRY

The natural-gas industry is not organized in the sense of a monopoly, but certain parts of the industry's activities come under public utility regulations. Virtually all forms of corporate organization are found within the industry. Commonly the natural-gas industry is divided into four general divisions (*41*): (1) Production, including exploration and development of producing properties; (2) extraction and manufacture of byproducts, such as natural gasoline, liquefied petroleum gas, and petrochemicals; (3) transmission to markets, including gathering lines between wells and main pipelines; and (4) distribution in market areas. The transmission and distribution divisions form an integral part of the Nation's public utilities system, which includes electric power and water services.

These phases of the natural-gas industry are subject to the regulatory functions of Government (National, State, and local) for the purpose of securing equitable conditions for the producers, transporters, distributors, and consumers. Units of the industry engaged in producing, extracting, and manufacturing byproducts are numerous and may operate as individuals, partnerships, or separate corporations. On the other hand, some units in these divisions of the industry are parts of companies that transmit and distribute natural gas, although complete integration is relatively rare. Companies having such units function as public utilities and are organized accordingly.

The Bureau of Mines estimates that 743 natural-gas producers supply approximately 80 percent of the natural gas produced in the United States. The Bureau of Old Age and Survivors Insurance estimates that considerably more than 6,000 producers of crude petroleum (who are employers paying social security taxes) also are producers of associated natural gas. About 200 companies operate a total of 521 natural-gasoline plants; moreover 16 firms operate 52 carbon-black plants. The natural-gas industry has 80 pipeline companies and 850 distributors to ultimate consumers.

The production division of the natural-gas industry is concerned with the discovery of natural-gas supplies, the subsequent development of gas-bearing properties, operation of the gas wells, field compression of gas when the reservoir pressure is not high enough to feed gas into the gathering lines against their operating pressures, and economic depletion of the gas reserve. Because natural gas is produced in conjunction with crude oil and also is often discovered as a result of the search for oil, the production of natural gas and the extraction of hydrocarbon liquids from it may be carried on by oil companies. For organizations of large enough size, the production division is usually divided into the geological, geophysical, drilling, and production departments. Some companies also have an engineering department within the production division. The geological, geophysical, and drilling departments function in the same manner in both the petroleum and natural-gas industries. The production departments of the 2 industries operate similarly, the differences resulting from the physical characteristics of the 2 commodities. It is a general practice in both industries to contract for the performance of geophysical prospecting and drilling of wells although some companies are equipped and do their own work in these specialized fields. Moreover, some of the smaller companies hire geological and engineering consultants on a part-time basis.

The extraction and manufacture of byproducts comprises such functions as the operation of gasoline plants that produce natural gasoline and liquefied petroleum gas, carbon-black plants, petrochemical plants producing ammonia and a large number of organic chemicals, synthetic rubber plants, sulfur-extraction plants, and helium-extraction plants. Natural-gasoline plants usually are operated by either the production or transmission divisions of the natural-gas industry. In a few instances separate and independent companies operate the gasoline plants; however, some of them are subsidiaries of oil companies, and others operate carbon-black plants as their principal activity but also produce petroleum and natural gas. In general, separate companies operate the carbon-black plants. Petrochemical plants sometimes are owned and operated by large chemical-manufacturing concerns, by special departments of oil companies, and occasionally by corporations organized for this specialized purpose. Synthetic rubber plants are generally operated by rubber companies and/or petroleum companies. Some are joint operations of both the rubber and oil industries. In a few regions, the natural gas contains such large percentages of hydrogen sulfide that it becomes economically feasible to extract the hydrogen sulfide and produce sulfur therefrom for its own value. In a number of areas it is necessary to extract the hydrogen sulfide to avoid corrosion problems. These operations are conducted by oil or gas companies and also by sulfur companies. Because of the cost of plants and size of operations generally conducted in the division of extraction and manufacture of byproducts, the organizations are usually large companies, but in a few instances individuals and partnerships are operating. When several companies have holdings in a single or adjacent producing fields, it is not uncommon for them to construct the extractive facilities on a cooperative basis.

The transmission division of the natural-gas industry generally is operated by companies or corporations organized for this specialized transportation function. However, some companies operate in all divisions, producing, processing to extract products, transmission, and distribution. More often these companies operate in only the first three divisions and sell natural gas to the local distribution systems which are public utility companies. Transmission companies are organized much like railroads, with engineering, maintenance, operations (including compressor plants), measurement, and communication departments.

The distribution division is concerned with acquisition of franchises, operation of municipal systems, securing fair rates, and meeting the ultimate consumers' demands. It is organized to lay and maintain city mains, make connections for consumers, read consumers' meters, bill for gas used, and promote consumption. These are the usual activities of a utility business.

In the natural-gas industry there is no horizontal integration in the sense that a unit operates throughout the natural-gas producing and marketing areas of the United States. However, inasmuch as some companies operate long pipelines—for example, from Texas to New England—and provide gas to various localities along the route, there is a degree of such integration. A few natural-gas companies are completely integrated vertically from production to marketing, but even these companies purchase part of their supplies of gas; however, very few companies do all their own work. Almost all units of the industry engage contractors and consultants to do specialized jobs.

INTERRELATIONS BETWEEN PETROLEUM AND NATURAL-GAS INDUSTRIES

The organizations of some of the divisions of the petroleum and natural-gas industries engage in the same activities, and in other instances the two industries compete for the same market. For example, in the production division oil operators also produce natural gas, which is associated with the crude petroleum. This gas is oftentimes sold to natural-gas companies. Furthermore, oil operators sometimes own "dry" gas wells (wells that produce natural gas without petroleum), and gas from these wells is sold also to natural-gas companies. In other instances natural-gas producers own oil wells and sell the oil production to oil-pipeline companies. Most natural-gasoline plants are operated by the producing departments of oil companies; but some are also operated by the producing or pipeline departments of gas companies, and some are in the hands of companies engaged in this one activity. The geological and geophysical departments of the 2 industries operate along similar lines, and sometimes the 2 industries compete for leases on the same tracts of land.

The pipeline companies of the two industries are organized to handle transportation problems. Interstate oil pipelines are common carriers by law and therefore are subject to Interstate Commerce Commission regulations. Natural-gas pipelines engaged in interstate business are subject to regulations by the Federal Power Commission. Gas pipelines are obligated by contract to supply certain definite volumes of gas to their customers, including the local distribution utilities. Some commitments are required before the Federal Power Commission can issue a certificate of necessity to build the pipeline. These commitments furthermore prevent the natural-gas pipelines from being common carriers like railroads and oil pipelines.

The refining division of the petroleum industry has no related division of the natural-gas industry. However, refineries operate gasoline plants to extract gasoline from still gases. Thus the oil production and refinery divisions and the natural-gas production and pipeline divisions all operate gasoline plants which make various grades of gasoline and liquefied petroleum gases (LP-gases). At refineries the latter products are called liquefied refinery gases (LR-gases), although they correspond to the products extracted from natural gas.

From an organization standpoint the marketing divisions of the two industries are entirely separated. The marketing division of the petroleum industry includes the local wholesale or bulk plants, the well-known filling stations, and fuel-oil dealers. The marketing division of the natural-gas industry is the local gas-utility company. Although their organizations are entirely different, they often are good customers of each other and at other times compete keenly for the same business, as, for example, fuel oil and gas for heating and power generation.

SIZE OF PETROLEUM INDUSTRY

The size of the petroleum industry, no matter what units or what comparisons are used to measure it, is very difficult to comprehend. The 2.6 billion barrels of crude oil and natural gasoline produced in this country during 1954 would fill 12,400,000 average tank cars and form a train 94,300 miles long (not allowing for locomotives, tenders, or cabooses). This train would be long enough to circle the earth at the equator almost three times.

Fortunately, most of this oil does not have to move by tank car, for there are only about 122,000 such cars in petroleum service (*6*). Although they make a valuable contribution to the movement of both crude oil and refined products, only a relatively small share of the total oil transported moves by rail. Almost all oil moves at least once on its journey from well to consumer through a pipeline. The oil industry has provided a network of over 170,000 miles of pipe (*24*) reaching into virtually every part of the country. Through the network of trunk pipelines there moved during 1954, 1,894 million barrels of crude oil and 664 million barrels of refined products (*21*). This represented 82 percent of all the crude oil produced and 33 percent of the refinery output of products suited to pipeline movement; the combined tonnage equaled almost 30 percent of freight of all types moved by all the country's railroads.

Another large element in the oil-transportation picture is movement by water. The 430 ocean-going tankers operated chiefly by the oil companies on December 31, 1954, amounted to 13.2 percent of the total number of American flag vessels, while their deadweight tonnage of 6,838,000 was 19.5 percent of the total under the American flag (*43*). In addition, United States oil companies owned and operated a large although indeterminate number of vessels under foreign registry. These ocean-going tankers were supplemented by some 2,100 tank barges on inland and coastal waters, with a combined capacity of about 18.4 million barrels (*6*).

To supply the huge amount of crude oil used in this country, the industry operates approximately 500,000 wells. The output of these wells ranges from a fraction of a barrel up to several hundred barrels a day; some wells can produce several thousand barrels of oil daily. The productive capacity of each of these half million wells is constantly declining, although the rate of decrease in the stripper wells that comprise almost 90 percent of the total number (*6*) is relatively slow; however, in order to maintain the country's ability to produce oil 53,930 new wells were drilled during 1954, and of these 29,773 were oil producers. The total footage of these wells is 218.9 million feet or over 41,000 miles (*31a*).

Consumption of crude oil in the United States during 1954 was somewhat larger than production, the total being 2.57 billion barrels. This is enough to provide every man, woman, and child with 675 gallons, which, in terms of gasoline, will drive an automobile 10,000 miles. This rate of consumption is all the more remarkable when contrasted with the rest of the world, where annual consumption averages about 65 gallons per capita, less than 10 percent of the rate in this country (*9*).

To produce this huge amount of crude oil, move it to refineries, transform it into usable products, and distribute them to consumers requires a tremendous amount of property, plant, and equipment that represents a gross investment of some $43,200 million. In terms of capital investment, the petroleum industry ranks fourth and is exceeded in size only by agriculture, the railroads, and the combined divisions of the public utility field. Almost 1.7 million people are required to operate the industry (*57*).

SIZE OF NATURAL-GAS INDUSTRY

Natural gas is the most rapidly growing of all the energy sources in the United States, and the 8.7 trillion cubic feet of marketed production during 1954 almost doubled the output only 8 years earlier. During the same period the share of total energy supplied by natural gas rose from 13.8 percent to 25.0. Natural gas is collected by a system of gathering pipelines totaling 41,360 miles, transmitted to almost all parts of the country by 125,360 miles of transmission lines, and distributed to consumers through 227,170 miles of local distribution lines—in all, 393,890 miles of pipelines.

Although the natural-gas industry does not involve any complex manufacturing equipment corresponding to that at petroleum refineries, the extensive collection, transmission, and distribution facilities, combined with the production equipment, have an asset value of $11.8 billion (*4*). This extensive plant enabled the natural-gas industry to serve almost 20 million customers in 1953, including over 18 million residential consumers. This indicates that well over 35 percent of all dwelling units in the United States are served with natural gas, while 12.3 million or 26 percent are heated with gas; almost all gas for heating is natural gas.

ENERGY SUPPLIED BY PETROLEUM AND NATURAL GAS

The importance of petroleum and natural gas in our country's energy supply is emphasized by the fact that during 1954 almost 67 percent of the total was derived from these sources. Domestic crude petroleum provided 35.9 percent of the total, natural gas 25.0 percent, and bituminous coal 27.5 percent, while anthracite, waterpower, and petroleum imports supplied relatively small amounts. The growth of petroleum and natural gas relative to other sources of energy is clearly shown in table 3.

GEOGRAPHICAL DISTRIBUTION OF PETROLEUM INDUSTRY

Oil fields are distributed over the entire United States. In the Appalachian area, New York, Pennsylvania, West Virginia, Ohio, Ken-

TABLE 3.—*Percentage of total energy derived from various sources*

Year	Crude petroleum produced and imported	Natural gas	Total petroleum and natural gas	Coal: Bituminous and anthracite	Waterpower
1900	4.7	3.2	7.9	88.9	3.2
1910	7.9	3.6	11.5	85.0	3.5
1920	14.5	3.9	18.4	78.2	3.4
1930	24.8	9.3	34.1	62.5	3.4
1940	32.1	11.3	43.4	53.1	3.5
1950	[1] 36.6	18.6	55.2	40.5	4.3
1951	[1] 37.6	20.3	57.9	38.2	3.9
1952	[1] 39.5	22.2	61.7	34.2	4.1
1953	[1] 40.5	23.0	63.5	32.6	3.9
1954	[1] 41.8	25.0	66.8	29.3	3.9

[1] Include imports of distillate and residual fuel oils.

tucky, and Tennessee have producing oilfields. Some of the largest petroleum deposits exist in the midcontinent area in Indiana, Illinois, Arkansas, Louisiana, Texas, New Mexico, Oklahoma, and Kansas. In the Rocky Mountain area, in Montana, Wyoming, and Colorado, increasing quantities of petroleum and natural gas are produced each year. California is the only West Coast State producing petroleum and natural gas, yet it provides a substantial amount of our national requirements.

Common usage includes the terms "Appalachian," "Midcontinent," "Gulf coast," "Rocky Mountain," and "West coast" as broad subdivisions of production; however, Ver Wiebe (*73*) provides a classification of provinces that restricts these somewhat general terms (table 4).

Although about 55 percent of world crude oil is refined in the United States, the largest refineries are outside its boundaries. Most large refineries in this country are on the gulf coast of Louisiana and Texas, in the Philadelphia Harbor area, in the Chicago area, and in the vicinity of Los Angeles and San Francisco. Smaller plants in large refinery centers are in the Western Appalachian, Rocky Mountain, and midcontinent areas.

The refinery industry of the United States is distributed widely over the entire country and is characterized by a large number of plants that vary in capacity from a few hundred to nearly 300,000 barrels per day.

For statistical and economic analysis of its operations the petroleum industry is divided

TABLE 4.—*Petroliferous provinces*

Province	Districts	States involved
Appalachian geosyncline		New York, Pennsylvania, West Virginia, eastern Ohio, eastern Kentucky.
Cincinnati arch	Lima-Indiana	Western Ohio, eastern Indiana.
	Cumberland Saddle	South-central Kentucky, Tennessee.
	Alabama-Mississippi	Alabama, Mississippi.
Eastern interior coal basin		Northwestern Kentucky, Illinois, Indiana (southwestern).
Michigan Basin		Michigan.
Western interior coal basin		Kansas, northern Oklahoma.
Ouachita-Amarillo Mountains	Arbuckle-Wichita	Southern Oklahoma.
	Red River Uplift	North-central Texas.
	Amarillo Mountain	Texas Panhandle.
Bend arch		North-central Texas.
Gulf embayment	Balcones fault	Eastern Texas.
	Reynosa escarpment	Southern Texas.
	Sabine and Ouachita uplifts	Northern Louisiana, southern Arkansas.
	Salt dome	Southern Texas, southern Louisiana.
West Texas basin		West Texas, eastern New Mexico.
Rocky Mountain geosyncline [1]	Big Horn Basin	Wyoming, Montana.
	Wind River Basin	Wyoming.
	Green River Basin	Wyoming, northwestern Colorado.
	Laramie Basin	Wyoming.
	Powder River Basin	Do.
	Sweetgrass arch	Montana.
	Big Snowy anticlinorium	Do.
	Julesburg Basin	Colorado.
	Uinta Basin	Western Colorado.
	San Juan Basin	Northwestern New Mexico, Utah.
Pacific geosyncline	San Joaquin Valley	California.
	Santa Maria	Do.
	Ventura	Do.
	Los Angeles Basin	Do.

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[1] Because this table was published in 1930, the Williston Basin should be added to the Rocky Mountain geosyncline province, covering parts of Montana, North and South Dakota, and Canada.

into geographical areas known as districts, regions, or groups. Production of crude petroleum is reported by the Bureau of Mines from 8 districts, including the Midcontinent, which has 4 subdivisions. The manufacture of refinery products is reported from 10 districts. The manufacture of aviation gasoline is reported from 5 districts. For distillate fuel oil, production is reported from 10 districts and sales from 5 Petroleum Administration for War (PAW) districts. Bureau of Mines semiannual motor-gasoline surveys divide the 48 States and the District of Columbia among 17 districts. None of these districts and regions are coextensive but have been established to group States and areas with similar producing, refining, transportation, or other characteristics. It often is difficult to establish a supply-demand balance for a given area because of the dissimilarity of the groupings, but the existing system serves a definite purpose and is very generally accepted.

The economics and technology of the industry are affected by transportation rating points, such as tankship rates from Houston, Tex., to New York Harbor, rail freight rates such as those of group 3 in the Oklahoma-Kansas area, and pipeline tariffs for crude petroleum or pipeline mixtures and finished products.

EXTENT OF PETROLEUM-INDUSTRY ACTIVITIES

EXPLORATION AND DRILLING

Virtually each one of the 48 States has been prospected to some extent for petroleum and natural gas, and the search for new sources of supply is continuing. In 1954, 53,930 wells were drilled by the petroleum and natural-gas industry, of which 29,773 were oil wells, 3,977 were gas wells, 19,169 were "dry" and did not produce commercial quantities of either oil or gas, and 1,011 were service wells drilled to supply water, dispose of brine, or repressure fields. Texas led the States in 1954 with 18,865 wells, Oklahoma was second with 8,786, and Kansas was third with 4,722 productive and unproductive wells.

PRODUCTION

In 1954 commercial production of crude petroleum was reported to the Bureau of Mines from 29 States. The 3 principal States, in terms of volume of production, were Texas 42.4 percent, California 15.4 percent, and Louisiana 10.6 percent. The East Texas field remains the all-time champion, with a recorded production total to the end of 1954 of 3,149 million barrels, followed by the Midway-Sunset field of California with almost 800 million.

REFINING

The Bureau of Mines has reported that, as of January 1, 1954, 308 petroleum refineries were operating in 35 States, with a stated crude-oil throughput capacity of 7,782,103 barrels a day. Texas leads with 59 operating refineries, California has 43, and Oklahoma has 20. The 7 refineries that were located in New Jersey have the largest average crude-oil throughput capacity of any State, with an average for each refinery of 68,000 barrels a day. The largest refinery in the United States is that of Humble Oil & Refining Co. at Baytown, Tex., which has a stated crude-oil throughput capacity of 282,000 barrels a day. Operating refineries range downward to 160 barrels of crude-oil throughput capacity a day.

REFINED PRODUCTS

Petroleum refineries manufacture a wide variety of products, which are classified in Bureau of Mines statistics in categories and were manufactured in 1954 in the percentage yields in the following tabulation.

TABLE 5.—*Percentage yields of refined petroleum products in the United States, 1954*

Product:	*Yield, percent*
Gasoline	43.8
Kerosine	4.8
Distillate fuel oil	21.3
Residual fuel oil	16.4
Jet fuel	1.8
Lubricating oil	2.1
Wax	.2
Coke	1.0
Asphalt	2.9
Road oil	.3
Still gas	4.0
Other	1.7
Overage	—.3
Total	100.0

Modern petroleum-refining technology is so flexible that the percentage yield of any of the products listed in table 5, except gasoline and distillate and residual fuel oil, could be doubled at the expense of fuel-oil yield if a sustained demand should develop for these important minor (in percentage-yield) products. Meeting a sudden doubling of the demand might be difficult or impossible because the installed equipment may be operating too near its maximum capacity, and expensive facilities such as those for manufacturing lubricating oil or wax cannot be maintained economically in standby status.

GEOGRAPHICAL DISTRIBUTION OF NATURAL-GAS INDUSTRY

In connection with the allocation of scarce materials to the natural-gas industry, the country was divided into five Petroleum Administration for Defense (PAD) districts, which conform to the five districts for the petroleum industry. Roughly, district I comprises the east coast, district II the midcontinent, district III the gulf coast, district IV the Rocky Mountain States, and district V the west coast. Natural gas is produced in 29 States; Maryland produces natural gas only. In the other 28 States the producing branch of the natural-gas industry is often closely associated with the production of oil. Gas is produced with oil both as dissolved gas and from the gas cap. However, about two-thirds of all gas is found to exist independent of oil. The more important "dry" gas fields are in the Appalachian area, Michigan, the Western Kansas-Oklahoma Panhandle-Texas Panhandle area, and the North Louisiana-East Texas area. (See figs. 1 and 2.)

Part of the gas produced, especially casinghead gas, is used in the field in production operations and in natural-gasoline and cycling plants. The remainder of the gas (including that which has been processed through natural-gasoline plants), if a market can be found for it, is utilized predominantly in the areas adjacent to the producing field. In the heavily populated Appalachian and upper midwestern areas the local supply is not enough to serve any but its immediately surrounding markets. In the Southwest in the past large quantities of gas associated with oil have been flared for lack of market because it was not available in enough quantity for economical transportation to a market.

The development of materials and techniques that have made laying long-distance pipelines economically feasible has literally opened the entire country as a market for natural gas. From 1946 to 1954 the quantity of natural gas shipped interstate increased three and one-half times. Five States—Kansas, Louisiana, Mississippi, New Mexico, and Texas—were supplying 87 percent of the gas moving interstate in 1953.

In the following tabulation are shown the volumes of natural gas shipped interstate in 1946, 1951, and 1953 (table 6).

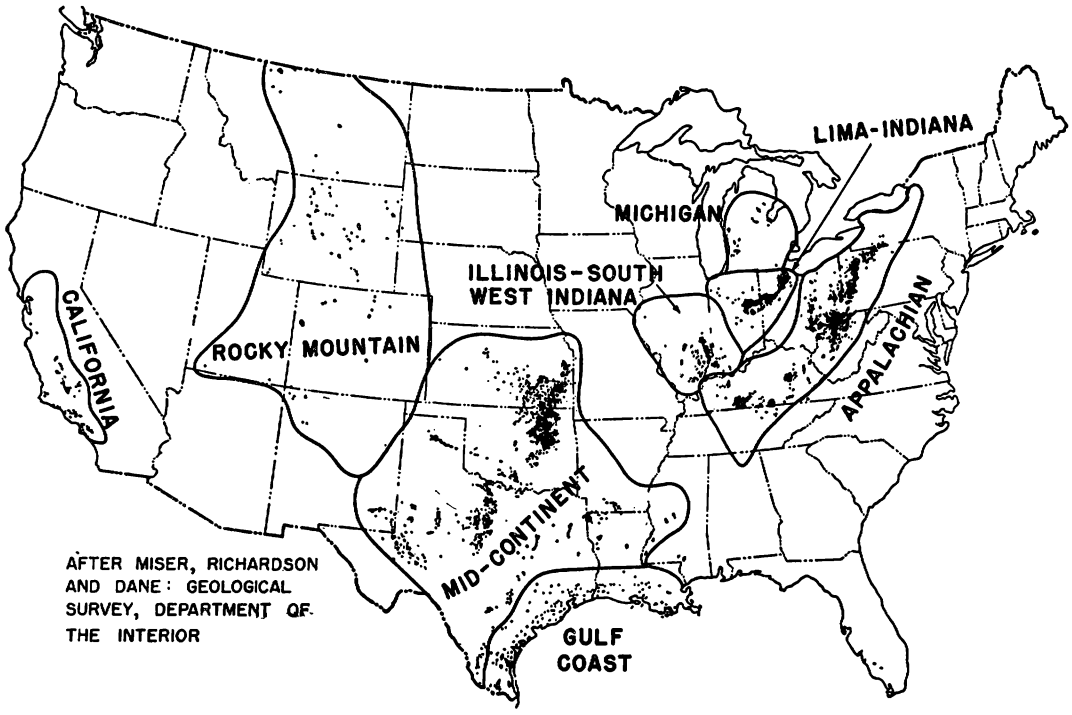
TABLE 6.—*Consumption of interstate gas*

[Million cubic feet]

	1946	1951	1953
Alabama	45, 445	114, 278	139, 890
Arizona	24, 198	63, 802	73, 709
Arkansas	42, 845	114, 854	142, 014
California		243, 749	357, 108
Colorado	38, 557	97, 435	108, 035
Connecticut			6, 412
Delaware		1, 892	2, 145
District of Columbia	7, 428	11, 904	13, 662
Florida	7, 059	13, 164	19, 670
Georgia	36, 679	96, 903	124, 793
Illinois	107, 118	252, 433	363, 609
Indiana	39, 097	72, 788	112, 826
Iowa	33, 163	82, 287	109, 197
Kansas	103, 805	140, 207	174, 520
Kentucky	10, 439	49, 823	80, 740
Louisiana	10, 442	50, 208	80, 974
Maryland	2, 830	22, 870	30, 434
Massachusetts		1, 934	19, 445
Michigan	48, 372	167, 404	195, 385
Minnesota	37, 624	85, 052	105, 462
Mississippi	34, 632	49, 142	86, 032
Missouri	74, 217	160, 252	179, 027
Montana	2, 910	10, 304	15, 909
Nebraska	33, 572	69, 713	77, 122
New Hampshire		3	921
New Jersey		32, 601	62, 850
New Mexico	2, 503	32, 571	52, 688
New York	28, 973	154, 274	209, 273
North Carolina		1, 759	6, 885
North Dakota	2, 519	3, 728	3, 186
Ohio	127, 729	377, 629	421, 241
Oklahoma	15, 179	16, 232	26, 863
Pennsylvania	86, 459	272, 356	312, 570
Rhode Island			694
South Carolina		917	9, 115
South Dakota	7, 521	11, 839	13, 963
Tennessee	24, 297	90, 271	109, 023
Texas	41, 480	61, 054	104, 347
Utah	11, 510	25, 952	28, 546
Virginia	2, 068	16, 083	29, 598
West Virginia	36, 073	113, 354	119, 973
Wisconsin	86	32, 783	39, 616
Wyoming	1, 397	2, 810	2, 999
Canada	200	3, 630	6, 014
Mexico	17, 475	20, 533	22, 308
Total	1,145,901	3,242,777	4,200,793

Annual interstate shipments increased by 3.1 trillion cubic feet during this period, roughly equivalent to the transmission capacity of 28 30-inch pipelines each with a capacity of 300 million cubic feet per day.

Consumption of this increased volume of interstate gas was spread over a greater number of states than was the production. All 33 States and the District of Columbia that used interstate gas in 1946 consumed a greater quantity in 1953. Nine States that were not receiving gas via interstate pipeline in 1946 were receiving it in 1953. Almost half of increased interstate shipments, however, went to six States. Five of these—Illinois, Michigan, Ohio, Pennsylvania, and New York—are in the



AFTER MISER, RICHARDSON AND DANE: GEOLOGICAL SURVEY, DEPARTMENT OF THE INTERIOR

FIGURE 1.—Oilfields of the United States.

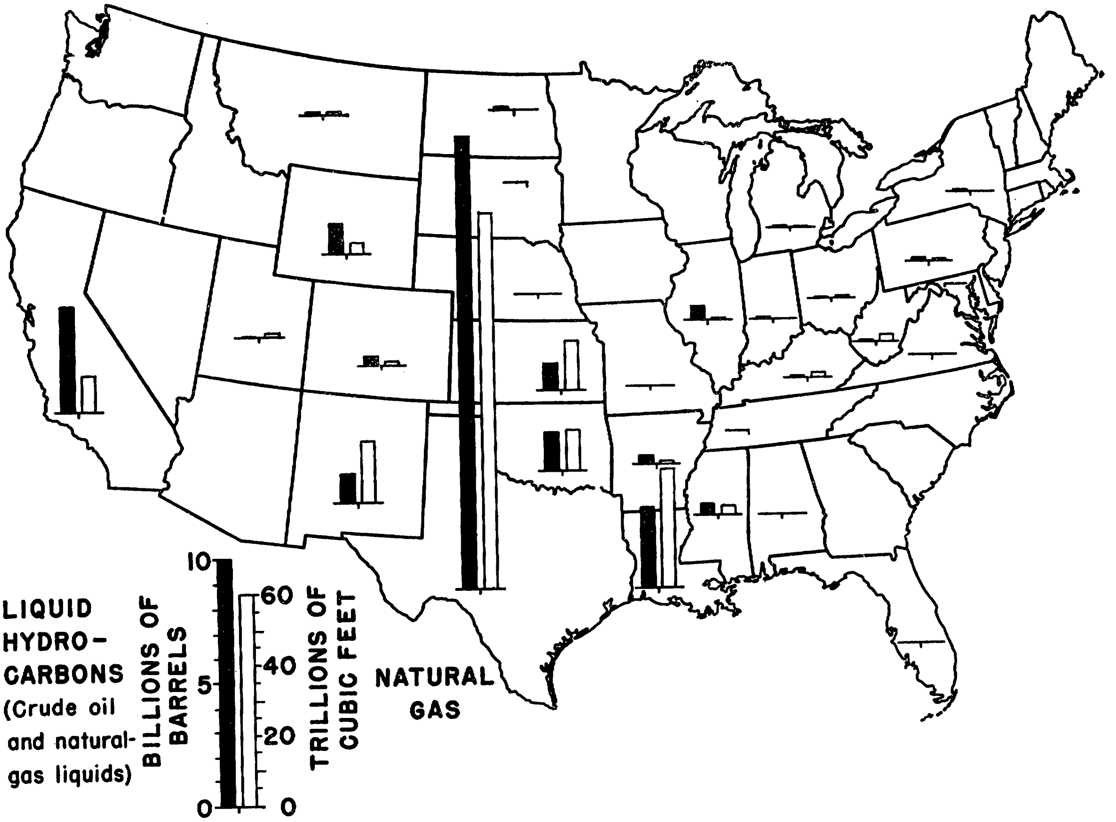


FIGURE 2.—Estimates of Proved Reserves of Liquid Hydrocarbons and Natural Gas, by States, as of December 31, 1953.

(Data from reports of API–AGA Committee on Reserves) same general area, and a number of new pipelines traverse two or more of them. The sixth State—California—was receiving no out-of-State gas in 1946, and in 1951 it depended upon out-of-State gas for over 40 percent of the gas consumed. Natural-gas service has been extended to all New England States except Maine and Vermont in the last few years. The Pacific Northwest remains the only general area to which the pipeline system has not been extended, but plans are being made for an additional pipeline coverage for this area.

Nearly 40 percent of the total consumption of gas in the United States in 1953 took place in the west south-central region (Arkansas, Louisiana, Oklahoma, and Texas). This is due to the large industrial uses in this region, including field uses and carbon-black manufacture. The consumption of natural gas for residential and commercial use is far less concentrated regionally. The west south-central region consumed only 13 percent of the natural gas used by residential and commercial consumers. The largest consumption in these categories takes place in the east north-central region (Illinois, Indiana, Michigan, Ohio, and Wisconsin). This region includes three of the States that showed the largest gain in consumption of interstate gas.

GEOLOGIC AND GEOPHYSICAL EXPLORATION

Paleontological classification of occurrences the world over indicates that petroleum is not confined to any one geologic period but occurs in rocks ranging from the Cambrian to the recent. Although petroleum is distributed through a considerable portion of the geologic column, it is found that the more prolific horizons may be classified within narrower limits. Existing evidence indicates that most commercial oil pools are confined to areas of unmetamorphosed rocks (rocks that have not been changed by pressure, heat, or water) of Ordovician or younger age and in "marine sediments" in shallow seas. However, most of the oil is found in geologic formations of the Tertiary, Cretaceous, Carboniferous, and Ordovician systems. Productive formations occur less frequently in the Jurassic, Devonian, and Silurian. Systems of comparatively small productivity include the Quaternary, Triassic, and Cambrian. However, the occurrence of an oil pool depends not so much upon the geologic period in which the oil originated as upon the rocks into which it migrated and became entrapped.

Exploration for new supplies of petroleum to replace the everwaning production from known fields is an important function of the petroleum industry. It is well known that petroleum is found only where the geologic structure is favorable and that it is confined for the most part to definite horizons with impervious "caprock" overlying certain types of porous and permeable rocks that contain the oil, gas, and associated water in spaces between the solid material. The solution of the geologic problems concerned with the search for oil involves development of new applied techniques and modification or abandonment of old methods.

METHODS EMPLOYED IN PETROLEUM GEOLOGY

Geologic maps showing areal geology, paleontology, structure, thickness of beds, structural history, and differences in character of sedimentary rocks from place to place provide basic data needed in the search for oil and gas. With the assistance of these maps the geologist can determine where potential oil- and gas-bearing rocks are present, their thickness and the depth at which they may be reached by drilling, the structure of the beds, the location of reservoirs (*36*), and if the rock is porous enough to serve as a reservoir for oil and gas.

Geologic maps are prepared by the geological staffs of oil and gas companies, consulting geologists, and State and Federal agencies. The work of the Federal Geological Survey on the oil and gas resources of the United States dates back to the late 1880's and early 90's when reports were published on petroleum and natural gas in Ohio and Indiana. The continued and intensive program of oil and gas mapping was begun by the Geological Survey about 1900. These Government reports, as well as those of company geologists, demonstrate the relation of oil and gas accumulations to anticlinal structure. As a consequence, the early part of the present century witnessed the beginning of an intensive search for anticlines in the search for oil that is still in progress in the United States and has spread to other petroleum regions of the world.

TOPOGRAPHIC MAPS

Topographic maps are an essential tool in the search for and conservation of our natural resources. They are essential for the expansion of our transportation system, our basic industries, and our national security and economic welfare. Topographic maps are an invaluable aid to geologists as base maps for geologic mapping in the field. The important function of preparing and publishing these maps is handled by the Geological Survey.

AERIAL PHOTOGRAPHY

Aerial photography was first utilized for military purposes during World War I and was employed in about 1920 by oil geologists.

As a result of the tremendous improvement in airplanes, cameras, and photographic methods during World War II, the petroleum industry has universally adopted this indispensable tool. An oil geologist uses the mosaics as a base on which to plot the geology; and the individual photographs from which the mosaic is made are studied with the aid of stereoscope instruments to obtain an exact photographic model of the land surface. Many conditions bearing on the geology are more readily recognized in stereoscopic photographs than by actual field surveys. These include such features as change in vegetation and soil, irregularities of stream courses, and minor topographic features. Photographic maps also are used in designing pipeline routes, so that necessary variations from direct routing to avoid hazards can be anticipated. Indeed, aerial photographs are useful to so many industries and Government activities that about two-thirds of the area of the United States has been photographed in this manner.

SURFACE GEOLOGY

Surveys made to find favorable areas for oil by the mapping of anticlines, domes, and other structural features reached their peak between 1920 and 1925 and have thus occupied for many years a place of decreasing importance in the finding of new oilfields. Accordingly, from year to year an increasing amount of effort and funds is devoted to the study of subsurface geologic features that are not discernible from an examination of the exposed formations. Among such features are oil reservoirs of a type commonly known as stratigraphic traps. Such reservoirs may be the result of variation in porosity, permeability, and lithology, lensing, shoreline phenomena, and various lateral gradations (*40*). One-fourth to one-third of the past domestic production has been obtained from stratigraphic traps. Many oil pools in such reservoirs are of large size. Among the more outstanding are the East Texas field of Texas, the Glen pool in Oklahoma, and the Coalinga and Midway-Sunset pools in California.

SUBSURFACE GEOLOGY

Because of the increasing importance of subsurface geology in comparison with surface geologic mapping in the discovery of oil, the current methods employed in petroleum geology, as mentioned below, are used for the most part as tools for the acquisition of information in the solution of subsurface stratigraphic and structural problems.

Core drilling, for the determination of geologic structure, was introduced in the United States in 1919 and still is being employed although to a smaller extent than in former years because of the development of geophysical methods. In the Midcontinent and other regions it is current practice to obtain cores, especially from formations that are productive in other areas, for studies of their physical character and fossil content.

The microscopic examination of well cuttings was begun on a large scale in 1917 and since that time has reached a place of fundamental importance. A special method for the study of well cuttings of limestone and dolomite was introduced in 1924 by McQueen. It consists of a microscopic examination of insoluble residues obtained by dissolving the soluble part of the cuttings in hydrochloric acid.

Micropaleontology, which has been an integral part of the oil business in the United States since 1924, was employed for determining the age and the local and distant correlations of the physical properties of rock strata.

The orientation of well cores by magnetic polarity (developed in 1928) was much improved in 1935 and 1936 and provides a rapid and inexpensive method of determining in the laboratory the dip of the rock strata from core samples that have good bedding planes and enough heavier minerals in which polarity has been induced by the earth's magnetic field. Small-diameter cores obtained with sample-taking bullets (sidewall cores) shot from a device patterned after the casing perforator were first used in 1938 in conjunction with electrical logging, mentioned elsewhere. These small samples are taken at precisely located spots to check the properties of beds not adequately revealed by the electrical logs, to locate gas-oil and oil-water contacts more accurately, and to measure porosity and other physical properties of the cored sands.

In recent years rapid routine methods for measuring the physical characteristics of sandstone cores have been developed. These characteristics include permeability, porosity, grain size, and fluid content; the results obtained from their measurement are utilized in the estimation of oil and gas reserves in a pool.

GEOPHYSICAL EXPLORATION

Geophysical methods, including magnetic, gravimetric, electrical, and seismic, are widely used in the United States by the oil industry as a means for finding and mapping subsurface structural features. The adoption of the seismograph followed the discovery of its applicability during World War I for locating long-range guns. In its use by the oil industry the shots from long-range guns are replaced by shallow explosive charges in prospective oil localities. In 1924 oil companies discovered three salt domes in the gulf coast region by geophysical methods, but since then over 100 salt domes in the gulf coast region and many other structural features from New York to California have been located by such methods and later established by drilling.

The period 1934–38 was especially active in expansion of the use of geophysical instruments in the search for oil; their application has been rewarded during each year of the 5-year period by the discovery of an increasing number of oilfields. The limitations of special adaptabilities of the several geophysical methods for the solution of geologic problems have become better understood. A fuller understanding of the objectives sought has led to refinements of instrument construction and operational techniques.

GRAVITY METHODS

Gravity methods are designed to detect variations in the gravitational pull of the rocks that lie as much as several miles below the earth's surface. Structural highs, such as anticlines, often involve uparching of denser rock layers that increase the local attraction of gravity on the surface directly above. A salt dome, on the other hand, composed of lighter material than the surrounding sediments, usually will result in a slightly smaller gravitational force on the surface than will the sediments which surround it.

Since the difference may only approximate 1 part or less in 10 million, it has been necessary to develop gravity instruments that measure changes less than one ten-millionth the total force of gravity. From gravity data the geophysicist can estimate the location and magnitude of the corresponding density variation in the underlying rocks. Such estimates are never unique when obtained on gravity information alone, but on this basis locations can be singled out for further geophysical work or exploratory drilling.

MAGNETIC METHODS

In magnetic prospecting one looks for variations in the earth's magnetic field that can be attributed to anomalous magnetic properties in relatively shallow subsurface rocks. The variations could result from changes in depth of buried magnetic rock material, from changes in its magnetic properties, or from both.

During and after World War II newer techniques were perfected for magnetic mapping from the air. These have been highly effective, and widespread areas in all of the continents of the world have been surveyed by airborne magnetic instruments. The advantages of airborne over other older established techniques are: (1) Greater speed; (2) possible elimination of disturbing magnetic effects from near-surface irregularities; and (3) the ability to obtain data from terrain inaccessible to a ground surveying party.

As few sedimentary rocks are appreciably magnetized, the magnetic method of prospecting generally will give information only on igneous rocks or on ore deposits with magnetic constituents. In petroleum prospecting, magnetic methods are useful where structural features on the basement surface, such as buried ridges, control overlying sedimentary structure.

SEISMIC METHODS

These make it possible to determine the structure of rocks from variations in their elastic properties. Sound waves sent into the earth from explosions just below the surface are returned to the surface either by reflection from subsurface interfaces or by refraction along such surfaces. From the time required for the waves to reach detectors at various points along the surface are determined the depth and structure of the interfaces. Seismic methods give the most detailed information and also a unique picture of the subsurface in areas where characteristic formations show contrast in their elastic properties. Of all current geophysical techniques seismic methods generally are the most acceptable.

ELECTRICAL METHODS

Electrical methods utilize variations in the electrical properties of rocks or minerals and are so diversified that all types cannot be described in this summary. They do not afford the penetration of the rock obtained with other geophysical tools and consequently are not employed to any great extent in petroleum prospecting, where information on deep structures is generally required.

A widely used application of electrical methods in geophysics is electrical well logging. Here electrical properties of underground formations are measured continuously as electrodes are lowered down a drillhole. The logs thus obtained are used for correlation between wells and for other purposes. Radioactive, thermal, and other physical properties of rocks also are logged by instruments lowered into boreholes.

GEOCHEMICAL METHODS

As a result of field and laboratory studies of samples of surface soils gathered in areas overlying oil and gas pools, it has been established that such soils often contain small amounts of natural gas or even liquid hydrocarbons that apparently have found their way through the intervening rock masses by leakage through the caprocks. By oxidation of certain gaseous or liquid hydrocarbons, paraffin waxes may be formed in surface soils. This discovery has been responsible for a practice of field exploration, known as "soil analysis," that has been applied during recent years in the search for new oil and gas fields (*39*).

In using this method samples of soil are gathered at intervals over the area to be examined in accordance with a definite grid or profile, preferably several feet below the surface. An auger hand drill is used for this purpose. Upon removal the samples are placed in airtight containers and taken to a laboratory where they are extracted and tested for hydrocarbon content.

In the laboratory the analysis is conducted by driving the occluded or adsorbed gases from the soil samples by application of heat. The methane, ethane, and other hydrocarbon constituents are separated by fractional distillation and determined quantitatively by ignition. A more recent technique involves use of the mass spectrometer as a means of identifying and quantitatively measuring the hydrocarbon constituents of gases evolved from soil samples (*32*).

The hydrocarbon values thus determined are plotted on a map at the sample points, and the pattern of relative amounts of hydrocarbons is indicated by contouring. When the resulting map is inspected, it is often found that the hydrocarbon values are grouped around the productive area in a characteristic "halo," not only indicating the presence of an oil accumulation below but also suggesting its areal expanse. The methods of geochemical exploration also may be applied as a process of well logging by analyses of drill cuttings obtained from drilling.

It also has been noted that the ash obtained by burning various plants contains elements obtained from the soil in which the plants have grown. The concentrations of certain elements in the ash may be correlated with the presence or absence of hydrocarbons in the soils.

PRESENT ECONOMIC STATURE OF GEOPHYSICAL EXPLORATION

Geophysical exploration for oil in 1953 maintained but did not exceed the record high established on a global scale in 1952 (*31*). In 1953 (as in 1952) it is estimated that 350 to 400 million dollars was invested in worldwide geophysical exploration by the oil industry alone.

Activity has increased phenomenally in 1951 over 1950 by 100 million dollars—which represents an increase of nearly 50 percent. Additional millions undoubtedly were spent in mining explorations and for research in improving present techniques and developing new methods. The increased expenditures were due partly to inflationary rises in cost.

Geophysical exploration activity in 1953, on a worldwide scale, in maintaining the alltime high of 1952 employed 941 seismic crews, 129 gravity, 19 magnetic, and 16 others. The increased activity in foreign countries was accompanied by a slight decrease in the United States and Canada.

TECHNICAL ACHIEVEMENTS

Modern geophysical instruments and techniques are well developed, especially in the United States and Canada. Qualified geophysicists throughout the world concede freely that American instruments and geophysicists are the best to be had. Twenty-odd years ago the establishment of a single gravity station required two full days and a large truckload of equipment; the result, which did not become available until the pendulum could be standardized at the home base several months later, had a probable error of plus or minus 2 or 3 milligals. Today, only two short decades later, a modern gravimeter about the size of a 2-quart Mason jar and weighing only 6 pounds will establish a gravity station in about 2 minutes with an accuracy of plus or minus 0.02 milligal. This represents a fifty-fold advantage in weight, a one hundred-fold gain in accuracy, and a five hundred-fold gain in speed.

DISCOVERY AND DEVELOPMENT OF OILFIELDS

BASIC PRINCIPLES OF OIL-WELL DRILLING

Major oil-producing companies usually are divided into a land and geological department, a drilling department, a producing department, and an executive department (*13*). These departments, being complementary, share the responsibility of finding the oil, producing it, and preparing it for shipment to be processed.

After the land and geological departments have acquired the right to drill and indicated the locations to be drilled, the drilling department takes over and follows the program outlined for it. However, geologists continue to guide the drilling of wells, with the objective of guiding the proper and most efficient development of the properties. In smaller organizations the responsibilities of several departments may be assigned to one individual.

PROSPECTING OR "WILDCATTING"

After the geologist and geophysicist have found what appears to be a favorable location for drilling, the prospect is sometimes further checked by drilling a number of small-diameter holes, usually with a lightweight, portable drilling rig called a core drill (*59*). Where troublesome or badly caving formations are expected, wells are drilled of such size that emergency casing strings may be inserted to protect that part of the well already drilled. "Slim holes" sometimes are drilled primarily to gather information.

All wells drilled with the objective of discovering a new structure or a new pay zone or of extending the productive area of known pay zones are "exploratory" wells. The first well in a field usually is termed a "wildcat" well. Those wells extending a known field are "extension" or "outpost" wells. Wells drilled for deeper pay sands heretofore undiscovered in the same field are "vertical exploration" wells. Those wells drilled within the known boundaries of a proved field are known as "exploitation" or "development" wells.

While drilling several exploratory wells in a newly discovered field, the characteristics of the formations, pay zones, and drilling requirements are determined. The drilling department then organizes its skill and ingenuity to enable the exploitation or development wells to be drilled as economically and simply as possible.

Since oldtime "wildcat" wells were often located by pure guesswork or superstition, the term "wildcat" gained its colloquial meaning of "highly speculative." The present-day wildcat well still is highly speculative; however, its location usually is supported favorably by geological, geochemical, gravimetric, seismic, or electrical surveys; by aerial mapping, core drilling or other scientific work; and often by several of the above.

That drilling wildcat wells is still a venturesome business (*38*) is shown by the fact that during 1953, 13,313 exploratory holes were drilled in the United States. Of these, 6,925 were new-field wildcats, 3,358 were new-pool tests (including new-pool wildcats, deeper-pool tests, and shallower-pool tests), and 3,030 were outposts. Among the new-field wildcats, 774 were successful; among the new-pool tests, 924 were successful; and among the outposts, 982 were successful.

The total exploratory footage drilled in the United States in 1953 was 60,702,371 feet in the 13,313 holes or 4,560 feet per hole. These figures compare with 55,615,389 feet drilled in 12,425 exploratory holes, with an average depth of 4,476 feet, in 1952.

Table 7 gives information for 1944 through 1953 on the bases used in determining the locations of wildcat wells. The benefit of a technical approach to seeking new oil deposits is clearly illustrated.

DRILLING OPERATIONS

When the location of the wildcat well has been decided, the site of the well is cleared for operations. For wildcat wells in remote countries, houses and quarters must first be built for the drilling crews. The crew, which usually works in three shifts, consists, in all, of between 15 and 24 men including engineer, tool pusher, drillers and crews. The personnel concerned with the machine shop, materials and spare parts, warehouse, transportation, and office may sometimes raise to over 100 the staff required to serve the wildcat. Special services that may be required in addition include electric power, electric and acetylene welding, radio or telephone facilities, and automotive, boat, or even aircraft transportation.

Depending on the type or depth of the well to be drilled, the drilling machinery varies from a "light rig," using perhaps 200 horse-

TABLE 7.—*Methods used in locating new-field wildcats*

	Geology		Geophysics [1]		Geology and geophysics		Total technical		Nontechnical		Unknown		Total		Grand total
	Producers	Dry holes	Producers	Dry holes	Producers	Dry holes	Producers	Dry holes	Producers	Dry holes	Producers	Dry holes	Producers	Dry holes	
1944	148	1, 328	108	589	40	177	296	2, 094	38	570	8	88	342	2, 752	3, 094
1945	180	1, 488	89	468	53	220	322	2, 176	17	302	13	207	352	2, 685	3, 037
1946	154	1, 584	91	413	57	211	302	2, 208	10	363	21	229	333	2, 800	3, 133
1947	200	1, 691	101	516	63	293	364	2, 500	15	389	15	197	394	3, 086	3, 480
1948	232	2, 039	124	595	99	400	455	3, 034	12	372	34	389	501	3, 795	4, 296
1949	222	2, 133	180	650	62	360	464	3, 143	11	367	30	433	506	3, 943	4, 449
1950	279	2, 523	173	739	81	403	533	3, 665	19	526	40	507	592	4, 698	5, 290
1951	311	2, 932	208	990	86	556	605	4, 478	16	375	63	652	684	5, 505	6, 189
1952	359	2, 950	179	1, 115	143	705	680	4, 818	16	466	44	672	741	5, 957	6, 698
1953	368	3, 227	221	1, 080	100	745	689	5, 058	21	426	64	667	774	6, 151	6, 925

[1] Includes 5 dry holes based on geochemistry.

power for the drilling operations and capable of 3,000-foot drilling, to a "heavy rig," using approximately 2,000 horsepower at the rig on land and as much as 3,500 horsepower on a big offshore drilling barge capable of drilling to possibly 18,000 feet.

Depending on available fuel, drilling rigs are powered by steam, using locomotive-type portable boilers, or by internal-combustion engines. The wildcat well of today most often is powered by diesel, butane, or gas engines because of the greater fuel economy obtained in remote locations. A heavy steam rig may consume an average of 200 barrels of fuel daily, whereas a heavy diesel rig may average only 20 barrels a day. Water is an important item for drilling, and in arid countries internal-combustion engines frequently are used to save water also. As an illustration (*23*), a trade journal announced that a farmer near Old Glory, West Texas, has 11 oil wells on his farm, but he is making more money from one spring-fed tank. His customers are oil companies drilling wells in his area.

HISTORY OF DRILLING

The art of well drilling dates back to 221 B. C., when it was first practiced in China; there brine wells as deep as 3,500 feet have been drilled with very primitive equipment (*59*). Ancient illustrations of the tools then used show that they bore a striking resemblance to those used in modern cable-tool drilling, especially those for retrieving articles left in the hole. The Chinese drilled their wells by means of drilling bits that were moved up and down, a method similar to the percussion, or cable-tool, type of drilling. This method was used almost exclusively in oil-well drilling in the early days of the petroleum industry but now has been largely superseded by a rotary method of drilling; however, it is interesting to note that a substantial number of cable-tool holes are drilled in the United States today (table 8).

TABLE 8.—*Method of drilling* (*81*)

Year	Total wells drilled	Rotary	Cable tool
1945	26, 694	17, 908	8, 741
1946	30, 230	20, 859	9, 371
1947	33, 147	23, 269	9, 878
1948	39, 477	29, 410	10, 067
1949	38, 962	30, 975	7, 987
1950	43, 307	35, 295	8, 012
1951	45, 996	37, 717	8, 279
1952	46, 507	38, 368	8, 139
1953	49, 039	40, 948	8, 091
1954	51, 902	44, 117	7, 785

CABLE-TOOL DRILLING

For shallow drilling, portable machines sometimes called "spudders" are most often used. These are much lighter than standard cable-tool rigs, but they operate on the same principle and often with quite similar equipment. These portable rigs are generally equipped with a braced mast instead of a derrick, and the machinery is mounted on a four-wheel truck that can be readily moved about as a unit from one location to another and put into active drilling service within a few hours' time. For example, a well-known type of portable spudder is designed for drilling with cable tools to depths as great as 3,000 feet. This model is powered with a 128-horsepower engine and can handle 7,000 pounds of tools and drilling cable and lift 27,800 pounds at reduced speed. A chain-driven sand reel will lift a 7-inch by 30-foot bailer from a depth of 3,000 feet in 3½ minutes.

A string of cable tools is usually 40 feet long or more and consists of several parts securely fastened together by tapered screw ("pin") joints. The rope socket that connects the tools with the drilling cable is screwed to the top of a pair of massive telescoping metal links called "jars." These in turn connect at their lower ends with a long, cylindrical, steel "drill stem," and the latter is screwed to the top of the drilling bit. Occasionally a "sinker bar," a short cylindrical steel bar, is inserted between the top link of the jars and the rope socket. The aggregate weight depends upon the diameter of the hole to be drilled; for a 10-inch hole the weight of the tools usually averages about 3,600 pounds.

Cable-drilling bits are of several types that differ slightly from each other in form and purpose. The bit is made of a heavy bar of forged steel 4 to 11 feet long (commonly 7 or 8 feet) and somewhat wider than thick; it is dressed to a blunt edge on one end and terminates in a tapered "tool joint" at the other. The form of the cutting edge is varied to adapt it to the character of the rock formation to be drilled.

The rate of progress in cable-tool drilling will vary within wide limits. In soft rocks and shallow depths an advance of 100 feet or more may be made in a 24-hour day under favorable conditions. In a hard layer of rock 5 feet of hole may represent a good day's work. The equipment is usually operated day and night, with 3 crews of men working 8 hours each. Each crew consists of two men—the driller, who is in responsible charge of the work, and his helper or "tool dresser."

COST OF CABLE DRILLING

The cost of drilling with cable tools varies in different regions, depending upon accessibility, depth and character of the formations to be penetrated, prevailing wage scales, and other related factors. Within recent years contracts have been let for drilling wells to depths of less than 1,500 feet through hard formations overlying the oil sands in the northern Pennsylvania region for less than $1.00 per foot. Contract prices for cable-drilled wells in the Texas and California fields, on the same basis, using standard rigs, have been in many instances over $5 per foot.

Although the average cable-tool well drilled in the United States is approximately 1,900 feet, a new world record for a cable-tool rig was made in the New York State Natural Gas Corp. well in Cattaraugus County, N. Y. This well was drilled to a total depth of 11,145 feet with cable tools.

ROTARY DRILLING

Rotary drilling is of comparatively recent origin. The first recorded use of rotary methods in drilling for oil was in 1901 in the Spindletop field, near Beaumont, Tex.; however, it was not until the early 1920's that the rotary method of drilling came into its own.

The rotary method was developed because, as deeper horizons were exploited, wells became considerably more expensive and many months were necessary to complete them. Also, the cable-tool method is not well adapted to drilling in the semiconsolidated formations characteristic of the Texas Gulf coast and California formations. Caving of the hole wall created difficulties in maintaining clearance and inserting casing.

In many areas where wells drilled with cable tools penetrated semiconsolidated formations (particularly in California) casing had to be driven into the wells; individual strings quickly became "frozen" to the walls, so that as many as 5 telescoping strings of casing had to be used to reach depths of 2,000 to 3,000 feet.

It was thought for many years that rotary drilling was applicable only in fields where relatively soft formations were to be penetrated, but during more recent years development of hard-rock and diamond bits for rotary equipment has extended its field of usefulness, until at present there is scarcely any type of rock ordinarily encountered in drilling oil wells that cannot be drilled satisfactorily by the rotary method.

In drilling with rotary tools the rock through which the well is drilled is removed by the chipping, cutting, and abrasive action of a revolving steel bit. The bit is revolved by steel pipe (drill pipe) to which it is screwed; the drill pipe extends from the bit to a distance above the derrick floor through a power-driven rotary table that turns the drill pipe. The gripping device at the rotary table is such that the drill stem is free to move vertically as the table rotates.

Attached to the top of the drill pipe is a swivel through which water or mud is pumped down through the hollow drill pipe to the bit and out into the well through holes in the bit. The drilling fluid carries the drill cuttings to the surface through the annular space between the drill pipe and the wall of the borehole, as well as having other purposes, such as cooling the bit, sealing the wall of the hole with mud, and controlling subsurface pressures in oil and gas sands as the drilling operation is being completed.

A steel cable operating through sheaves in the top of the derrick is used to raise and lower the swivel, drill stem, and bit assembly. Other essential features of the equipment for rotary drilling include the source of power, the draw works, the derrick, mud pumps, and slush pits.

Preventing caving is one of the important functions of a drilling fluid in rotary drilling. Many shallow formations, including fresh-water-bearing sands, are comparatively soft and unconsolidated, so that they are easily drilled through. These porous rocks act as a strainer, holding the clay particles at the wall and permitting a limited amount of the water in the mud to filter through so that a filter cake forms at the sand face.

Much research has been done on improving the properties of drilling mud, so that now it is possible to form an almost impervious mud sheath very quickly, allowing little water or filtrate from the mud to enter the formation. Starch and various chemicals are used as additives to improve the properties of drilling fluids. Commercial oil-base muds have been developed, which are effective where water-base mud may be harmful because of penetration of the water into formations containing clays that swell upon contact with water.

In completing a rotary-drilled well, casing normally is set and cemented either at the top of the producing formation or at some selected distance into the producing formation. If the casing is set "through", or below, the producing formation, the well is completed by perforating the casing at intervals opposite the oil-bearing sand.

Final preparation of the well before being placed on production may include injection of acid into the reservoir rock, shooting with nitroglycerin, or hydraulic fracturing. This last method, a relatively recent development, has been effective in increasing the productivity of oil wells where the reservoir rock is relatively low in permeability; by fracturing the producing formation in the vicinity of the well bore, flow channels are extended out into the reservoir rock, thus increasing the effective drainage area of the well.

Various types of equipment and testing procedures are used in determining how wells are to be completed, the type and extent of the oil deposit at hand, and the procedure to be applied in operating the wells. Some of the more commonly used testing procedures include coring the producing formation and core-sample analyses, formation sample logging, electrical and radioactivity logging, temperature and pressure surveys, drill-stem productivity tests before completion of the well, and productivity after completion. Often caliper logging (measuring the diameter of the well bore) may be used to assist in determining the procedure for further drilling, for setting and cementing casing, or in determining the conditions in the well bore after completions. Special tools also may be used to scrape or wash the open hole in a well before a liner is set or the well is placed on production status.

The drilling of oil wells is a highly competitive business, and the costs for drilling vary over a relatively wide range in the United States. Many factors enter into drilling costs, such as type of terrain, availability of water and other supplies, and accommodations for employees, types of formations to be drilled, and specifications of and depths to which the wells are to be drilled. For example, drilling a 12,000-foot well in West Texas may cost as high as $12.50 a foot, while in Illinois a 3,000-foot well may cost about $4 a foot; a 6,000-foot well may be drilled in West Texas for about $5.50 a foot while the same depth well could be drilled in the gulf coast region of South Texas for $3.75 per foot.

PRODUCTION OF PETROLEUM AND NATURAL GAS

TYPES OF OIL DEPOSITS

Fluids within a reservoir flow in the direction of lowest pressure, that is, toward the producing wells (*59*). The movement of the oil is the result of (1) the propulsive action of the gas either free or liberated from solution in the oil which expands continuously as the pressure becomes lower and always exerts a force on the oil flowing through the pore spaces in the producing reservoir; (2) the displacement action of water from the adjoining parts of the sand; and (3) gravity drainage. Under these conditions, the underground space previously occupied by produced oil is filled completely by the expansion and movement of the oil, gas, and water remaining in the formations.

These three forces frequently are present simultaneously, depending on the nature of the sand, the structural features of the accumulation, and the rate of production. However, "free" gas, because of its lower specific gravity compared with that of the surrounding mixture of oil plus dissolved gas, often follows a path of its own and tends to collect in the highest part of the structure or natural trap. From the foregoing simplified description of fluid flow within a reservoir, the general types of pools may be classified according to the recovery mechanism that dominates as follows:

(1) Gas-drive fields are reservoirs where the oil accumulations are overlain by a gas cap. Movement of the oil results primarily from pressure of the gas within the gas cap, supplemented by the expanding gas liberated from solution in the oil.

(2) Water-drive fields, in which the space voided by the oil produced is refilled entirely or in part by water from the adjoining parts of the sand. No "free" gas is present in the reservoir as long as formation pressures are maintained above the saturation point—that is, above the pressure at which gas comes out of solution from the oil. A small fraction of the space voided by the produced oil is taken up by the expansion of the oil plus dissolved gas still present in the reservoir. After the East Texas field had produced 2 billion barrels of oil and the pressure had stabilized at around 1,000 pounds per square inch, it is computed that only 1 percent of the oil produced was replaced by expansion of oil still present in the reservoir; the remainder of the void space was filled with water.

(3) Depletion-type reservoirs are those from which the oil is produced by gas liberated from solution in the oil, and aided by gravitational force. After the gas originally present in the oil is depleted, gravitation is the only remaining force.

Displacement of oil by water usually will yield a greater ultimate recovery per unit volume of pore space than is obtainable when gas or gas and gravity are the driving forces in the reservoir system. Accordingly, water drive is recognized as being the most efficient recovery mechanism.

If an oil reservoir is exploited in a wasteful manner—that is, if large quantities of gas are produced per unit volume of oil—the energy soon will be expended, and a large quantity of oil will be left in the formation, and only the force of gravity can bring the oil to a well. Since, in general, this force is small compared to the forces originally present in the reservoir, the oil remaining after the gas pressure has been dissipated is often considered unrecoverable, although flooding the sand with water or repressuring with gas may result in the recovery of some portion of it.

In pursuance of the idea that useful work can be obtained from the natural energy in the reservoir, it is evident that the first step will be to rework or shut in wells producing gas only and wells producing disproportionately large volumes of gas with oil (high-gas-oil-ratio wells). In the first instance, the gas is propelling no oil to the well, and in the second it is being used inefficiently and therefore wastefully.

Water-drive fields may contain oil, gas, and water in the reservoir in the same relative positions as gas-drive fields. However, many water-drive fields have no "free" gas cap. In a water-drive reservoir most of the work in propelling oil through the formation is done by the water and gas plays only a minor part. The extent to which the pressure in the reservoir will be maintained by a water drive depends largely on the permeability and the thickness of the sand and on the rate of production. Thus, if the formation has considerable thickness and is highly permeable and an adequate natural supply of water is available, the reservoir pressure will remain close to the original, even at relatively high rates of production. On the other hand, the reduction in pressure in thin "blankets" of sands of low permeability can be appreciable even at restricted rates of flow, and the force of the water drive may soon diminish.

In depletion-type reservoirs the total decrease in pressure depends on the cumulative oil production. When the pressures have declined to the saturation pressure the oil is produced at increasing gas-oil ratios.

OIL-RECOVERY CHARACTERISTICS OF RESERVOIRS

The principal characteristics that distinguish gas-drive and depletion-type reservoirs from water-drive fields are the following:

GAS-DRIVE AND DEPLETION-TYPE FIELDS

As the amount of energy available for moving oil to the wells and lifting it to the surface is limited and therefore constantly declining during the life of the field, the production of the wells also will decline until finally fluid can no longer be produced economically. The first wells drilled will flow for a time, but the quantity produced daily will steadily decline as the formation pressure is reduced. Eventually, a stage will be reached when there is no longer enough energy to lift the oil to the surface, and this must then be done by artificial means (pumping the well). Finally enough energy will not be present to move oil through the formation to the bottom of the well. When this condition occurs, a quantity of oil, ranging from possibly 40 to as high as 95 percent of that originally in place, remains in the formation, and it may then be possible economically to extract some of this oil by secondary-recovery methods, which are discussed later.

WATER-DRIVE FIELDS

In those reservoirs where the supply of underground water is sometimes virtually inexhaustible, because the water follows the oil through the producing formation as rapidly as fluid is produced, wells often show no decline in the rate of production throughout their life. The proportion of water to total fluid produced by a well increases until finally the yield of oil is no longer economical. At this stage, the well is said to be flooded or "drowned out" and is of no further use as a "producer."

Water-drive fields are the easiest and cheapest to operate when the water pressure is great enough to lift the oil to the surface, thereby saving the capital and operating expenditure normally required for lifting equipment and services in gas-drive fields. Because of the relatively high efficiency of oil recovery by natural water drive, secondary-recovery methods usually are unnecessary. Well-known examples of fields under strong water drive are the Tampico field, Mexico, and the East Texas field, Texas.

PRODUCTION OF OIL FROM WELLS

FLOWING WELLS

When reservoir pressures are high enough, wells will flow; that is, the oil and gas are produced because of natural energy latent within the fluids as they exist within the reservoir rock. In flowing wells this energy is utilized not only to expel the oil from the reservoir rocks into the well bore but also to lift it to the surface. Pioneer wells drilled into high-pressure deposits often "come in" as "gushers," although modern methods and equipment for drilling and completing wells usually prevent the oil and gas from being blown to the atmosphere upon completion of the well.

Oil and gas may be allowed to flow to the surface through the casing, but usually the oil flow is confined within a smaller auxiliary column of tubing suspended within the casing. Side outlets equipped with suitable gate valves permit the release of fluids from the space between the casing and the tubing. High-pressure wells producing large volumes of gas and oil usually are equipped with an assemblage of valves and fittings known as a "Christmas tree," which is attached to the tubing or control head immediately above the master control valve.

When the well begins to flow slugs of dead oil by "heads" the installation of automatic flow equipment may greatly extend the natural flowing life of the well.

Flowing by heads could be corrected by replacing the tubing string with a string of smaller diameter, or even with several different diameters to simulate a tapered string, to maintain the velocity necessary to give steady continuous flow. However, in practice, tubing diameters of less than 2 inches are not generally workable for several reasons. These include the difficulty of swabbing to start the flow, the running of subsurface-pressure instruments, and the removal of wax accumulations with mechanical tubing scrapers.

ARTIFICIAL LIFT

After natural flow stops, it usually is necessary to install additional equipment to lift the oil to the surface. Lifting methods fall into two general classes: (1) Gas-lift methods and (2) mechanical pumping.

Gas Lift.—Gas lift is the best known pneumatic method of lifting oil and has been most widely employed. In a broad sense, the term "gas lift" also includes "air lift," a method identical in principle, which is widely used for lifting water from wells and for dewatering mines.

According to Uren (*70*), air lift was used to remove brine from wells in northern Pennsylvania as early as 1846. In 1864, during the early period of development of the petroleum industry—again in northern Pennsylvania—compressed air was used in a few wells to stimulate the flow of oil. In 1901 the method was introduced into the coastal fields of Texas and Louisiana and attained wide use. For a number of years, beginning about 1909, air lift was used in wells in some of the California fields. It was not until 1923, however, with development of the deeper California fields, that the gas-lift method of well operation came to be accepted generally by oil producers as a serious competitor of mechanical pumping. Soon thereafter, the gas lift was enthusiastically adopted by operators in the midcontinent area; a period of intensive study and development of the method followed that has brought gas-lift practice to its present high degree of efficiency.

For isolated wells the compressor equipment may be a single unit, but where several nearby wells are to be operated, the usual practice is to concentrate all the compressor equipment into one central plant.

Advantages and disadvantages of gas lift.—Gas lift has a number of advantages over mechanical methods of lifting petroleum from wells. It has large capacity, thus providing a means of securing high daily yields; daily capacities as large as 10,000 barrels of liquid have been handled successfully through 4-inch tubing in some wells. Gas lift operates continuously and at low unit-production cost. For pressures of about 300 pounds per square inch, compression costs range from 1.75 to 6.5 cents per 1,000 cubic feet, depending upon plant capacity, type of power employed, and if gas is not purchased. Before the inflation period, total lifting costs in California fields by the gas-lift method ranged from 5 to 13 cents per barrel, averaging about 8 cents per barrel for wells producing 100 to 550 barrels daily from depths of 2,600 to 5,000 feet. Another advantage is that gas lift is adaptable to deep wells and crooked holes—conditions that often make mechanical pumping difficult and inefficient.

There also are certain disadvantages. The gas-lift plant requires a large initial capital outlay, and the payout period is usually short. The method is not adaptable to handling production at the low rates characteristic of the later productive life of oil properties and hence must be replaced by other lifting methods. If water enters the well with the oil, gas lift may result in the formation of an emulsion difficult to demulsify. Corrosive effects of water are aggravated if air is used as the lifting medium; and if the gas contains sulfur compounds, the tubing and well casing are rapidly destroyed by chemical action. Paraffin accumulations in the flowline are troublesome in handling some types of oil by gas lift.

Mechanical Devices for Lifting Oil From Wells.—The mechanical devices employed for lifting liquid in wells include the various types and styles of oil-well pumps, which may be broadly classified into three groups: (1) Rod-actuated plunger-displacement pumps: (2) fluid-actuated plunger-displacement pumps; and (3) centrifugal pumps. In addition, bailing and swabbing may properly be included among the mechanical methods of lifting oil from wells, although these normally are used only temporarily during the early life of wells or when for some reason other methods are impossible or inappropriate.

Most of the present-day pumping wells are equipped with rod-operated plunger pumps. Pumps of this type are adaptable to a wide variety of conditions, particularly those characteristic of the later period of productivity of wells when the rate of production is comparatively low. However, such pumps have depth and capacity limitations, and in many fields the other types are finding increasing utility. Oil producers are vitally concerned with lifting costs. Therefore, they must consider the efficiencies of the oil-well pumps employed and the relative advantages of different prime movers and methods used in applying power to the oil-lifting devices.

A plunger pump is simply a suction pump with a hollow moving piston. Attached to the lower end of the tubing is a pump cylinder or "working barrel," with an inlet or "standing valve" at the lower end. The moving plunger with rubber cups and "traveling valve" are attached to the lower end of the rods. The reciprocating motion (up-and-down movement) of the rods is supplied by a cable-tool "walking beam" or a pump jack.

Many different types of plunger pumps with many types of valves and seats are in use. The brass or bronze valves are used where corrosive waters are handled, and stainless steel or chromium-plated balls and seats are used occasionally under highly corrosive conditions. Chromium and other special alloy steels are employed where sand is produced with the oil.

Under difficult pumping conditions, an insert pump is used; all parts of the pump that are subjected to wear are brought to the surface merely by pulling the rods. Insert pumps generally have a lower capacity than the standard form of pump normally used; consequently, they are best adapted for use in wells that have declined to relatively low rates of production.

A fluid-packed pump is used for handling oil with a considerable amount of sand in suspension. Sand between the polished surfaces of the plunger and working barrel of the normal type pump soon results in leakage, and the operating efficiency rapidly declines. The fluid-packed pump is designed to prevent wear by providing an unusually large clearance between the barrel and the plunger, and as a result it is capable of successfully handling fluid containing larger amounts of sand than pumps equipped with close-fitting plungers. The fluid-packed pump also operates better at higher underground temperatures.

Circulating pumps also are specially designed to handle sand-laden oil. This type of pump employs a stream of clean oil that is forced down from the surface, outside of the tube that carries the ascending fluid. The clean oil lubricates the plunger, prevents sand from entering the space between plunger and the barrel, and by the additional volume of fluid thus provided increases the ascending velocity and sand-carrying capacity of the fluid.

Hydraulic and centrifugal pumps.—Plunger pumps of normal design are inefficient in deep wells because of the stretch of the sucker rods. In crooked holes the rods and tubing are subjected to destructive wear. So great are these difficulties, particularly the stretch of the rods, that it is scarcely feasible to operate rod-actuated pumps at depths much greater than 8,000 feet. Hydraulic pumps and multistage centrifugal pumps are used under the foregoing conditions.

Central pumping unit.—In fields where the wells are not very deep, the well spacing not too wide, and the topography favorable and there are few or no obstructions, the use of "central pumping power" may result in considerable savings. By means of eccentrics or cranks the central power imparts a reciprocating motion to attached rods serving a number of wells.

MAINTENANCE OF PRODUCTION

REMOVAL OF WAX

The production of certain types of oils is impeded by the formation of waxy residues that tend to clog the pores of the reservoir rock, well-casing perforations, screens, and tubing through which the oil flows or is pumped to the surface. This material, commonly referred to as "paraffin," varies in consistency from that of vaseline to that of hard, natural wax (ozocerite). The paraffin accumulation in wells may result from any one of a variety of causes. Solid-paraffin crystals formed as a result of the drop in temperature caused by rapid expansion of the associated gas, the presence of water causing emulsification, and suspended particles of inorganic silt all apparently serve as nuclei for the formation of waxy deposits.

A common method of removing wax is to introduce solvents, such as benzol, gasoline, and heavier distillates, into the producing well. Frequently the solvent is preheated to increase the solubility of the wax. This method is only partly effective, as wax is not readily soluble. Other commonly used methods involve direct application of heat to melt the wax so that it flows from the sand face of the well bore. Superheated steam frequently is used. Heat also is generated with mixtures of chemicals such as calcium carbide or caustic soda or lye and metallic aluminum filings with water.

Electric heating devices, lowered into the well on a two-wire insulated and armored cable, have been effective provided nonmagnetic armor is used on the cable to minimize induction losses.

REMEDIAL WORK

Throughout the economic life of a producing oil well considerable care must be given to it to keep it in a producing status. An increase in the amount of water entering a well creates problems such as water disposal, with its attendant possibilities of pollution. Although the total amount of fluid lifted to the surface may be the same, the quantity of the net oil may dwindle to where the production is nearing the economical limit. Such wells may be deepened to a lower productive horizon, "plugged back" to a shallower productive horizon, converted into a waste-water disposal well, or abandoned.

SECONDARY RECOVERY OF OIL

STRIPPER WELLS

The term "stripper wells" has been used in the petroleum industry for many years in a more or less generic sense without specific definition. (Some engineers contend that a "stripper" well is not subject to exact definition.) In the early days of the industry a stripper well was one through which the last remnant of recoverable oil was being "stripped" from the producing sand. For the purpose of the following discussion and in a strictly economic sense, without implication of the various regulatory factors, stripper wells may be considered as "marginal"—those in which the value of the oil produced approaches the cost of production, having only a small margin of profit.

An oil producer whose properties fall in this category faces some perplexing problems regarding future operations. The price of crude oil at the wellhead, both now and in future, is among the first of the factors to be considered. When crude-oil prices are low, many operators plug stripper wells to recover equipment for use in more profitable ventures. Many small operators may reach the decision that, at prevailing prices, the salvage of casing and equipment will bring a larger return in dollar value than oil production in future years. If the decision is reached to salvage and abandon the wells recoverable crude-oil reserves will be lost. This problem is serious and must be faced by most of those who operate stripper wells.

SECONDARY-RECOVERY OPERATIONS

There are various definitions of "secondary recovery." The following is a definition by Paul D. Torrey, chairman, Secondary-Recovery Advisory Committee, Interstate Oil Compact Commission (7):

> Recovery by any method (natural flow or artificial lift) of that petroleum which enters a well as a result of augmentation of the remaining native reservoir energy (as by fluid injection) after a reservoir has approached its economic production limit by primary-recovery methods.

More recently, that committee, now called the Secondary Recovery and Pressure Maintenance Committee, shortened the definition to read:

> Secondary recovery is an application of fluid injection when a reservoir is approaching or has reached economic production limits.

VACUUM OPERATIONS

According to the above definition as strictly applied, the use of vacuum is not regarded as a secondary method; however, vacuum is mentioned because it was one of the first methods applied to wells to increase the oil-production rate. Many early patents state that a vacuum was first applied in the Old Triumph pool, Pennsylvania, in 1869. It has been well established that James D. Dinsmoor in 1895 attempted to apply a combination of vacuum and repressuring to wells in Clarion County, Pa. The attempt was unsuccessful because of the tightness of the producing formation. Shortly thereafter Dinsmoor successfully used combined vacuum and repressuring in Venango County, Pa.

By 1907 vacuum was being employed rather generally to stimulate oil production in the eastern and Midcontinent fields. It was soon recognized that many production benefits resulting from vacuum application were not permanent. However, vacuum was maintained on many properties because of competitive necessity or because of the increased recovery of casinghead gasoline. When it became better recognized that rarely will the application of vacuum materially increase the ultimate recovery of oil, many plants were converted for air or gas injection.

AIR AND GAS INJECTION

After production has proceeded for a time in an old field, so that reservoir pressure has been reduced, the pressure may be restored or partly restored by providing new energy for expelling residual oil through the injection of high-pressure gas into the partly drained pore spaces in the reservoir rock. Natural gas, air, flue gases, exhaust gases from internal-combustion engines, or other gases may be used for this purpose (*70*).

In 1888 Dinsmoor observed a producing phenomenon while working on the Hill property in Venango County, Pa. Wells on the Hill property produced oil from the Third Venango sand. On an adjoining property the Oil Well Supply Co. deepened one of its Third-sand oil wells to the underlying Speechley sand to obtain a gas supply for lease purposes. A substantial quantity of gas was found in the Speechley sand; but as the operator did not have a string of pipe and a packer immediately available, the well was shut in temporarily to save the gas. An immediate increase in the rate of oil production was observed in three nearby Third Venango-sand oil wells, and this increase was maintained until the Oil Well Supply Co. was able to obtain a string of casing and run it with a packer through the Third Venango sand. When this was done the rate of oil production from the adjoining wells returned to the previous normal level. This accidental repressuring operation represented the first known application of secondary-recovery methods in the United States.

Various attempts in adapting methods of repressuring and vacuum in Pennsylvania were unsuccessful, and it was not until 1903 that Dinsmoor and associates successfully operated a combined vacuum and repressuring operation on their shallow Cow Run-sand properties in the vicinity of St. Marys, W. Va. In 1909 a compressor was added to supplement the action of the gas pumps and to aid in condensing casinghead gasoline. In the following years, gas injection has spread to all the producing States.

WATER FLOODING

Torrey reported it is reasonably certain that intentional water flooding of oil sands was initiated in the Bradford field in Pennsylvania and New York. Increases in the rate of oil production from this method of operation first were noted in 1907, although it is believed that water flooding was being done secretly before that time. Because of the clandestine nature of most of the early water-flooding operations in northern Pennsylvania, little detailed information on the results obtained, such as is available in the early air-gas injection operations in Pennsylvania, West Virginia, and Ohio, has been preserved.

Intentional but haphazard water floods are believed to have been in operation in the shallow fields of Chautauqua County, Kans., as early as 1916; however, active development of water-flooding projects in the fields of southeastern Kansas awaited the successful results in 1935 of the first systematic floods in Oklahoma. Water-flooding operations in Kansas and Oklahoma have expanded tremendously since that time.

The first intentional water-flooding operation in West Virginia was begun in the Cabin Creek field, Kanawha County, in 1937. In Ohio operations started in the Chatham field, Medina County, in 1939 following the enactment of a law by the Ohio Legislature permitting the practice. Water flooding in Illinois was begun in 1924 by the partly accidental type. Two water floods were begun in Texas in 1930—one in Brown County and another in Wichita County. In 1942 Kentucky joined the other States in water flooding, but it was not until 1944 that an experimental water flood was started in California in the Richfield field of Orange County.

Many reputable authorities believe that, under favorable conditions, as much oil may be recovered by secondary-recovery operations as was recovered by "primary" operations; however, it is most important that all known factors relating to the subsurface conditions of any prospective secondary recovery field be analyzed by competent engineers before the operations are started. Many factors involved in water flooding may result in a narrow margin of profit and high operating costs.

SEPARATING, MEASURING, AND COLLECTING OIL AND GAS

Liquid petroleum, as produced at the wellhead, is commonly associated with natural gas and frequently is contaminated with water and suspended solids—mostly sand and shale. The maximum amount of water and suspended solids permitted by pipeline-transportation companies is 3 percent.

NATURAL-GAS SEPARATION

Natural gas is generally removed from crude petroleum with the aid of a suitable gas trap or oil and gas separator. The "trap" may be a cylindrical vessel 2 to 5 feet in diameter and 10 to 20 feet in height. The entrained gas is separated from the liquid, which may be clean oil, a simple mixture of oil and water, or an emulsion of oil and water. Gas separators may be "high pressure" or "low pressure" and often these are used in combination. The gas from the separators is used for lease operation or flows through gathering lines to the gasoline plant for treatment.

SEPARATING WATER FROM PETROLEUM

Wells may produce water with the oil, and the water must be separated from the oil in the field before it is handled in the transportation pipelines. Frequently it is necessary to dispose of the water—usually a strong brine—so that it will not damage growing crops or contaminate streams. Where there is no emulsification, oil and salt water are separated in a settling tank. The water is drained off by gravity separation from the bottom of the tank. Emulsions of oil and water are "broken" through the use of heat, chemicals, or both.

GATHERING SYSTEMS AND LEASE STORAGE

A petroleum producer must provide facilities for storing oil on the property and for transporting it from wells to storage facilities. Limited storage facilities usually are provided at or near each well when a lease first is being developed. The decision as to the size and number of lease tanks to be provided may be based upon the experience with earlier wells in the vicinity. Steel tanks normally are preferable because of ease of measurement and prevention of evaporation losses. When tanks are used for initial storage at a well they are connected directly with the leadline, which conducts oil from the well or oil and gas separator.

Topography permitting, the gathering system will be designed so as to take full benefit of gravity flow.

GAGING, SAMPLING, AND TESTING CRUDE PETROLEUM

To account for oil production from properties under lease, an oil producer regularly determines the volume of oil shipped. The volume of oil run from the producer's tanks to pipeline companies or other purchasers is measured or "gaged" in order that proper financial adjustments may be made between the buyer and the seller. The quality of the oil—that is, its gravity and the amount of water and suspended solids present—is equally important, because values are often based on density, and payment may be made for "net" oil or oil free from impurities. Furthermore, many pipeline companies refuse to accept oil containing more than a certain allowable percentage of impurity—usually from a trace to as high as 3 percent—and tests must be made on representative samples of the oil to be sure that it meets pipeline requirements. The gaging and sampling of oil and making of necessary tests to determine the quality of the product are usually intrusted to individuals called "gagers," who take measurements and samples in the field, make the necessary tests in the field laboratory, and prepare reports and records of production.

FIELD TESTS APPLIED TO CRUDE PETROLEUM

The routine tests ordinarily applied to crude petroleum are confined to a simple hydrometer test to determine density and a centrifuge test to determine the percentage of water and suspended solids. Occasionally, distillation tests, viscosity, flash point, calorific value, and other tests will be applied for special purposes, but the gager seldom is equipped for these.

Measurement of oil temperatures is at least as important as other measuring operations. In the United States all relatively large quantities of liquid petroleum products are handled and sold in volumetric quantities corrected to the standard temperature of 60° F. The tables widely used in the oil industry for correcting volumes to 60° F. are those contained in publications that have been issued by the American Society for Testing Materials. The tables permit a person to correct any given volume of the oil of known gravity and temperature to the corresponding volume at 60° F.

NATURAL GAS—OTHER LIQUID HYDROCARBONS

The methods of producing natural gas are similar to those used in producing crude petroleum, the only differences resulting from the physical characteristics of the two commodities; however, before proceeding any farther into the details of production operations it becomes necessary to review the chemical composition and some definitions relating to various kinds of natural gases.

COMPOSITION AND ANALYSIS

Natural gases are found in rocks. There are several types and many varieties of natural gases; but the term "natural gas," in the trade sense, includes all varieties in which the paraffin series of hydrocarbons predominates.

The natural gas of commerce varies in B. t. u. per cubic foot content. Ley (*1*) reports 850 to 1,100 B. t. u. per cubic foot as the range of heating value of pipeline gas, as delivered to consumers and an average composition (by combustion analysis) as shown in table 9.

TABLE 9.—*Average composition of pipeline gas, percent* (*1*)

Constituents:	
Methane (CH_4)	72.3 (active)
Ethane (C_2H_6)	14.4 (active)
Carbon dioxide (CO_2)	.5 (inert)
Nitrogen (N_2)	12.8 (inert)
Total	100

At ordinary temperatures and pressures the gaseous hydrocarbon constituents of commercial natural gas from producing wells are principally methane (CH_4) and ethane (C_2H_6), usually with varying amounts of propane (C_3H_8) and butane (C_4H_{10}) present as gases. The heavier members—pentane (C_5H_{12}), hexane (C_6H_{14}), heptane (C_7H_{16}), and others, if present—also occur as vapors. Most of the hydrocarbon constituents heavier than ethane, if present, are extracted before the natural gas enters main pipelines for transmission to consumers.

The gaseous impurities in natural gas at the well are nitrogen, carbon dioxide, and hydrogen sulfide, and there may be traces of helium, oxygen, hydrogen, and other substances. If much hydrogen sulfide is present (over 20 to 30 grains per 100 cubic feet), it is removed before the gas is used commercially. Generally nitrogen and carbon dioxide are not removed. If enough of these impurities are present, the B. t. u. value of the gas may be lowered to an extent that the gas has no market value under present conditions of price and availability.

Three types of reservoirs—dry gas, condensate, and oil—produce natural gas. Gas from dry-gas reservoirs does not change to liquids when the pressure and temperature are reduced. In a condensate reservoir the hydrocarbons existing originally in the gaseous state change to liquid (retrograde condensation) with a reduction in pressure. These hydrocarbons which change to liquids are those that normally are liquid at atmospheric conditions. In oil reservoirs gas is in solution in the oil or is in a "gas cap" in the crest of the reservoir. Free gas may be produced or solution gas may be liberated as pressure is reduced and the oil is subjected to surface conditions of pressure and temperature.

GAS-OIL RATIO

Since gas plays such an important role in the production of oil, the amount produced with the oil has become an index of production efficiency. This index of efficiency is commonly known as "gas-oil ratio" and is the ratio of the number of cubic feet of gas produced with each barrel of oil measured under standard conditions of temperature and pressure. When the depth of the well, the bottom-hole pressure, the gravity and water content of the oil, and other conditions are the same, the lower the gas-oil ratio, the more efficient is the use of the natural gas in lifting oil to the surface. Where the gas-oil ratio becomes extremely large, the well is no longer considered an oil well but a gas well.

To keep the gas-oil ratio at a minimum requires careful control of production from the reservoir. As the original sources of the gas may be solution gas in the oil plus gas from the "gas cap," production control becomes a determining factor in the gas-oil ratio obtained. The gas cap in an oil field is kept intact as much as possible by selective well completion, which consists in isolating the producing intervals to exclude such free gas. Where this is unsuccessful, the output of high-gas-oil-ratio wells at or near the gas cap may be restricted.

The purpose of conserving reservoir energy and producing oil with minimum gas-oil ratios is to obtain the maximum practical ultimate yield of oil from the reservoir. This result can be obtained only if the pressure of the reservoir is kept high. In the absence of natural means, such as water encroachment, the reservoir pressure may be maintained by injecting fluid such as gas, air, or water.

Pressure maintenance generally applies to early injection of gas or water into the reservoir, to prevent or retard the decline of reservoir pressure. A "repressuring project" refers to those reservoirs in which the pressure has already declined and injection of fluids of one type or another is employed to restore in part the lowered pressure.

Gas produced with oil is generally the source of injection gas when the reservoir is being produced by pressure maintenance or repressuring. Other sources may be dry gas from a separate horizon of the same field or gas from a different field. The gas is separated from the oil at a field separator and usually is processed in a natural-gasoline plant for extraction of its liquid hydrocarbons. The residue gas from the plant then is recompressed and injected into the reservoir through injection wells, usually at a high point on the structure.

CONDENSATE-BEARING GAS AND THE CYCLING PROCESS

In the late 1920's some wells drilled to deeper sands in the gulf coast area were found to produce a straw-colored or water-white liquid that resembled gasoline or distillate.

This liquid material recovered at the well separator is produced with large volumes of natural gas. The proportion of liquid to gas is different for each reservoir, ranging from 10 to 75 barrels per million cubic feet of gas. At first many thought that the liquid originally existed as such in the reservoir in much the same manner as crude oil; however, it was recognized later that the produced liquid was in the form of a gas when in the reservoir at the original temperature and pressure and became a liquid only after a reduction in pressure.

Condensation of the gas to form a liquid by a reduction in pressure is a phenomenon directly opposite to common concepts. Normally when a substance is near the dewpoint, an increase of pressure results in condensation, and a reduction in pressure causes evaporation. The apparent anomaly of condensation by pressure reduction was suggested by Kuenen, a German scientist, as early at 1892 but was not established as applicable to petroleum until 1932.

A gas-condensate reservoir therefore is identified as one in which hydrocarbons originally are in a single gaseous phase at reservoir pressure and temperature and where, on reduction of pressure, condensation of some of the hydrocarbons will result. The liquid so obtained is termed "condensate." A gas-condensate reservoir is considered a gas reservoir; such a producing well is classified as a gas well by most States, even though a considerable quantity of light oil or condensate is recovered. In addition to the condensate liquid, gas from this type of reservoir also contains natural gasoline and lighter hydrocarbons similar to those found in gas associated with oil. All of the extractable constituents, which exist in the gaseous phase in the reservoir, fall into the broad classification of "natural-gas liquids." In Texas and Louisiana, where most of the known condensate fields are situated, venting of gas from wells to the atmosphere is prohibited. As the amount of liquids recoverable from a reservoir of this type ranges from 10 to 75 barrels per million cubic feet of gas, large quantities of processed gas must be disposed of in producing such liquids. As present markets frequently will not absorb such volumes of gas, operators are returning large volumes of the dry residue gas to the reservoir and increasing the ultimate recovery of valuable liquid hydrocarbons by the operational procedure of "cycling."

In this process gas produced from gas-condensate wells is treated for the recovery of condensate and other natural-gas liquids. The dry residue gas is then compressed and injected into the reservoir; thus the processed gas will be available for marketing at some future time. The maintenance of the reservoir pressure by gas injection prevents or retards condensation and resulting loss of liquids. Cycling is an efficient production method in that both the maximum recovery of liquid hydrocarbons is obtained and waste of the dry residue gas is prevented where market outlets are limited.

OTHER EXTRACTION AND RECOVERY PROCESSES

Methods of extracting hydrocarbon liquids from natural gas have developed through three stages. In the early history of the industry, this extraction was accomplished chiefly by compression and cooling. The crude natural gasoline recovered by this method contained a considerable amount of butane and propane, which made it unstable. When vaporized these more volatile fractions carried with them some of the desirable heavier components.

Compression plants were simple in design and their cost was small. For this reason many small units were in operation. However, because of the inefficiency of this method of extraction, the number of compression plants has declined and they have been replaced by larger and more efficient "absorption-type" units.

Another method that came into limited use some 20 years ago was the charcoal-adsorption process, which consisted of passing the gas through adsorber towers containing activated charcoal to adsorb liquid hydrocarbons from the gas. Liquid hydrocarbons were then recovered by passing live steam through the charcoal. This method has been replaced almost entirely by the oil-absorption process, which gives high extraction efficiency and is capable of close operational control.

A plant using a refrigeration process for extracting natural-gas liquids from natural gas being transported through a high-pressure transmission line was placed in operation by the Tennessee Gas Transmission Co. in 1951. This gas had previously been treated in the field to remove enough condensables to make it suitable for pipeline transmission. By cooling the gas stream to −96° F. about 15 percent of the ethane and most of the heavier hydrocarbons are removed from the gas stream. The primary purpose of the plant is to recover light hydrocarbons for use in the manufacture of chemicals.

TRANSPORTATION AND STORAGE

A separate major division of the petroleum industry known as transportation handles the movement of petroleum and natural gas and their products from their points of production to their points of consumption. Although the transportation phase of the oil industry can be considered to include movement of refined products from the refinery to the consumer, there is a natural line of demarcation between transportation of crude oil and natural gas and transportation of refined products.

PIPELINE TRANSPORTATION OF CRUDE PETROLEUM

Although railroad tank cars, inland-river and lake barges, oceangoing tankships and motortrucks share, to an important extent, the transportation of crude petroleum, by far the most important modern method of transportation of crude petroleum is by pipeline. Virtually all movement of oil from the wellhead to the shipping point is accomplished by field-gathering lines, and a large part of the crude oil is moved by trunk pipelines from field shipping points to the refineries.

Discovery of new oilfields in many scattered regions of the United States and the necessity of distributing petroleum among many of the more important industrial centers of the country have brought about a gradual development of the present-day pipeline system. In 1952 nearly 143,268 miles of crude-oil pipelines served the petroleum industry in the United States, 75,228 miles of which were classified as "trunklines" engaged in long-distance transportation of oil from the producing fields to refineries and transshipping points. The balance were "gathering lines" used in moving crude petroleum from wells to field shipping points. At the beginning of the century there were approximately 28,000 miles of all types of pipelines.

Pipe was one of man's early inventions. Ancient cities made extensive use of pipelines for water. With discovery of oil, it was inevitable that someone would use pipe for transporting petroleum. The early pipelines were gravity lines and practical only for short distances. The use of a pump for forcing oil through pipe marked the real beginning of modern oil pipelines.

Improved techniques in operating and constructing pipelines have been attended by gradual increases in capacity. For oil trunklines, the size of pipe ranges from 6 to 24 inches in diameter. Some pipelines with diameters up to 30 inches are being constructed.

CONSTRUCTION ADVANCEMENT

Before 1920 most pipelines were the screw-connection type, which used a threaded joint, collar, or coupling to join the pipe ends (*50*). Many of these lines were primarily gathering lines, with a local refinery or loading point for railroads or barges as their destination.

From 1920 to 1937 many major changes in pipeline construction were made. During this period the industry reached its greatest growth percentagewise. Construction contracts became standardized, and contractors began doing work all over the country, moving their heavy equipment by rail and truck.

Probably the greatest change in pipeline construction was in the way the pipe was jointed. At first the screw couplings were acetylene-welded to eliminate leakage. This was quickly changed to acetylene welding of unthreaded pipe to save the cost of threads and couplings. Electric welding came into use about 1928, and for several years was in close competition with acetylene welding. Now it is predominant.

Also, during the period 1920–37 protective coating of pipe was recognized to have major importance in keeping future maintenance costs low. Before this time a cold application of coating had been used. Later, a hot coating of bituminous material was applied on top of a primer with an outside wrap.

Advancement in pipeline construction to present-day methods is mainly the result of improvement in equipment. To a great extent ditching is done with machines. The pipe coating and wrapping operation is entirely mechanized. Sideboom tractors move and lower the pipe in the ditch, and backfilling is accomplished with modern-type bulldozers. Certain operations, however, such as welding the joints in a pipeline, still are done manually.

Improved techniques in operating and constructing pipelines have been attended by gradual increases in their carrying capacity. Possibly the most widely publicized was wartime construction of the "Big Inch," a pipeline 24 inches in diameter completed in 1943 to transport oil from Longview, Tex., to refineries in the New Jersey and Philadelphia, Pa., areas. The throughput of the Big Inch was almost 110 million barrels of oil annually.

Operation of a trunk pipeline requires a complex system of terminals, pump stations, and communication facilities. Pumping equipment and storage facilities at the terminals and pump stations vary with the type of crude oil to be handled. Frequently facilities for heating the oil are required to maintain the viscosity low enough for efficient movement during cold seasons. Many pipeline operations are automatically controlled. An efficient communication system, including private telephone lines and radio and television, are necessary for maintaining normal operation and for emergencies.

COST OF PIPELINE TRANSPORTATION OF PETROLEUM

Transportation costs vary widely, depending upon the character of the oil, the size of the pipe, the character of the terrain traversed, the number of pumping stations, the kind of power used and other variables. Pipeline construction costs range from $5,000 to $70,000 per mile. A trunkline pumping station may cost from $75,000 to more than $1,000,000.

REGULATIONS GOVERNING COMMON-CARRIER PIPELINES

The oil-carrying pipelines of the United States have been declared common carriers and as such have been placed under the jurisdiction of the Interstate Commerce Commission and the appropriate State commissions. In addition to numerous special regulations on different pipelines, several conditions generally are imposed upon pipeline shippers: (1) The minimum quantity that will be accepted for shipment ranges from 10,000 to 100,000 barrels; (2) the individual shipments must be of the same gravity, and impurities (sand and water) cannot exceed a certain maximum percent; (3) transportation companies do not bind themselves to deliver the identical oil received for shipment, but merely equivalent oil; and (4) most schedules provide that a pipeline company may deduct 2 or 3 percent to cover oil lost in transit.

PRODUCTS PIPELINES

Products pipeline systems cannot be described in such broad terms as have been employed for crude-oil pipelines; however, the increased total movement of products by pipeline and the probable total tonnage of steel required for new construction are of interest.

According to the Interstate Commerce Commission products-pipeline mileage in the United States in 1952 totaled 19,305 miles. This includes only carriers that report to the Interstate Commerce Commission.

According to Bureau of Mines statistics, the mileage of refined-products trunklines was increased 18,154 miles during the period from 1941 to 1953. Products lines are operated in 39 States and the District of Columbia; since 1950 lines have been laid in four additional States—Idaho, Oregon, Utah, and Washington. Pennsylvania leads the other States, with a total of 2,785 miles of products lines.

An interesting feature about products pipelines is that a number of different products can be pumped into a line consecutively; by maintaining the flow rate in the line above a predetermined minimum the amount of contamination is negligible, being only a small fraction of 1 percent of the volume of the products handled.

WATER TRANSPORTATION OF PETROLEUM

Water transportation of petroleum in tankers ranks second only to pipelines in the volume of crude petroleum transported in the United States. From an international viewpoint this method of transportation would rank first. In addition to a large domestic coastal movement, virtually all foreign trade of the United States in crude petroleum and its products is necessarily conducted with tankers. Also, a large portion of the inland distribution of petroleum and petroleum products is accomplished by the use of barges operating on canals and the larger rivers and lakes.

In 1952 the gross investment of the industry in marine equipment for water transportation of petroleum and its products was well over a billion dollars.

TANKERS

The first cargo of petroleum to be transported across the Atlantic Ocean was carried in wooden barrels stowed in the single hold of the brig *Elizabeth Watts* in the year 1861 (*59*). Although this method was employed for several years, it was soon abandoned because of the high cost of filling and emptying the barrels and returning the empties; also, because of the circular shape of barrels, a large space was required in proportion to the volume of oil carried. The first all-steel tanker of modern design was built in 1888 (*5*).

During World War II the tendency was to build speedy oil tankers to enable them to operate independently of convoys, which required low average speeds. The United States adopted the "T-2" tanker, which had a capacity of 138,000 barrels, was an all-welded vessel, and was powered with turboelectric machinery, rated at 6,000 shaft-horsepower. Since the war the tendency has been to build tankers with greater capacities; because of their increased drafts, these ships can operate from fewer ports.

BARGES

One of the earliest solutions of the transportation problems arising from discovery of oil in Pennsylvania was the use of flatboats on which oil in wooden barrels was floated down the Pennsylvania streams. The flatboats were later replaced by scows that could carry oil in their hulls. From these early beginnings modern barges have been developed that provide economical and efficient means for transporting petroleum and petroleum products in bulk. The Mississippi River and its tributaries, the Atlantic coastal waters, western rivers, the Great Lakes, and the Pacific coastal waters serve as principal waterways for the movement of oil by barges. All petroleum shipments that move by inland-type tank vessels are identified as inland-waterway shipments, even though in some instances a portion of the movement may be made over open water. The total tonnage of crude oil and refined-petroleum products transported by inland and coastal waterways was 14,592,000 net tons in 1952 compared to 11,551,000 tons in 1949.

RAILROAD TANK CARS

Early in 1865 Amos Densmore (*5*) built 2 wooden tanks on the ends of an ordinary railroad flatcar, filled each tank with 45 barrels of crude oil, and shipped them to New York City. This mode of transportation proved so successful that several hundred such cars were placed in operation shortly thereafter. Later in the 1860's the ancestor of the present tank car appeared—a single, horizontal, cylindrical unit fitted with a dome which permitted oil expansion without injury to the tank. Initially the capacity was about 90 barrels of oil but some time later was increased to 100 barrels; and now the average is over 200 barrels.

Modern tank cars have kept pace with the development of new products in the petroleum industry and today are lined with aluminum, nickel, rubber, zinc, lithcote, glass, or one of many other materials to protect the products transported. Some are insulated to retain or to exclude heat. Some have coils to apply heat for liquefying products such as asphalt. Others, such as those used for transporting liquefied-petroleum gas, are constructed to withstand higher than normal pressures. Over 200 different types of tank cars have been designed for special purposes.

Tank cars range in capacity from less than 4,000 gallons to 16,000; they contain 1 to 6 compartments and vary in cost from $3,000 to $18,000 a car.

Railroad tank cars rank third in importance (*70*) as bulk carriers of crude petroleum and play a much more important role in distributing refined products from refineries to the many inland distributing centers. Petroleum and its products make up 3.5 percent of all railroad tonnage and 4 percent of all railroad revenue in the United States. These data were submitted by Class I railroads to the Association of American Railroads. Although less than 3 percent of the crude production is moved by rail, in some regions where pipeline facilities are not well developed—as in the Rocky Mountain region—rail transportation of crude oil is an essential phase of the petroleum industry. Although tank-car transportation has the disadvantage of higher costs than either marine or pipeline transportation, under certain conditions other means of transportation are neither economical nor practical.

MOTORTRUCKS

Tank trucks were widely used in marketing before they found a place in bulk transportation (*54*, *70*). Their use was generally limited to local distribution, and the tanks were designed with small compartments and for bucket deliveries. These trucks, whose capacity ranged from 500 to 1,500 gallons, made deliveries to service stations, oil-heated homes, farmers, and large consumers within a radius of 100 miles. In the early 1930's large bulk trucks and tractor semitrailers with a capacity of 4,000 to 5,000 gallons were introduced. It was also about this time that the first products pipelines were built. Pipeline-plus-truck transport or barge-plus-truck transport afforded attractive economies as compared with long tank-car hauls. A few years before World War II, truck transports began to make a noticeable dent in the medium-distance hauling of gasoline and heating oil. Regular runs up to 200 miles and a few even longer were not unusual.

Truck transportation of petroleum products received a tremendous stimulus during World War II as means were sought to free railway tank cars for long-distance movements and to release tankers for ocean shipments. Substitution of tank trucks for tank cars in short-haul service reached a peak in 1943, when approximately 18,000 tank cars were released and placed in long-haul operations.

Following World War II tank cars never regained their relative prewar position in transporting petroleum products. From their wartime experience, shippers realized to a greater extent that tank trucks have a definite economic place in the short-haul movement of petroleum and its products. Short-haul trucks range in size from 750 to 2,000 gallons, but for long-distance movements some truck-and-trailer combinations, where State laws permit, have reached a capacity of more than 8,000 gallons; almost the capacity of the average tank car.

Although tank trucks are used by the producers of petroleum only to a limited extent, the more than 125,000 trucks and tractor-trailer units in use represent an investment of about $375 million. Many companies operate fleets of over 1,000 tank trucks. No data are available on the petroleum tonnage hauled by these carriers.

NATURAL-GAS TRANSPORTATION BY PIPELINE

Like other major industries, the business of transporting natural gas from areas of production to consuming markets has grown from small beginnings. The first iron pipeline of appreciable length for gas transportation was laid in 1872 (*27*). It was 2 inches in inside diameter and 5½ miles in length and extended from a newly discovered well at Newton to Titusville, Pa.

In the period between 1872 and 1890 pipelines laid for transporting natural gas were relatively short and did not exceed 8 inches in diameter. The pipe was made of wrought iron with the joints connected by screw couplings and was operated at a normal pressure of about 80 pounds per square inch (p. s. i.)

High-pressure gas transmission was first used in 1891 by the Indiana Natural Gas & Oil Co., when 2 parallel lines, each 8 inches in diameter, were constructed to carry gas 120 miles from natural-gas fields in northern Indiana to Chicago, Ill., at an initial pressure of 525 p. s. i.

Development of natural-gas transmission lines in the midcontinent area was begun soon after the turn of the century. The first important natural-gas transportation company, the Kansas Natural Gas Co., was formed in 1904 and constructed a 16-inch pipeline from the Allen, Neosho, and Wilson County gas fields in Kansas to Kansas City and Joplin, Mo. With construction of additional transmission lines, Kansas City, Mo., was completely converted to natural gas by 1907. In 1906 the Wichita Natural Gas Co. laid a 12-inch pipeline from the Montgomery County field in southeastern Kansas to Wichita, Newton, and Hutchinson, Kans. As this field was short lived, in 1910 new pipelines were extended to the Hogshooter field in northeastern Oklahoma. With discovery of the latter field, the Quapaw Gas Co. constructed a 16-inch pipeline across northern Oklahoma to Joplin, Mo., where it served gas to the lead and zinc industry and to nearby towns.

The early experiences of these companies soon indicated that, if a continuous and reliable gas service was to be assured, the industry not only would have to develop transmission facilities but also acquire adequate gas reserves to meet the market demands for an extended period to protect pipeline investments. With a plan to combine production, transmission, and distribution facilities with adequate reserves of its own, eastern capital entered the midcontinent natural-gas business in 1912. Immediately, integrated natural-gas systems began to be developed, such as the Oklahoma Natural Gas Co.

in central Oklahoma, the Lone Star Gas Co. in north central Texas and southern Oklahoma, and the Arkansas Natural Gas Co. in Arkansas and western Louisiana. The principal cities served by these systems were Oklahoma City and Tulsa, Okla., Dallas and Fort Worth, Tex., Shreveport, La., and Little Rock, Ark.

Meanwhile, further developments were taking place in California and in the Appalachian area. By 1925 pipelines of varying lengths up to 300 miles had been built to serve 3,500,000 customers in 23 States with over 1 trillion cubic feet of natural gas a year. At that time, two outstanding natural-gas reserves—the Monroe field in northern Louisiana and the Amarillo (Panhandle) field in Texas—were being developed. Technological improvements in high-pressure transmission had been improved to such an extent that studies were undertaken to determine the feasibility of transmitting gas from these prolific fields to new and more distant markets. Rapid expansion of natural-gas transportation lines was soon underway.

By 1934 approximately 150,000 miles of field, transmission, and distribution lines—some as large as 24 inches in diameter and 1,200 miles in length—had been constructed to market gas in 32 States. As of January 1, 1954, the total length of such lines had grown to 394,000 miles, of which 125,000 miles was classed as transmission lines.

Most of the long-distance natural-gas transportation systems originate in the great producing fields of the midcontinent and gulf coast areas, which include Texas, Louisiana, Mississippi, Oklahoma, Kansas, and New Mexico. From East Texas, the Texas gulf coast, Louisiana, and Mississippi, these lines run predominantly northeasterly to serve the east coast, New England, and the Midwest east of the Mississippi River. In general lines originating in the Texas Panhandle, Oklahoma, and Kansas serve the Midwest west of the Mississippi River. From West Texas and New Mexico gas is piped to Arizona and California.

Thus, the relatively abundant natural-gas reserves of the Southwest are increasingly becoming the major source of supply for distant markets in many different parts of the Nation.

Numerous projects have been initiated since World War II for extending and enlarging long-distance pipelines designed to meet increased demands for natural gas in both new and existing markets. The Transcontinental Gas Pipeline Corp. pipeline extending far east of the usual pipeline routes serves a number of utility companies in the South and terminates at New York City. The systems of the Texas Eastern Transmission Corp., and the Tennessee Gas Transmission Co. were extended to serve New England. The Michigan-Wisconsin Pipeline Co. was constructed from the Texas Panhandle to serve customers on both sides of Lake Michigan. California is served by the El Paso Natural Gas Co. lines, which originate in West Texas and follow two separate routes to California. Plans are under discussion to bring natural gas from both the San Juan Basin area of New Mexico and from Canada to the Pacific Northwest.

A quantitative measure of the postwar expansion of the gas-transmission industry is the expenditures made for new facilities. At the end of 1945 the gross plant investment of all natural-gas-pipeline companies reporting to the Federal Power Commission was $2,016,000,000. This sum included gas production, transmission, and distribution. In the following 8 years expenditures by the gas industry for natural-gas-transmission facilities alone were $4,222 million—over twice the total investment in all facilities at the end of 1945.

Pipeline-construction costs vary with the capacity and distance traversed and also with the locality and time of construction (*69*). A study based on average conditions in the United States about 1949 indicated a construction cost of about $33,000 per mile for 16-inch gas-transmission lines and $60,000 for 24-inch pipelines. The estimates include the cost of compression, metering and regulating stations, a communication system, the pipeline, and its installation. Assuming that a 16-inch line delivers 29,200,000,000 cu. ft. annually when operated at full capacity and that a 24-inch line delivers 87,600,000,000 cu. ft. the average transmission cost for 16-inch lines is estimated at $0.00021 per 1,000 cu. ft. per mile; for 24-inch lines, $0.00014 per 1,000 cu. ft. per mile. These costs include operating expenses, taxes, and depreciation on the pipeline and transmission facilities, but no provision has been made for profits. A proposed 34-inch line, designed to transmit gas from southwest Texas and southeast New Mexico to San Francisco, Calif., a distance of 1,484 miles, in 1948 was estimated to cost $146 million or nearly $100,000 per mile.

The gas industry is planning to spend $4 billion during the period 1955 through 1957, the major part of which will be used in expanding natural-gas facilities, according to the 1954 year-end report of F. M. Banks, president of the American Gas Association (*51*). A 1,200-mile, 30-inch gas-transmission line, proposed for construction from Texas to Michigan, has been estimated to cost about $130 million.

Gas-distribution companies are concerned with acquisition of franchises, operation of municipal plants, securing fair rates, and meeting the ultimate consumers' demands. This involves laying and maintaining city mains, making connections, reading the consumers' meters, giving proper service, and promoting consumption.

One-half of the households in the United States served by gas utilities in 1953 were heated by natural gas. In 1932 there were 15,532,000 gas users in the United States; this was increased to 26,708,000 in 1953. In contrast, 5,499,000 used straight natural gas in 1932 compared with 19,960,000 in 1953.

Natural gas was marketed to domestic, commercial, and industrial customers in 42 States and the District of Columbia in 1954, according to the Bureau of Mines.

The total volume of marketed natural gas was 8.4 trillion cubic feet in 1953, the most recent year for which complete data are available; this includes gas stored and that lost in transmission but excludes gas returned to natural formations in pressure maintenance and similar operations or wasted by venting and flaring. The following summary (table 10) shows the comparative volumes of natural gas used by major consumers in 1946 and 1953.

TABLE 10.—*Summary data on United States natural-gas consumption, 1953*

	Billion cubic feet		Percent of total	
	1946	1953	1946	1953
Industrial:				
Field use	898	1, 471	22. 4	18. 4
Carbon-black production	478	301	11. 9	3. 8
Petroleum refineries	332	559	8. 3	7. 0
Electric public utilities	307	1, 034	7. 7	13. 0
Portland-cement plants	58	115	1. 4	1. 4
Other industrial	1, 037	2, 283	25. 8	28. 6
Total industrial	3, 110	5, 763	77. 5	72. 2
Residential	661	1, 685	16. 5	21. 1
Commercial	242	531	6. 0	6. 7
Grand total	4, 013	7, 979	100. 0	100. 0

STORAGE OF PETROLEUM AND PETROLEUM PRODUCTS

An important factor in the economy of the petroleum industry is the aboveground storage reserve of crude and refined and semirefined products maintained partly by the producers, partly by the transportation and refining interests, and to a smaller degree by the marketing agencies. A storage reserve equivalent to several months' supply is necessary to provide for seasonal variations in market demand and interruptions in continuity of supply (*69*). In times of market depression large volumes of crude petroleum may accumulate because competitive production exceeds consumption. This practice is uneconomic because it tends to depress crude prices and requires additional capital outlay to provide additional storage facilities. Evaporation and occasional fire losses plus loss of interest on working capital tied up in surplus oil are other factors against surface storage of surplus oil. Progress has been made in crude-production control by proration and cooperative or unit operation of fields to avoid production in excess of market needs. Under normal conditions, it is better from every point of view to leave reserve petroleum in its belowground reservoirs provided by nature rather than to produce and store it in surface tanks and reservoirs.

Stocks of crude petroleum and petroleum products in storage have varied considerably in different periods. During years of industrial depression they tend to accumulate to undesirable levels, while in times of active demand for oil products—during the war years, for example—they may fall to such low levels as to create uneasiness concerning the continuity of supply. Normally, a reserve equal to 4 to 6 months' supply of all categories of oils in aboveground storage is considered a "comfortable" yet not excessive reserve. Reserves of crude oil in storage usually range from a 2 to 4 months' supply.

In 1929 reserve stocks in the United States reached the record high of 689 million barrels and in 1945 fell to a low of 463 million barrels. These reserves represented approximately 270 days of supply in 1929 and less than 100 days in 1945.

STORAGE OF PETROLEUM

A petroleum producer must provide storage facilities for the oil after it reaches the surface while awaiting shipment by a purchaser or transportation agency (*70*). Lease storage has been described previously. The following discussion of storage is concerned with the oil after it has been gaged by the purchasing company or transportation department and while in transit or waiting transmission at field terminals.

Large "tank farms" designed for this purpose often provide storage capacity for millions of barrels of oil. Vast sums have been expended to provide these storage facilities. The physical equipment must be properly designed and maintained to give maximum security against fire and explosion hazards and to reduce evaporation and leakage.

Storage containers for crude petroleum may conveniently be classified into two groups: (1) Tanks and (2) reservoirs. A wide variety of sizes and capacities of tanks made of wood, steel, or reinforced concrete is available. Reinforced-concrete reservoirs of large capacity cost less per unit volume but are suitable only for storing the less volatile oils. Steel tanks are used more commonly and generally are preferred for petroleum storage. Tanks as large as 2,500 barrels in capacity may be of the bolted variety, but larger tanks usually have riveted or welded joints throughout. To reduce evaporation losses from volatile oils, special tanks capable of withstanding high internal pressures are available. The usual form of oil-storage tank is one with a vertical cylindrical shell, a flat bottom, and a low conical roof. Such a form cannot resist much internal pressure without leakage of vapor. To minimize such evaporation losses from volatile oils, special tanks with flexible roof construction, known as "breather tanks," have been developed. Other tanks are spherical or spheroidal in order to resist high internal pressures. Another type of tank, known as a "watertop" tank, is equipped with a flat, depressed roof supporting a few inches of water, the purpose of which is to keep the vapor in the upper part of the tank cooler than it otherwise would be. In "floating-roof" tanks the roof floats on the oil and leaves no space for accumulation of vapors.

As standardized by the American Petroleum Institute, the capacity of riveted-steel tanks ranges from 240 to 134,000 barrels. Larger tanks of this type, some with a capacity as great as 178,000 barrels, have been built but are not considered "standard." A 240-barrel tank is 12 feet in diameter and 12 feet in height; in comparison, the 134,000-barrel tank is 144 feet in diameter and 46 feet in height.

Spherical and spheroidal tanks are used for storing the more volatile oils and hydrocarbon liquids. Tanks of these types can withstand internal pressures as high as 100 pounds per square inch.

Wood tanks have been used rather widely for storing crude petroleum in the older fields and in fields where corrosion causes rapid destruction of steel tankage. The capacity of a wood-stave tank seldom exceeds 2,000 barrels, and the 500- and 800-barrel sizes are most common. Either pine or redwood is used for staves.

EVAPORATION LOSSES

The loss of petroleum through evaporation while in storage is a factor of great economic importance and one that has not received the attention it deserves. Evaporation during storage is particularly noticeable in the lighter oils and may range from 1 to 25 percent, depending upon the volatility of the oil, temperature, wind velocity, the amount of agitation that the oil undergoes, and the length of time in storage. Bureau of Mines investigations (*77*) have shown that petroleum of 34° API gravity, stored in 500-barrel steel storage tanks at an average temperature of 78° F., may suffer a loss of 4.5 percent in bulk during a 10-day storage period. A loss of 6 percent in volume, accompanied by a reduction of 2.46° API gravity, resulted from filling a storage tank over a period of 5 days, with 40.7° API oil at an average temperature of 37° F., with the tank hatch open. Although normally the losses are lower, evaporation still is an important consideration, because the lighter and more valuable gasoline-forming constituents are lost. The evaporation loss of even 1 percent of the contents of a 55,000-barrel tank of petroleum represents a loss of 550 barrels, which, at current refinery-gasoline prices, represents a loss of over $2,000.

Petroleum probably undergoes its greatest evaporation loss during the first few days of storage. Agitation of the oil during its admission to the tank doubtless contributes largely to this result. Once the tank is filled, the evaporation rate will be reduced somewhat. There is, however, continual vertical movement in stored oil; the surface oil, being reduced in gravity by loss of its lighter constituents, sinks and is replaced by lighter oil from lower levels.

Control of storage temperature offers the most direct means of minimizing evaporation loss. Experiments conducted by the Bureau of Mines have shown that the temperature of oil stored in tanks coated with light-color paints is 5° to 10° F. lower than that of oil in tanks painted with dark colors. In a series of experiments, a light naphtha suffered a loss of 9 percent in a black tank, whereas, under identical conditions, the same product stored in a white tank lost only 4 percent. Oil in pale-blue, cream, and tan tanks lost 5 percent; in battleship gray tanks, 6 percent; and in red and green tanks, 8 percent (*60*).

CORROSION

Corrosion of steel tanks may result from: (1) Direct oxidation in the presence of air and moisture; (2) active electrolytes in the water settling on the bottom of the tank or in the foundation soil under the tank; (3) sulfur compounds in the produced oil or water (particularly hydrogen sulfide); and (4) galvanic corrosion, an electrochemical effect, resulting from a difference of potential between various portions of the tank or substances in contact with the tank.

Exterior corrosion is minimized by occasionally painting the tank shell and roof. Customarily the bottom is painted inside and outside before it is lowered to the foundation grade. The exterior of the shell and roof is painted after construction is completed.

The effectiveness of ordinary paint is limited in preventing corrosion where severe chemical action or galvanic effects are involved. More complete protection is afforded the upper surface of the tank bottom by covering it with a layer of hot asphalt, neat cement, or concrete 1 or 2 inches thick. Corrosive waters accumulating in the bottoms of storage tanks should be drained frequently and the residue diluted with fresh water. Acidic waters may be neutralized by the addition of sodium carbonate or other alkalies.

Where galvanic action is troublesome, cathodic protection may be employed. An alternative method involves bars of zinc in the water layer a few inches above the bottom of the tank on nonconductive supports with electrical connections to the tank roof. Zinc is electropositive to iron and steel; and, when used in this way, the galvanic activity is absorbed by the zinc rather than by the metal of the tank. Careful selection of tank metals, rivets, fittings, and other material in the tank bottom and shell to avoid marked differences in chemical character will minimize self-induced galvanic effects.

STORAGE OF NATURAL GAS

The storage of natural gas at or near major consuming centers to meet extreme peak loads of short duration and also as a safeguard against equipment breakdowns has long been practiced by gas utilities (*27*). Aboveground low-pressure holders have been in common use by the manufactured-gas industry for many years but seldom have been used for natural gas. The terminal storage of natural gas in buried tanks at pressures ranging up to 2,200 p. s. i. was developed in 1946. Owing to the rather small capacity of such gas holders, storage by this means is effective primarily in meeting hourly peak demands and, to some extent, extreme daily peak requirements. This method is not practicable, however, in providing for sustained peaks of heavy seasonal (winter) loads. Manufactured gas and LP-gas are being employed to an increasing extent to meet peak demands. The natural-gas industry has also experimented recently with the storage of natural gas in liquid form, but general adoption of this method met an unfortunate setback several years ago in consequence of explosion of the first major installation in Cleveland, Ohio.

A more widespread practice in the natural-gas industry has been the use of underground storage, employing depleted or partly depleted gas or oil reserves.

HISTORY OF UNDERGROUND NATURAL-GAS STORAGE

Natural gas was first stored underground in Welland County, Ontario Province, Canada, in 1915 (*75*). The following year natural gas was first stored underground successfully in the United States in the Zoar field near Buffalo, N. Y. Only 3 additional storage fields were in use in the United States in 1930, but by 1940 the number had increased to over 30, and since 1940 growth has been more rapid. At least 80 fields, with a combined storage capacity of 440,828 million cubic feet, were utilized by 1949. In 1953, 167 storage projects having an estimated ultimate capacity of 1,735 billion cubic feet were in operation (*4*). By far the greatest number of these projects are in the Appalachian area, in West Virginia, Pennsylvania, New York, Ohio, and Kentucky. There are likewise large underground storage projects in Michigan, Illinois, the midcontinent area, and California.

Underground storage fields offer two major advantages over other types of natural-gas storage; their capacity is greater, and they are less costly. Underground will not entirely replace other types of storage because there is a definite need for gas holders, steel-pipe storage, and other types with less capacity; but, if larger capacity is required, underground storage is the answer, provided (of course) that a suitable underground reservoir can be found.

The estimated capital costs for various types of storage are listed in table 11. The weighted average cost of storing gas underground, referred to in the tabulation, is $0.27 per 1,000 cubic feet. The average capacity of the 13 fields on which the estimate of cost for underground storage was made is almost 5 billion

TABLE 11.—*Comparison of estimated capital costs for various types of storage*

Type of storage:	*Cost per 1,000 cubic feet*
Sphere (80 p. s. i.)	$227.00
Steel pipe (2,240 p. s. i.)	207.00
Gas holder (low pressure)	156.00
Steel pipe (1,560 p. s. i.)	48.00
Natural gas in propane	32.00
Liquefaction	13.00
Salt cavity storage[1]	1.42
Underground storage (water sand)	.53
Underground storage (depleted wells—13 fields)	0.02–0.50

[1] Assuming excavation of 1 million cubic feet cavity and disposal of salt brine in deep brine strata.

cubic feet. In recent years, however, the trend has been to larger reservoirs, such as the Oakford storage field near Jeannette, Pa., of the New York State Natural Gas Corp. This field is reported to have a capacity exceeding 100 billion cubic feet.

In addition to the essential function of underground storage in helping to balance the demands upon transmission facilities, other desirable features of such operations may be specifically noted (*27*):

(1) The transmission system's total annual deliverability through the storage cycle of input and withdrawal is increased by maintaining a high load factor throughout the year.

(2) A smaller amount of transmission capacity is required to meet the total firm demands of consumers; consequently, unless the cost incident to storage operations is unfavorable, the average cost of assured supplies of delivered gas is lowered.

(3) Storage provides greater assurance of continuous service to all consumers, without resorting to special contracts providing interruptible service to some industrial customers. Efforts to increase the system load factor by attaching interruptible loads are likewise discouraged, in that large volumes of gas are required to replenish the storage reservoirs during offpeak periods.

(4) Supplies of gas that have been firmed up by terminal storage near the market are worth more to those customers who might otherwise be subjected to curtailment during emergencies or peak periods of market demands, even though their contracts specify a firm or continuous service.

(5) The offseason storage of gas under high pressure is not only the accumulation of the volume of the gas itself but the concentration of pressure energy as well, insofar as the reservoir pressure can be built up enough to deliver gas into the transmission system during the withdrawal phase without the need for additional compression. Economy in the use of transmission-line compressors is thus achieved by their operation during offpeak periods to develop the necessary driving energy in the reservoir for use at times of peak demand.

(6) The increased flexibility made possible by storage operations is particularly desirable where gas is being produced from many stripper wells that are nearly depleted. In such instances, wells of low pressure and deliverability can be produced continuously rather than drawn upon heavily for short periods—a practice that is often uneconomical and may be detrimental to wells producing gas with a natural water drive.

(7) If underground storage is available, it may also be possible to prevent the waste of gas produced with oil. Since the peak period of oil production usually occurs during the offseason for heavy gas consumption, underground storage facilities make it possible to compensate for these seasonal differences without blowing casinghead gas to the air.

MANUFACTURED GAS OR LIQUEFIED PETROLEUM GAS (LP-GAS) AS STANDBY

In many instances of conversion from manufactured to natural gas, the manufactured-gas-generating facilities have been retained for years by the utility company. These facilities serve as standby to meet emergencies and, by providing gas for peak-load requirements, they help to stabilize the demand upon natural-gas transmission lines. Among the reasons for this practice has been that the equipment, already installed, requires little or no additional investment, and with some modifications it could be made to deliver gas of the required British thermal unit content to meet peak demands.

Since natural-gas shortages have been experienced in many consuming areas, distribution companies have augmented their gas supply with liquefied petroleum gas (LP-gas). Such installations, using propane, butane, or proportions of both, have been introduced, for example, in the Cincinnati, Detroit, Memphis, and Pittsburgh areas, as well as in many smaller communities.

The advantage of a propane-butane installation for meeting occasional sharp peaks is the relatively small capital investment required. The installation consists of necessary storage facilities and certain essential automatic equipment for converting the liquid to a gas interchangeable with that normally used. The cost of the fuel is relatively high, although this may not be prohibitive when it is considered that any method of providing a peak supply of dependable gas fuel involves extra costs that should not be compared with the average costs for a continuous year-round supply.

UNDERGROUND STORAGE OF HYDROCARBON LIQUIDS

A public demonstration of the first man-made underground storage reservoir for propane and butane was held at the Keystone field, Kermit, Tex., in 1950. The underground storage reservoir was dissolved from a salt bed by injecting fresh water from the surface. This massive salt formation of varying thickness and depth is extensive in the Permian Basin area of West Texas and southeast New Mexico and is overlain by an impervious bed of anhydrite. In the Keystone field the top of the salt formation is approximately 1,150 feet below the surface and approximately 600 feet thick. The cavern was formed by circulating fresh water from the surface through the tubing in a well drilled through 590 feet of salt to a total depth of 1,750 feet. In other areas of the country underground LP-gas storage reservoirs have been successfully excavated.

PRINCIPLES OF PETROLEUM REFINING

Because of its chemical constitution, crude petroleum has few uses in the raw or unrefined state. Crude petroleum consists predominantly of compounds of carbon and hydrogen, called hydrocarbons, ranging from highly volatile constituents that vaporize at or near atmospheric temperature to solids or extremely viscous fluids that will not boil but will decompose at approximately 800° F. and higher temperatures.

The wide range of boiling points of the constituents of petroleum is used to separate it into usable products. Distillation—the first process used for refining petroleum—still is the basic operation in modern refineries.

DISTILLATION

Gildemeister and Hoffman (*30*) have provided an interesting illustrated history of early methods of distillation and the apparatus used. According to these authors, the first definite statement found in ancient writings which indicates a kind of primitive distillation is the method for obtaining oil of cedar (or turpentine) which is described in the writings of Herodotus, Dioscorides, and Pliny. This oil is said to have been obtained from the oleoresin by boiling with water in an open earthen kettle. The oil either collected at the surface of the water and was removed, or its vapors were condensed in layers of wool spread over sticks laid crosswise over the kettle. The wool when saturated was replaced from time to time by fresh portions and the oil expressed by hand.

Gildemeister's treatise includes drawings of several ancient and medieval forms of distilling apparatus and shows for contrast a "giant distilling apparatus of 60,000 liters capacity" (*30*).

In the early days of petroleum refining crude oil was charged to batch-operated "cheesebox" stills—squat vertical, steel cylinders with a flat bottom and a wide-angle cone top or flat-curved roof to which a vapor-outlet pipe was attached at the center (*8*). Later, horizontal cylindrical "shell" stills were adopted and are still being used. These stills typically have a charging capacity of 1,000 barrels of oil (42,000 gallons), and the vapor-outlet pipe is at one end of the cylindrical surface.

With the increasing demand for petroleum products, the mechanical equipment used in refineries was improved. The batch-operated cheesebox and shell stills were replaced by continuously operated "batteries" of stills. A battery consisted of a series of interconnected shell stills through which the oil charge was pumped continuously. Each succeeding still was heated to a higher temperature than the one preceding it, so that succeedingly heavier, higher boiling point material was distilled from the oil as it passed down the series until a heavy residuum was formed and removed from the last still of the battery. Although the type of products removed from each still varied in different refineries, depending upon the character of the crude oil being processed and the products desired, a typical range of products from a five-still battery was: Still 1, light gasoline; still 2, heavy gasoline; still 3, kerosine; still 4, light gas oil; and still 5, heavy gas oil. The remainder of the crude oil was pumped as a liquid from the bottom of the fifth still.

If lubricants were being manufactured in the refinery the "bottoms" (residue from the fifth still described above) were run to one or more "steam-reducing" stills of the shell type, usually batch operated. The stills were heated as in the crude-oil battery, but in addition steam was bubbled through the hot oil, reducing the temperature necessary to vaporize the oil. Usually one or more light lubricating-oil distillates were obtained. Reducing stills using steam were employed also to convert the residue from asphalt-bearing crude oils to desired asphaltic products.

FRACTIONATION

The early batch stills provided a means of separating, from the crude oil, materials of different ranges of boiling points to produce usable products. The efficiency of these separations was poor because of "carryover" of heavy constituents in the lighter fractions. To correct this situation, fractionating towers (vertical cylindrical drums containing packing materials, such as stones, iron chains, and short pieces of iron pipe) were placed between the vapor outlets of the stills and the condensers. Towers containing baffles, perforated plates, or more efficient types of packing, such as "Lessing" or "Raschig" rings, also were used.

These fractionating towers exercised a scrubbing action upon the vapors passing through them, resulting in better separation of the distillates. Upon entering the bottom of the tower, the distilled vapors came into contact with the packing material, causing the entrained liquid to cling to the surface of the packing and the heaviest part of the vapor to condense. Because of the intimate contact between the uprising vapor and the downflowing liquid some fractionation of products was obtained, resulting in improvement in the quality of products.

PIPE STILLS

Early in the present century a different type of continuously operated equipment was developed for heating crude petroleum to distillation temperatures. Instead of large vessels, such as 1,000-barrel shell stills, oil was heated as it was being pumped through coils of tubes or pipes of 3 to 6 inches diameter arranged in furnaces. Several years before 1911, equipment of this type was operated in California to remove water and a distillate known as "tops" from crude petroleum (*74*). The tops were refined to gasoline and kerosine and the residual oil was sold as fuel. As time went on these "pipe retorts" or "tube stills" were improved to function at higher temperatures and either under pressure or vacuum. They have become the basic unit of petroleum-refining equipment. Crude oil may be separated into fractions ranging from gasoline to lubricating oils as distillates, and asphalt as residue in a single continuous operation by multicoil pipe stills known as "combination" units. A distillation unit with a rated capacity of 125,000 barrels of crude oil a day has been constructed in the refinery of Gulf Oil Corp. at Philadelphia, Pa.

CRACKING PROCESSES

Petroleum refining underwent marked changes following inception of the first "cracking" process in 1913. Before that time the quantity and quality of products obtained by refining methods depended entirely on the character of the crude oil, but cracking increased the yield of gasoline from a crude oil by decomposition of the heavier portions of the oil.

The first cracking stills were of the shell type and were batch-operated. To withstand higher pressures they were built of heavier metal than those used for normal distillation. In the first units gas oil was charged to the still, the vaporline was closed, and heat was applied until a predetermined pressure was attained. The vaporline valve then was opened enough to permit vapor to escape to the condenser while maintaining the pressure in the still. Heating of the still was continued until the contents were reduced to a heavy oil and a small quantity of coke.

The recovered distillate, called "cracked" or "pressure" distillate, was redistilled to produce gasoline and some heavier products. The noncondensable gas formed in the cracking reaction was blown to the air; in later processing the gas was used as refinery fuel.

Cracking is characterized frequently as pressure distillation, which is not strictly accurate. Cracking, as practiced by the petroleum industry, is in effect a combined decomposition and synthesis of a petroleum fraction under the influence of heat, pressure, and time.

In recent years petroleum-cracking processes have been divided into two classes, namely, "thermal" cracking and "catalytic" cracking, to distinguish between two types of processing. Thermal cracking is divided further into two types known as "liquid-phase" cracking and "vapor-phase" cracking. The procedure described in the foregoing was a primitive form of liquid-phase thermal cracking. Vapor-phase thermal cracking was a somewhat later development in which the charge was vaporized and heated in tubes set in furnaces.

In catalytic cracking materials called catalysts are important agents. Catalysis is defined by Webster as "acceleration of a reaction produced by a substance (called the catalyst) which may be recovered practically unchanged at the end of the reaction."

In thermal cracking the element of time perhaps is more important than in catalytic cracking. In either type of process the products are both lighter and heavier than the material charged to the "cracker" and range from hydrogen to coke. The proportional distribution of the types of compounds in the products of cracking usually differs markedly from the composition of the charge stock.

Batch operation of cracking stills was modified as early as 1919 by pumping gas oil or reduced crude oil into a still as the liquid content was depleted by the cracking reaction. The "makeup" was continued until so much coke had been deposited in the tubes and shell that the still had to be shut down for cleaning. Operating cycles of 72 hours were achieved by this procedure (Riverside Refinery, Louisville, Ky.).

CATALYTIC CRACKING

Catalytic cracking is similar to thermal cracking in that the chemical constitution of petroleum fractions is changed. A typical catalytic cracking operation consists of converting a medium- or high-boiling gas-oil fraction in vapor phase into a low-boiling gasoline having a motor octane number of 78 to 82, with residues of gas, heavy oil, and coke. Some licensers of catalytic cracking processes claim that their processes are not limited to naphtha as charging stock and that kerosine, light gas oil, heavy (vacuum) gas oil, and deasphalted reduced crude oil can be charged.

Catalysts are of two general types—natural and synthetic. The natural catalysts typically are chemically treated bentonitic clays. These catalysts are made in the form of pellets approximately 4 mm. (1/6 inch) in diameter and thickness. Synthetic catalysts generally are silica-alumina, silica-magnesia, or silica-zirconia-alumina. A recently developed catalytic process employs a platinum catalyst for reforming naphthas of low octane number into aromatic or naphthenic compounds. Synthetic catalysts are used as pellets or in the form of beads.

Operating conditions in catalytic cracking range from atmospheric pressure to 700 p. s. i. and higher and temperatures from 800° to 1,100° F.

ALKYLATION

Thermal and catalytic cracking processes produce quantities of light hydrocarbons, such as ethylene, propylene, butene, and isobutane. These products have high octane ratings, but their vapor pressures are too high to permit them to be used as components of gasoline, except a portion of the butanes. Alkylation is a method for converting these hydrocarbons into less volatile materials of high octane number by catalytic fusion of the lighter hydrocarbons. A complex mixture of hydrocarbons is produced, from which an aviation-gasoline blending stock is extracted that has high aviation octane rating, high heat of combustion per pound, good stability in storage, and low sulfur content. The supply of alkylate of suitable characteristics is the limiting factor in the manufacture of high-rated aviation gasoline.

Nearly all alkylation plants in the United States employ either concentrated sulfuric acid or hydrofluoric acid as a catalyst.

In the course of World War II a persisting demand developed for aromatic hydrocarbons, mainly benzene and toluene, but also cumene (isopropyl benzene), styrene (phenyl ethylene), and other compounds. The traditional source of these compounds (the coking of coal) was inadequate, and the petroleum industry was called upon to supply a large share of the demand. This was done in two general types of processes.

One type separates contained or inherent aromatic compounds from various refinery streams, usually by means of a selective solvent or mixture of solvents and by distillation of the petroleum fraction in the presence of the extractive solvent. The solvent usually used is phenol (carbolic acid), but in special instances other solvents such as cresylic acids may be preferable. In another process liquid sulfur dioxide is used as the solvent and as a refrigerant at −20° F. Another process employs a mixture of glycol and water as a solvent.

The second type is exemplified by the platforming process mentioned in the section on catalytic cracking. This method converts nonaromatic constituents of refinery streams into a product that contains aromatics either as separate compounds or in admixture, depending upon the charge stock selected. Mixtures of aromatics may be separated into pure compounds by one of the selective solvent processes.

Other examples of both types of process are in various stages of development and use(*56a*).

TREATMENT

Distillation under atmospheric pressure or vacuum, in which little decomposition occurs in the fractions that boil at low temperatures, is known as "straight" distillation. Gasoline made in this way is known as "straight-run" gasoline as disinguished from cracked gasoline. Gas oil separated by straight distillation sometimes is designated "virgin" gas oil to differentiate it from gas oil made by cracking.

Distillates, such as gasoline and kerosine, which have been separated from crude petroleum by straight distillation, have undergone little decomposition. Such distillates usually contain only naturally occurring, nonreactive compounds and therefore require little chemical treatment to fit them for use. Cracked distillates, having been subjected to severe temperatures and pressures, contain unstable compounds that react with oxygen and with each other to form undesirable gums. These distillates may also contain varying proportions of malodorous impurities, including sulfur compounds, which are removed to increase the response of the gasoline to tetraethyllead and to improve the odor and burning qualities of the products. Some lubricating-oil fractions require processing, which is classed as "treatment" in modern petroleum-refinery technology.

Sulfuric acid was the principal chemical reagent used in treating petroleum fractions in the early refining processes. Caustic soda solution and sodium plumbite ("doctor" solution) were next in importance. Sulfuric acid is used principally to remove gums and gum-forming compounds from light distillates. Doctor solution is used to convert odorous sulfur compounds (hydrogen sulfide and mercaptans) in light distillates, such as gasoline, naphthas, and kerosine, to insoluble lead sulfate and odorless oil-soluble disulfides.

A number of other processes for treating gasoline, naphthas, and kerosine have been developed and placed in use. Batch treatment still is being used, but continuous methods are employed where they are applicable. Continuous treating processes utilize such equipment as mixers, settling tanks, and water washers. The raw petroleum distillate enters the system at one end, and the treated product leaves at the other end.

SOLVENT REFINING

The use of solvent refining for manufacturing lubricating oils is another example of petroleum technology that has contributed markedly to the utility of petroleum. This process consists of mixing a petroleum fraction with another liquid that is not completely soluble in the petroleum fraction but exerts a preferential solubility for some petroleum constituents. When the mixed liquids are allowed to settle and separate into two layers, the petroleum fraction has been freed of certain undesired constituents by the solvent, and the petroleum fraction will have dissolved a small proportion of the solvent. The solvent is separated from each of the two mixtures by a simple, continuous method, such as distillation with steam, and reused in the solvent-refining process. The "raffinate" (usually the petroleum constituent that was not dissolved by the solvent) of this process has desirable characteristics, such as low rate of change of viscosity with temperature (high viscosity index) and good heat stability. Lubricating oils that have characteristics similar to those of lubricating oils made by the older processes from Pennsylvania Grade crude oil are manufactured in this manner from Midcontinent and California crude oils.

Solvent refining is not new. A process in which liquid sulfur dioxide at low temperature was used to extract aromatic constituents from the kerosine fraction of Rumanian petroleum was used commercially as early as 1914, but the general application of solvent refining to the manufacture of lubricating oils is of relatively recent date.

A variety of solvents and mixtures of solvents is used in petroleum refining. The principal application of solvent refining is the refining of unfinished lubricating-oil fractions. In addition, naphthas, kerosine distillate, diesel fuel oil, burning oils, and catalytic cracking stocks are refined and improved.

Solvents that are used singly include liquid sulfur dioxide, liquid propane, furfural, and phenol. Two well-established processes use two solvents. Other solvents are used in "one-company" or single-unit processes and in processes that have been developed experimentally but have not been operated on a commercial scale.

The first commercial unit was installed in 1927 and probably was the first commercial-scale solvent-refining unit for lubricating oils. At present approximately 70 commercial units are in operation.

The solvent-refining processes are characterized by the distinction that they are nondestructive. Except for comparatively small handling losses, the materials that emerge from the processes are the same in the aggregate as those that were charged to the treating plant. The solvents merely separate a petroleum fraction into two or more portions of different commercial value and desirability, but together nearly equal in volume to the volume of the original charged stock. The solvent is recovered and reused, only small losses having resulted from evaporation and washing to remove traces of solvent from the petroleum fractions.

BYPRODUCTS OF PETROLEUM AND NATURAL GAS

The major petroleum products, such as gasoline, kerosine, diesel fuel, and distillate fuel oil, are processed to conform to product specifications designed to insure the best performance of the various products in their end use. Some hydrocarbon constituents that exceed the specification ranges for these major products may be removed from the primary product during processing and utilized otherwise. As each of the standard petroleum fuels is composed of a mixture of hydrocarbons, there often develops a use for a particular hydrocarbon in the fuel, so that separation of the hydrocarbon from the fuel results in more advantageous usage.

From the very early days of crude-oil production some effort was made to recover liquids contained in the casinghead gas. The casinghead gasoline recovered was blended into gasoline for motor fuel. However, as motor fuel would not tolerate an excessive vapor pressure, propane and part of the butane were not recovered from the casinghead gas. Markets now have developed for these materials outside the gasoline range, and liquefied propane and butane satisfy important fuel needs in this country.

Sulfur, in the form of hydrogen sulfide, often found in natural gas, prevents transportation and use of the gas as a commercial fuel. Processes have been developed for profitable recovery of the sulfur from such gas, thus increasing the supply of natural gas suitable for fuel use.

In making kerosine it is often necessary to remove from the kerosine fraction those hydrocarbons that would cause the finished product to burn with a smoky flame. In making gasoline it is common practice to remove undesirable constituents by acid treating and by newer, improved extraction processes. These undesirable components generally are incorporated in other useful products at a profit.

An outstanding and well-known product derived from petroleum is synthetic rubber. In World War II the United States was denied access to natural rubber from the Far East. Through the closest cooperation between Government and the rubber and petroleum industries, synthetic rubber was made available in the necessary quantities to permit the United States and its allies to prosecute World War II successfully. Synthetic rubber was and is made from light refinery-gas fractions and isobutane, plus styrene made from benzene. During the war some synthetic rubber components came from grain alcohol, but this costlier process gradually was replaced by the petroleum process. Today synthetic rubber has made possible better, longer lasting tires and tubes, as well as the new tubeless tires, and its lowered price has placed a price ceiling on natural rubber. Carbon black, another product of both natural gas and liquid petroleum fractions, is a vital part of the finished synthetic rubber product used in tires. For many years carbon black was produced almost entirely from sour natural gas, which had no other market. Today better carbon blacks for certain purposes are produced from some of the heavy, less desirable petroleum fractions, in addition to the improved grades still produced from natural gas.

Synthetic detergents—"soapless" soaps, developed from an unwanted petroleum residue—now represent over 50 percent of the total soap sales in this country. Sulfuric acid has been in common use for treating petroleum fractions to improve quality since the early refinery operations. The undesirable material removed in the treatment reacts with the sulfuric acid to form a crude detergent. Further processing of the crude detergent yields synthetic soap of excellent quality.

These examples of byproduct development could be continued almost endlessly, and undoubtedly further developments of new and improved byproducts will continue in the future. With petroleum containing the basic compounds for the production of any organic compound, it is evident that the potential number of byproducts is unlimited. Realization of this potential will be governed by the need for these compounds and by the cost of such byproducts from petroleum compared to the cost from other sources.

RELATIONSHIPS TO OTHER SOURCES OF ENERGY

As a source of energy, petroleum products compete with nonpetroleum fuels for some uses and have a virtual monopoly for others.

Gasoline has had no competitor for family passenger cars since invention of the automobile. Other sources of energy, such as steam and electricity, were tried but were unable to compete with gasoline on a basis of performance and economics. For certain classes of trucking there is now competition among three petroleum fuels, gasoline, diesel oil, and LP-gases. Diesel locomotives have performed so well on American railroads that, except for specialized conditions, there may soon be almost complete conversion to this type. Petroleum and other fuels are on a more equal basis where fuel is used for domestic and industrial heating. A price advantage for a particular fuel may outweigh the convenience advantage of the handling and use of other fuels.

In railroad transportation it appears possible that gas turbines may be used in locomotives for certain operations. Both oil and coal are being used experimentally in gas-turbine-powered locomotives, and the ultimate choice of fuel will be determined by the costs of operation.

Petroleum fuels in 1953 supplied over 60 percent of the total energy produced by mineral fuels and waterpower in the United States (*14, 1953 ed.*). During the depression of the 1930's, when total energy requirements declined, the demand for petroleum was reduced only slightly, while the overall decline was shown principally in the decrease in coal output. During World War II, when all fuels were in high demand, coal consumption increased even more than oil. In the immediate postwar years, when oil was in short supply and exports were high, coal held its own. Since 1949, when petroleum again became plentiful, the trend in coal consumption has again turned downward.

Waterpower has for over 50 years contributed 3 to 4 percent of our energy supply. Wood used as fuel appears to have become a decreasing factor; in 1953 it contributed only about 2 percent of the total. Unquestionably, future developments in atomic research will change the comparative energy demands of the United States and of the world. However, the practical application of atomic energy to present energy demands still is in the experimental stage, and its significance as yet cannot be estimated.

PETROLEUM AND NATURAL-GAS RESERVES

The complexity of the concept of reserves has led to diverse classifications of reserves, with a multiplicity of terms to describe our mineral and fuel resources: Original reserves; measured, indicated, and inferred reserves; proved, probable, and possible reserves; potential reserves; minable and recoverable reserves. These are among the terms most frequently employed. All have different meanings, and the computation of each results in a different quantity.

The computed size of reserves depends on many factors, of which some are independent of each other and others interrelated in varying ways. The most important of these include the distribution of the material in the earth's crust, its accessibility, techniques of recovery, processing and utilization, and the economics of price, transportation, and substitution.

These factors determine the approach to be taken to the concept of reserves. Petroleum reserves are never static and unchangeable. They are dynamically linked with the technological progress of mankind and continually are subject to advances of scientific development in discovery, recoverability, and utilization of our mineral-fuel resources. Any discussion of the subject of reserves must be founded on clear understanding of the meaning of the various categories of reserves.

Estimates of the total quantity of the various elements or minerals present in the earth have been made, but these are not pertinent to our understanding of reserves. Although exceedingly voluminous in the aggregate, the substances may be disseminated so minutely through the rocks or buried so deeply that they cannot be considered recoverable for use in the foreseeable future. Only those quantities of a substance for which some evidence is available to indicate concentrations of potential economic value, at depths in the earth's crust believed to be accessible for exploitation, may be regarded as reserves. These quantities—total reserves known to be present in the ground at present—are our basic reserves. These basic reserves can be estimated or inferred with varying degrees of accuracy, depending on the nature of the occurrence and the character and extent of the available geologic data. Certain fundamental assumptions are, of course, required before basic reserves can be computed. Assumptions must be made for the size and thickness of the deposits to be included, their grade, and their depth.

The quantities of basic reserves thus calculated are independent of the percentage of recovery attainable by current methods and the costs of extraction that may or may not permit their recovery at a profit. The quantities furnish the basic data to which economic and technologic considerations can be applied.

Computations of basic reserves, then, are subject to revision and change in three principal ways. But when compared with "recoverable reserves" they are relatively stable. Obviously, not all reserves computed as "basic reserves" can normally be extracted for use. Depending on conditions and on the substance involved, recoverability of the basic reserve may range from a very low percentage to almost 100 percent. Recoverability depends on the efficiency of extraction methods or production techniques, on restrictions imposed on underground removal methods by the need for protecting surface or subsurface installations, and on profit incentives for production. Recoverability, and hence "recoverable reserves," thus differ from date to date and change almost constantly. In the strictest sense, a mineral substance does not become a "reserve" until it is "recoverable."

In any attempt to make quantitative comparisons of fuel reserves, the widely varying modes of occurrence of mineral fuels and the different methods of extraction produce great differences in reserve estimates. Virtually no data on the ultimately expectable size of basic reserves of petroleum and natural gas have been assembled. The basic reserves of these fuels are subject to continuing large, relatively unpredictable increases through new discoveries, and estimates of petroleum and natural gas yet to be discovered cannot be more than approximations.

HISTORY OF ESTIMATES OF PROVED PETROLEUM RESERVES

One of the early published estimates of the petroleum reserves in the United States was made in 1909 by David T. Day, Federal Geological Survey, for a report on the conservation of mineral resources for the National Conservation Commission. Day estimated a minimum recovery of 10 billion barrels of petroleum and a maximum recovery of 24½ billion barrels from the fields known at that time. He stated that the amount of oil obtainable from the known fields was largely conjecture, as it could only be based upon what the fields had already yielded and upon the thickness and relative porosity of the sands; further, that estimates of different authorities must vary between wide limits. From 1859, when the Drake well was drilled at Titusville, Pa., to the end of 1909, when Day made his estimate, the total oil production slightly exceeded 1 billion barrels.

In 1921 the Federal Geological Survey and the American Association of Petroleum Geologists estimated that the recoverable oil "in sight" was 5 billion barrels.

The first estimate of proved reserves by the American Petroleum Institute was prepared in 1925. The estimated proved reserve of 5,321,427,000 barrels, as of December 31, 1924, was based upon existing wells, the acreage proved by those wells to be oil bearing, and the recoverability by the current methods of production.

The Federal Oil Conservation Board in September 1926 stated that, according to most recent estimates at that time, the recoverable reserves in producing and proved fields amounted to 4½ billion barrels of oil; and in its fifth report (October 1932) the Board stated that reserves by later discoveries raised the known recoverable reserves to 10 billion barrels.

The reserves of petroleum recoverable by current methods from known fields, as of December 31, 1933, were estimated to be 13,360 million barrels by Hale B. Soyster, Federal Geological Survey, who prepared the estimate for the Special Subcommittee on Petroleum Investigations of the Committee on Interstate and

Foreign Commerce of the House of Representatives, 73d Congress. Proved reserve estimates also have been made in different years by Pratt, Garfias, McCoy, Marland, Osborn, Garfias and Whetsel, and Thomas. Estimates have also been published annually by World Oil (formerly Oil Weekly) and the Oil and Gas Journal for a number of years.

For 1934, 1936, and each succeeding year the American Petroleum Institute, through its Committee on Petroleum Reserves, has prepared estimates of proved petroleum reserves of the United States which represent only the amount of crude oil recoverable under current economic and existing operating conditions from fields now completely developed or drilled or sufficiently drilled and explored to permit reasonably accurate estimates of the ultimate recovery. The API estimates up to and including those of December 31, 1945, included cycle-plant and lease condensate but did not include all classes of natural-gas liquids. Beginning with the report for 1946, the estimates of the proved reserves included crude oil and the hydrocarbon liquids which are gaseous in the reservoir but may be reclaimed at the surface by condensation or absorption.

For each annual estimate of petroleum reserves, it must be emphasized that changes in the rate of consumption, production practices, prices of crude oil, and improved recovery techniques are factors that affect the estimated proved reserves or the ultimate recovery of petroleum from those reserves.

The discovery of new reserves by the industry is closely interrelated with the supply of crude oil, the demand for its products, and the payment of prices by the consumer that will compensate the cost of exploration. For example, the flood of oil that became available after the discovery of the East Texas field in 1930 and the decreased demand during the depression years in the early 1930's are reflected by the decreased discoveries of new reserves during the years immediately following 1930. Similarly, the large quantity of reserves added by discoveries in the Illinois Basin in 1937 and subsequent years affected the annual rate of discovery. During World War II the difficulty of obtaining steel for exploration and development is reflected in the lower-than-normal amount of reserves discovered in 1943.

Proved reserves of recoverable liquid hydrocarbons, which include crude oil, condensate, natural gasoline, and LP-gases, in the United States as of December 31, 1954, amounted to approximately 34.8 billion barrels. Of the quantity reported for 1954, 51.5 percent of the reserves is in Texas and 12.1 percent in California. Louisiana and Oklahoma rank next in importance, with 11.1 and 6.6 percent, respectively, of the total proved reserves. The estimated proved reserves of liquid hydrocarbons by States (table 12), prepared by the American Petroleum Institute and the American Gas Association as of December 31, 1954, are as follows (*3*):

TABLE 12.—*Estimated proved reserves of liquid hydrocarbons, December 31, 1954*

[Thousand barrels of 42 U. S. gallons]

Alabama [1]	15, 565
Arkansas	398, 339
California [2]	4, 218, 837
Colorado	341, 398
Illinois	680, 631
Indiana	67, 334
Kansas	1, 153, 646
Kentucky	96, 081
Louisiana [2]	3, 845, 697
Michigan	60, 533
Mississippi	472, 696
Montana	279, 573
Nebraska	40, 299
New Mexico	1, 145, 877
New York [1]	45, 831
North Dakota [1]	134, 349
Ohio	38, 211
Oklahoma	2, 289, 259
Pennsylvania	104, 718
Texas [2]	17, 910, 140
Utah	36, 461
West Virginia	71, 306
Wyoming	1, 354, 263
Miscellaneous [3]	4, 159
Total United States	34, 805, 203

[1] Crude oil only.
[2] Includes offshore reserves.
[3] Includes Alabama and North Dakota natural-gas liquids; Missouri, Nevada, South Dakota, Tennessee, and Virginia crude; and Florida crude and natural-gas liquids.

HISTORY OF ESTIMATES OF NATURAL-GAS RESERVES

Natural gas is a mixture of hydrocarbons and varying amounts of other gases. It is a companion to crude petroleum and was formed by similar processes. Both petroleum and natural gas are produced from similar and, at many places, common underground reservoirs. The gas and oil are found under pressure in pores, cracks, or other openings in reservoir rock. Natural gas may occur with or be dissolved in oil, or it may be found in accumulations that are not associated with oil. These usually are called "dry" gas fields. Because oil and gas, like water, are mobile and have different specific gravities, their process of accumulation in the reservoirs where they are now found has involved movement through permeable rock strata to the tops of domed structures or others sealed by overlying impervious rock strata. In a reservoir where both oil and gas are present, much or all of the gas may be dissolved in the oil, or a "gas cap" may occur above the oil in the upper part of the reservoir.

Natural gas was first marketed in the United States in 1821 at Fredonia, N. Y., where it was used for illuminating homes in the village. A small but growing demand for natural gas as a fuel and illuminant continued in New York, Pennsylvania, Ohio, and West Virginia until 1878, when the first large gas well in the United States was discovered near Murrysville, Pa. Expanded marketing of natural gas followed discovery of commercial production in Ohio, West Virginia, Illinois, Indiana, and Kentucky.

During the period from 1918 (when one of the early estimates of the reserve of natural gas was published) to 1953 the amount of natural gas marketed annually increased from 721 billion to over 9 trillion cubic feet. Utilization of natural gas reached an alltime high in 1953, and final figures will show that it was even greater in 1954 because of the continued extension of natural-gas-transmission systems, with resultant greater marketing facilities.

An early estimate of total recoverable and marketable reserves in both discovered and undiscovered fields of the United States was made by Eugene W. Shaw, valuation engineer in charge of natural gas, United States Internal Revenue Bureau and formerly geologist, Federal Geological Survey. In a paper presented at the annual meeting of the Natural Gas Association of America, Cleveland, Ohio, in May 1919 Shaw stated that the total recoverable and marketable reserves of natural gas in both discovered and undiscovered pools in the United States probably amounted to 5 to 15 trillion cubic feet.

Before systematic annual reporting of reserve estimates of natural gas by the American Gas Association in 1946, reports of gas reserves were made periodically by various individuals and during World War II by the Petroleum Administration for War.

As natural gas is commonly associated with petroleum, the areas having the most reserves of liquid hydrocarbons usually will have large reserves of natural gas. Texas, which has 51.5 percent of the Nation's reserves of liquid hydrocarbons, also has 49.7 percent of the natural-gas reserves of the United States. Louisiana, New Mexico, Kansas, and Oklahoma, in the order of magnitude of natural-gas reserves held, have almost 39 percent of the total estimated proved reserves. The proved reserves of natural gas in the United States at the beginning of 1955 amounted to approximately 211.7 trillion cubic feet. The estimated proved reserves of natural gas by States, prepared by the American Gas Association, as of December 31, 1954, are summarized in table 13 (*3*).

TABLE 13.—*Estimated proved recoverable reserves of natural gas in the United States, December 31, 1954*

[Million cubic feet—14.65 p. s. i. a., at 65° F.]

Arkansas	1, 165, 379
California	9, 026, 603
Colorado	1, 932, 913
Illinois	253, 756
Indiana	36, 049
Kansas	15, 758, 332
Kentucky	1, 286, 607
Louisiana [1]	36, 799, 986
Michigan	330, 100
Mississippi	2, 772, 683
Montana	723, 731
Nebraska	192, 946
New Mexico	17, 240, 669
New York	69, 362
Ohio	774, 741
Oklahoma	12, 396, 148
Pennsylvania	732, 163
Texas [1]	105, 129, 062
Utah	387, 375
West Virginia	1, 607, 290
Wyoming	2, 855, 071
Miscellaneous [2]	239, 766
Total	211, 710, 089

[1] Includes offshore reserves.
[2] Includes Alabama, Arizona, Florida, Maryland, Missouri, North Dakota, and Virginia.

ESTIMATION OF RESERVES

Estimation of the quantity of liquid or gaseous hydrocarbons in a reservoir that can be classed as a proved reserve involves knowledge of the physical characteristics of the reservoir rock and of the oil or gas contained therein. Among the geologic, engineering, and economic factors involved in estimating reserves are: The areal extent of the reservoir; thickness, porosity, and permeability of the reservoir rock; presence or absence of gas with the oil; existence of an effective water drive; reservoir pressure; production rate; and the economic life of the field. During development of a field, knowledge of the conditions that control the occurrence of the oil or gas is limited, and reserves generally may be estimated with reasonable accuracy only as development approaches completion. Preliminary estimates of the reserves of oil or gas in a field made during its development must be based upon few data, inferences regarding subsurface conditions, and comparisons with other fields producing from similar reservoirs. Such preliminary estimates may closely approximate the final estimates or they may be subject to considerable modification.

Estimates of the proved reserves of oil or gas of the United States include those for fields in all stages of development, ranging from newly discovered fields to fields approaching the end of their productive life. The American Petroleum Institute has adopted the practice of including in its estimate of oil reserves the quantity of oil that may be considered to be proved by current development; their estimates are revised in subsequent years as development progresses. Their practice is described in the following quotation from the report of proved reserves at the end of 1953 (*3*):

> The estimates in this report, as in all previous annual reports of this committee, refer solely to proved or blocked-out reserves. They include only oil and natural-gas liquids recoverable under existing economic and operating conditions.
>
> The estimates made for this report by your committee do not include:
>
> 1. Oil[3] under the unproved portions of partly developed fields.
> 2. Oil in untested prospects.
> 3. Oil that may be present in unknown prospects in regions believed to be generally favorable.
> 4. Oil that may become available by fluid injection methods from fields where such methods have not yet been applied.
> 5. Oil that may become available through chemical processing of natural gas.
> 6. Oil that can be made from oil shale, coal, or other substitute sources.
>
> Proved reserves are both drilled and undrilled. The proved drilled reserves, in any pool, include the oil estimated to be recoverable by the production systems now in operation, whether with or without fluid injection, and from the area actually drilled up on the spacing pattern in vogue in that pool. The proved undrilled reserves, in any pool, include reserves under undrilled spacing units which are so close, and so related, to the drilled units that there is every reasonable probability that they will produce when drilled.
>
> This committee uses the term "fluid injection" to include (1) what is commonly called "pressure maintenance"; (2) cycling; and (3) secondary recovery in its original sense, namely, fluid injection applied relatively late in the development history of a reservoir (pool) with the purpose of stimulating petroleum production after recovery by primary methods of flowing and pumping has approached an economic limit. The reserves which may become available as a result of fluid injection are regarded as proved only after thorough testing by a pilot plant, or better after operation of an installed fluid injection procedure, has actually demonstrated certainty of increased recovery. Then, and then only, are such reserves included in this committee's estimates of proved reserves.
>
> In the case of new discoveries, both of new fields and of new pools (pays, reservoirs) in old fields, which are seldom fully developed in the first year and in fact for several years thereafter, the estimates of proved reserves necessarily represent but a part of the reserves which may ultimately be assigned to the new reservoirs discovered each year. For a one-well field, where development has not yet gone beyond the discovery well, the area assigned as proved is usually small in regions of complex geological conditions but may be larger where the geology is relatively simple. In a sparsely drilled pool the area between wells is considered to be proved only if the geological and engineering data assure that such area will produce when drilled. The total of new oil through discoveries estimated as proved in each year is comparatively small, because development is usually not extensive during the first year. The total of new oil through extensions, on the other hand, is comparatively large. As knowledge of the factors affecting production and reservoir performance becomes available, and as these factors are studied, reserves in older fields can be estimated with greater precision and revised accordingly. Therefore, the total quantity of the new proved reserves for the year includes the oil from discoveries and extensions, modified by revisions of previous estimates where new data have made better information available.
>
> The committee again wishes especially to stress the fact that its estimates of proved reserves cannot be used in measuring the rate at which these reserves can be produced with or without physical waste. Oil cannot be produced from the permeable rocks in which it occurs at any desired rate, because the flow of oil through the pores of the oil-bearing rocks is definitely controlled by the physical factors of the reservoir. As a matter of fact, today's known oil can be recovered only over a period of many years and at gradually declining annual rates. This has been widely demonstrated by past performance under all kinds of operating conditions. Therefore, only incorrect conclusions as to the life of these reserves can be obtained by dividing these reserves by the current rate of production.

Similarly, the Committee on Natural-Gas Reserves of the American Gas Association includes the following statement in its report for 1953 (*3*):

> The committee wishes to point out that it is often not possible to estimate the total reserves of a field in the year of its discovery. Satisfactory estimates can be made only after there has been sufficient drilling in the fields and, in some cases, adequate production history established. For these reasons, the reserves listed as discovered during any current year must be considered only as the reserves indicated by the drilling in that year. The estimated reserves of the new fields and pools are revised in each succeeding annual report to show the effect of later developments and are shown as "Extensions and Revisions."

The industry obtains the basic data used in preparing estimates of reserves in its normal exploratory and development program. For their own efficient operation and for maintaining their positions in a highly competitive field, companies must assemble data on the availability of their reserves. The aggregate of these individual estimates by innumerable individuals on the technical staffs of the companies is believed to be the most accurate and comprehensive estimate now available.

POTENTIAL RESERVES

The following discussion reviews briefly the history of estimates of possible oil reserves (*28*). Many criteria have been used in these evaluations, such as volume of sediments; whether marine or nonmarine; character of sediments—that is, whether sandstone, shale, limestone, or other types; amount of organic matter contained in sediments; projection into the future of past trends of production from various types of rocks of various ages. Undoubtedly as geologic knowledge of oil-bearing rocks becomes

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[3] The word "oil," unless defined as crude oil, is used in this report as equivalent to liquid hydrocarbons.

more complete and production research develops, each estimate based on exhaustive research will be more accurate, but the time as yet has not arrived when a definitely accurate estimate of ultimate potential petroleum reserves can be made.

In 1921 the Federal Geological Survey, in cooperation with the American Association of Petroleum Geologists, prepared an estimate of oil "in sight" amounting to 5 billion barrels and "prospective and possible" oil amounting to 4 billion barrels as of January 1, 1922. The committee making the estimates emphasized that the estimates of oil in "possible" territory were speculative and hazardous as the questions involved in a particularly doubtful region are not only how much oil it will yield but also if it will yield any oil. The estimates were considered to be distinctly conservative by the committee, and subsequent discoveries have made it abundantly clear that the estimates were ultraconservative.

These estimated reserves of petroleum were considered to be adequate for the then current requirements of the United States for only 20 years, if the oil could be produced as rapidly as it was needed; however, the committee pointed out that, although there was enough oil for 20 years at the then current rate of consumption, all of the oil pools would not be found within that length of time, that drilling would be spread over many years as pools were found, and that the wells could not be quickly exhausted. Some wells would yield oil for more than 25 years, and some would not have been drilled by 1950.

Unknown quantities of oil and gas remain yet to be discovered in new fields and in deeper productive zones in the areas indicated as "possible" on the following figure 3 (*28*). In attempting to estimate the ultimate petroleum resources of the United States, Pratt stated that probably the best guide is a projection into the future of the trends estimated by our past experience in the search for oil and gas. Only 1 to 2 percent of the total area in the United States definitely tested by drilling has proved to be productive; in favorable sedimentary basins the average commercially productive area has been more than double this average. Pratt concluded that if 1.5 percent of the 577 million acres of the distinctly favorable parts of the land area of the United States would ultimately prove to be productive, as may reasonably be expected on the basis of past experience, then 8.6 million acres should be oil productive if exploration is carried to completion. The total productive area in the United States in 1948 was estimated to be 4.5 million acres for which ultimate recovery was estimated to be 57.2 billion barrels of

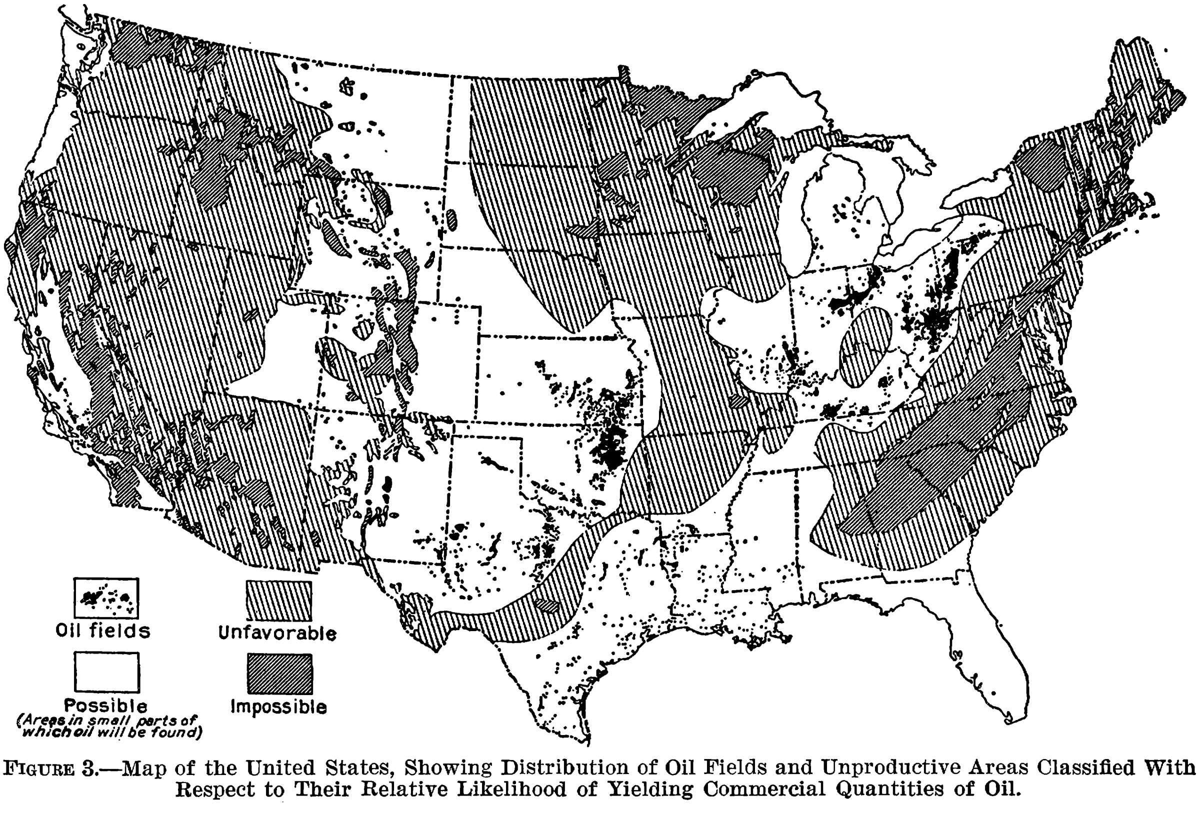

FIGURE 3.—Map of the United States, Showing Distribution of Oil Fields and Unproductive Areas Classified With Respect to Their Relative Likelihood of Yielding Commercial Quantities of Oil.

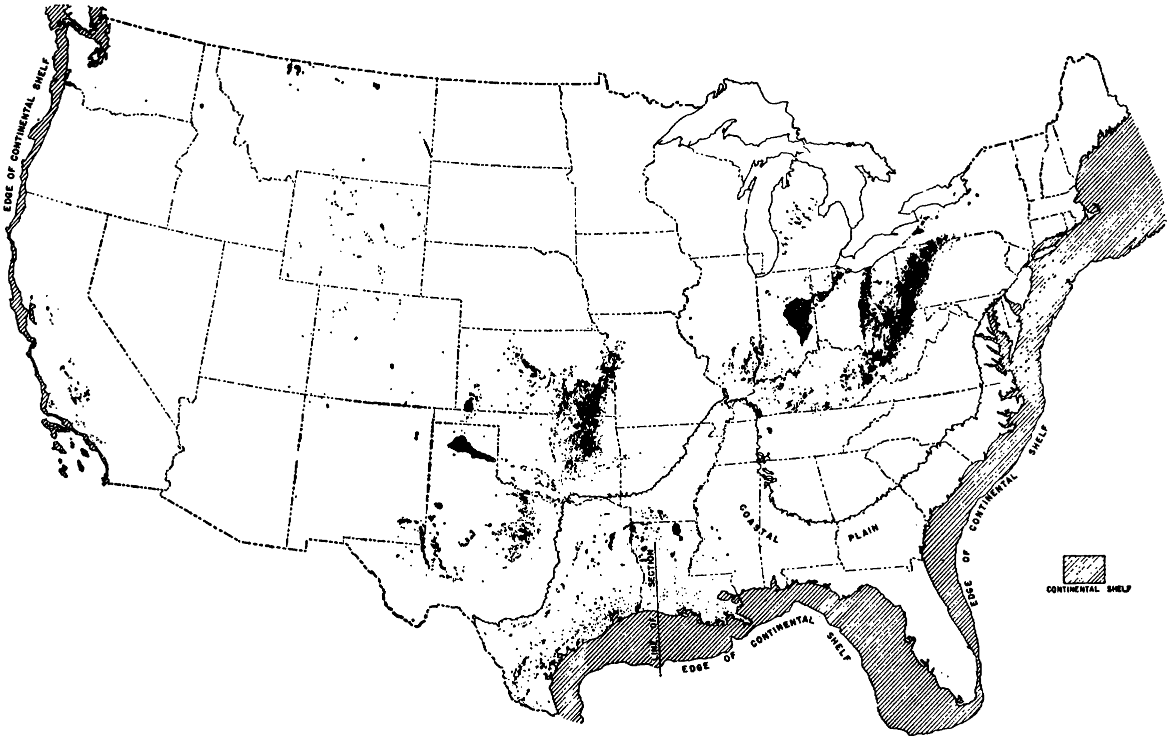

FIGURE 4.—Map of Oil and Gas Fields of the United States, Showing Gulf and Atlantic Coastal Plain and the Continental Shelf.

oil. Pratt concluded that the ultimate oil resources of the land area, excluding the Continental Shelf of the United States (fig. 4) (*28*), are of the order of 100 billion barrels.

In regard to the oil-productive potentialities of the Continental Shelf contiguous to the United States and Alaska, Pratt has compared estimates of volumes of sedimentary rocks beneath the Continental Shelf and the slope at the outer edge of the shelf to estimates of the volume of sedimentary rocks in the United States. On the basis of past discoveries and production in the United States he concluded that the continental shelves of the earth should contain more than 1,000 billion barrels of oil. As the portions of the Continental Shelf contiguous to the coasts of the United States and Alaska include nearly a tenth of the total shelf area, one might infer from Pratt's estimate that the shelves adjacent to the United States and Alaska contain a grand total of nearly 100 billion barrels of oil. In evaluating the estimate, however, it must be remembered that the seaward margins of the sediments involved in the estimate extend far out to sea and less than half of the estimate is applicable to the shelf area shoreward from the 100-fathom line (fig. 4) (*28*).

Based on the quantity of petroleum that has already been discovered in the coastal belt of Texas and Louisiana in an area comparable in size to the area of the adjoining Continental Shelf, the petroleum reserves of the Continental Shelf adjacent to the Texas and Louisiana coasts may total approximately 13 billion barrels. By a similar comparison the potential reserves of the shelf area adjacent to California are estimated to be about 2 billion barrels. It should be emphasized, however, that these two areas probably have greater oil and gas possibilities than any other part of the Continental Shelf adjacent to the United States.

Weeks (*28*) stated that, if his estimate of 610 billion barrels of oil for the land areas of the world is assumed to be approximately correct, an estimate of 400 billion barrels of potential reserves for the Continental Shelf sediments of the world would be more nearly accurate than the 1,000 billion barrels estimated by Pratt. The estimate by Weeks was based on an area of 10 million square miles for the Continental Shelf area of the world. It can be inferred from his estimate that the potential reserves of the Continental Shelf contiguous to the United States and Alaska would be approximately 40 billion barrels. With an area of about 290,000 square miles for the Continental Shelf area adjacent to the United States, it can be inferred from Weeks' conclusions that the estimated potential reserves adjacent to the continental United States would be approximately 11.6 billion barrels; the Continental Shelf area adjacent to Alaska is about 590,000 square miles, and the reserve estimate would be 23.6 billion barrels of oil.

Oil in sedimentary rocks cannot be considered to constitute reserves unless it can be economically exploited; hence, as the cost of producing oil on the Continental Shelf will be much greater than the cost of producing on land, the economic limit of production in the fields on the Continental Shelf would be reached earlier, and the ultimate recovery would be less.

Based upon the record of petroleum discovery in the past and on careful analyses of geologic data, Weeks estimated the ultimate potential petroleum resources of the United States to be 110 billion barrels, with a total of about 53.4 billion barrels yet to be discovered. In 1946 Joseph E. Pogue had estimated that the oil remaining to be discovered in the United States amounted to 50 billion barrels.

An important factor in attempts to evaluate potential reserves is the quantity of oil left in the reservoirs when oil fields are depleted by current production methods. Although an accurate figure for the average recovery of petroleum from producing fields is not available, it may be assumed to approach 50 percent. Thus the existence, location, and conditions of occurrence of vast quantities of oil that is not recoverable by techniques now in use are known. The development or application of techniques for recovery of increasingly large amounts of this oil may be anticipated. Any additional discoveries of new fields or increases in recovery of oil from present productive fields will result in consequent increases in estimates of the quantity of recoverable oil reserves.

On the basis of past results, there is no question but that increasingly detailed studies of the stratigraphy and paleontology of potentially oil- and gas-bearing rocks will lead to many new discoveries. Because most of the easily found traps, particularly those on folded structures, have already been exploited and new discoveries are becoming more and more the result of intelligent research and application of modern technology, it is believed that the most effective research will be in the fields of stratigraphy, paleontology, sedimentation, and geologic history.

Only the broader aspects of the stratigraphy and structure of the Continental Shelf areas adjacent to the United States and Alaska are known. Pratt stated that the sediments of the continental shelves of the world are better adapted to contain great quantities of petroleum than are the marine strata deposited from time to time in the past on what is now land area. He concluded that the volume of the sediments on the continental shelves of the world favorable for petroleum resources is greater than the aggregate volume of the marine sediments of this character beneath the present land surfaces and approximates 50 to 60 million cubic miles as compared with 20 million cubic miles of marine strata within the present land areas in the world. Within the boundaries of the United States, he estimated the volume of marine sediments of favorable character to be about 3 million cubic miles, and in these sediments about 57 billion barrels of oil has been discovered.

Terry (*63*) estimated the future potential production of natural gas to be 300 trillion cubic feet or more in 1944 when the highest estimate of proved gas reserves was 110 trillion cubic feet and the annual production about 4 trillion cubic feet. Since then new fields have been discovered and old fields enlarged, and the annual rate of net production has increased to 8.9 trillion cubic feet for 1952. In a more recent article (*64*) Terry stated that, in the light of new evidence, his 1944 estimate of future natural-gas supply appeared to have been entirely too conservative and from new evidence he considered that the future recoverable natural gas in this country would exceed 500 trillion cubic feet.

No precise statement as to the amount of potential reserves of oil and gas in the United States can yet be made. If the future can be judged by the past, oil and gas will be found in large enough quantities for many years to come. In the United States adequate production has been a direct function of economic incentive. Until the unpredictable date at which that incentive fails to provide the needed supplies, there will be no convincing evidence that we have reached the limits of our ability to expand the potential ultimately recoverable reserves of petroleum.

SOURCES AND ADEQUACY OF STATISTICAL INFORMATION

The collection of statistical information concerning the petroleum and related industries is designed to serve several purposes:

(1) The information provides a guide for Government agencies in establishing policies for taxation, tariffs, and other matters affecting the petroleum industry.

(2) It informs the industry of the overall rates of operations of its various branches, the stocks of crude petroleum and products, and the demand for these items, so that the individual companies can regulate their operations to provide the country with adequate supplies and, at the same time, avoid excessive operations with resultant burdensome inventories.

(3) It provides the Government with guideposts for its technical research programs by indicating the adequacy of the supplies of crude oil and the various petroleum products, and the constantly changing trends in the requirements for those products.

(4) It is useful to the fuel-consuming industries in indicating the relationship of supply to consumption at any particular time and long-term variations in this relationship.

The collection of statistics on petroleum by the United States Government dates back to 1859, the year in which Col. Drake drilled the first oil well, when the Bureau of the Census first collected data on refining of crude oil. Starting in 1882 the Geological Survey collected annual statistics on crude-oil production, prices, number of wells, and related matters.

The relatively large demands for petroleum products that developed during World War I made it necessary for Government and industry to have available current information on the supply and demand for petroleum. To provide this information, the Bureau of Mines began the collection and publication of monthly statistics on production of crude oil and natural gasoline, movement to refineries, output of refined products, and stocks of crude oil and the major products. This report has been expanded to include other products as their importance justified and to cover other phases of the industry, such as pipeline movements of products.

The statistics of the petroleum industry are provided by the various operating companies; the Bureau of Mines serves only as a collecting, compiling, and publishing agency. Only a Government agency can properly perform these functions because most companies are exceedingly reluctant to release detailed information concerning their individual operations to the public and require that their reports be held in strictest confidence. Statisticians and economists of the various oil companies, both individually and through their trade associations, assist the Bureau of Mines in designing questionnaires and publications and in securing prompt and complete reports from nearly all petroleum companies. The effectiveness of this statistical program is indicated by the fact that the Bureau receives reports in each of the surveys from companies representing over 95 percent of the total volume of crude-oil products.

PRODUCTION, CONSUMPTION, AND FOREIGN TRADE

The petroleum industry of the United States occupies a unique position of having maintained almost continuous growth throughout its entire life, now nearing a full century. Two major wars affected it only to the extent of accelerating its growth, and these accelerations continued long after the wars had ceased. Even the severe depression of the 1930's retarded the oil industry only slightly compared with its reaction on other industries.

Except for some rather violent fluctuations during the early history of the industry, crude-oil production has risen steadily, reaching an average of 100,000 barrels daily in 1890, 1 million barrels in 1919, 2 million in 1936, 4 million in 1942, 5 million in 1947, and 6.34 million in 1954. From a rate of 2,760,000 barrels of oil daily in 1929, production fell to 2,145,000 barrels in 1932, but by 1935 the rate of production had almost regained the 1929 level.

As the petroleum industry grew, crude oil was discovered in State after State until 29 States now contribute to the Nation's supply. In 1954 Texas, continuing as the largest producing State, contributed 42 percent of the total oil production in the United States. Following in order were California, 15 percent; Louisiana, 11 percent; Oklahoma, 8 percent; and Kansas, 5 percent. With these 5 States producing over 80 percent of the total, it is obvious that the other 24 producing States individually produce oil currently at low rates, yet collectively they make a substantial contribution to the Nation's petroleum supply, and for many of them petroleum production is a major industry. For example, in 7 of these States (exclusive of the 5 mentioned above), crude petroleum ranks first among all minerals from the aspect of value of production; in 4 others it ranks second; and in 2 it is third.

Quantitatively consumption has followed the same general pattern as production, with a steady and fairly sharp increase throughout the life of the industry. Geographically, the pattern of consumption has shifted notably, reflecting movements of population and industry. In the early years consumption was concentrated in the east coast area; in 1876, with the discovery of oil in California, the west coast became a factor in consumption. After that, supplies of petroleum products rapidly became available in all areas at attractive prices, and their use spread rapidly throughout the country.

Probably the most interesting and violent changes in consumption have been in the shifting emphasis on the various products. In the earliest days crude oil was refined, with the objective of obtaining a maximum amount of kerosine and a limited amount of lubricating oil. At that time most of the other products were waste materials, the disposal of useless gasoline being a major problem. With the advent of automobiles, gasoline shifted from a waste product to the most desirable and all efforts were directed toward maximum production of motor fuel. More recently, the rapid growth in the use of diesel engines and of oil furnaces for home heating has moved the emphasis toward distillate fuel oil. Current developments indicate that the shifting of demand for various fractions of crude oil will continue. The jet engine, the use of petroleum components for synthetic rubber and plastics, and the developments in the petrochemical industry all are factors creating new demands for one or another of the many petroleum products.

The United States, being one of the first countries to develop commercial production and refining of petroleum, became an early factor in world supply. In 1864 some 550,000 barrels of crude oil and products (almost one-quarter of our total production) was exported, and since that time foreign shipments have been of considerable importance although not representing nearly as great a part of our total production. During 1954 total exports of petroleum and products were 130 million barrels or about 6 percent of the annual crude-oil production.

Petroleum was not imported before 1908. Volumewise, the imports were consistently less than exports except for 1920–23 and since 1947. On a value basis the foreign trade of the United States in petroleum is quite profitable, because the imports consist principally of low-value crude oil and residual fuel oil while exports are composed of the relatively much higher valued products. Thus, in 1953 the United States exported to foreign countries 127 million barrels of crude oil and products valued at $691 million while importing 351 million barrels with a value of $762 million.

Statistical data on domestic demand for petroleum products, petroleum exports and imports of the United States, and petroleum produced in the United States by States are shown in tables 14, 15, 16, and 17.

COSTS AND PRICES

The costs of finding and bringing a particular oil or gas field into production have no direct relation to the value that will be received for the output of that field. The costs of finding petroleum (exploratory costs) depend upon: (1) The area that must be surveyed before a favorable structure is found; (2) the complexity of the geologic formations studied in deciding a favorable location; (3) the accessibility of the selected site to men and equipment; and (4) the terms agreed upon with the landowner for his mineral rights. Drilling costs will depend upon the geographical location of the well, the type of formations penetrated, the depth of the producing formation, and the character of the oil-producing formation. There is no certainty that oil in producible quantities will be found by the first well drilled or by any well drilled—in which case the total expenditure is a loss. And again, if oil or gas is found the financial success of the venture still depends on the extent and quality of the reserves.

The price the producer will receive for his oil in relation to the going market price depends upon the price of competitive oil delivered to the market. Thus the quality of the oil and the transportation costs play a large part in determining the price that the producer receives. The operator has no control over a great variety of factors that influence the costs of finding oil and the price to be received for it. In addition, whether a particular well or a particular field shows a profit depends upon the relation of these expenses to the market price of crude throughout the life of the well, and the price of crude oil depends in part on the market for competitive fuels and also in part on competition between various sources of crude oil.

The search for oil has gone successively into more difficult locations, to deeper drilling, to offshore wells, to foreign countries. The price of oil must be high enough to make the risk of exploration attractive.

The field prices of crude oil and of natural gas and the wholesale price of petroleum products have in general followed the trends of the wholesale price index (fig. 5). Petroleum prices appear to have declined more steeply than prices in general before the depression of the 1930's and to have risen more sharply than other prices in the years after World War II.

In refining, the investment per barrel of throughput capacity has increased constantly as more and more complex equipment has been installed to meet the need for specialized products. This rising investment has been offset by two principal factors: (1) The more elaborate equipment has enabled refiners to convert a large part of their lower valued products, such as residual fuel, to higher valued products, principally gasoline; (2) by installation of elaborate automatic control devices in refineries the manpower required to process a given volume of oil has been substantially reduced.

There have been appreciable cost reductions in transportation by the installation of elaborate pipeline systems for moving both crude oil and products, by constantly enlarging the size (diameter) of these pipelines, and by building larger and more efficient tankers. By increasing the sizes of these transportation facilities there is a marked increase in their capacities to move petroleum with little or no increase in manpower and resultant economies in cost. In pipeline operation, as in refineries, there have been

TABLE 14.—*Domestic demand for petroleum products in the United States, 1935–54*

[Thousand barrels of 42 gallons]

Year	Gasoline	Kerosine	Distillate	Residual	Jet fuel	Lubricants
1935	434, 810	47, 645	86, 028	280, 695	------------	19, 661
1936	481, 606	51, 428	102, 757	307, 884	------------	22, 323
1937	519, 352	54, 972	116, 841	325, 514	------------	23, 323
1938	523, 003	56, 360	117, 449	291, 833	------------	21, 233
1939	555, 509	60, 503	134, 973	323, 488	------------	23, 713
1940	589, 490	68, 776	160, 851	340, 163	------------	24, 690
1941	667, 505	69, 469	172, 824	383, 422	------------	30, 255
1942	589, 110	69, 767	185, 661	405, 697	------------	29, 057
1943	568, 238	68, 598	208, 110	467, 008	------------	31, 459
1944	632, 482	71, 812	209, 320	512, 020	------------	32, 363
1945	696, 333	75, 573	226, 084	523, 423	------------	35, 334
1946	735, 417	89, 088	242, 894	480, 029	------------	34, 891
1947	795, 015	102, 519	298, 273	518, 510	------------	36, 481
1948	871, 270	112, 220	340, 576	500, 543	------------	35, 983
1949	913, 713	102, 672	329, 278	496, 021	------------	33, 101
1950	994, 290	117, 844	394, 885	553, 793	------------	38, 853
1951	1, 089, 566	123, 241	447, 278	564, 397	------------	42, 292
1952	1, 142, 987	121, 253	476, 986	555, 165	20, 126	38, 165
1953	1, 205, 775	114, 467	488, 075	560, 474	34, 483	40, 497
1954	1, 238, 346	118, 260	526, 369	522, 099	45, 852	38, 444

Year	Wax	Coke	Asphalt	Road oil	LP-gases	Miscellaneous	Total
1935	933	6, 704	15, 652	5, 962	----------	64, 650	962, 740
1936	1, 077	6, 266	20, 595	7, 279	----------	67, 864	1, 069, 079
1937	1, 062	5, 766	21, 876	7, 954	----------	72, 723	1, 149, 383
1938	995	5, 589	24, 154	7, 847	----------	71, 908	1, 120, 371
1939	1, 162	7, 109	27, 094	7, 846	----------	76, 625	1, 218, 022
1940	1, 275	7, 035	28, 182	7, 849	----------	81, 674	1, 309, 985
1941	1, 871	7, 144	35, 485	8, 980	----------	93, 966	1, 470, 921
1942	1, 950	5, 037	35, 733	8, 484	----------	98, 262	1, 428, 758
1943	2, 092	5, 250	36, 402	2, 450	----------	116, 649	1, 506, 256
1944	2, 261	8, 327	38, 129	1, 560	----------	149, 524	1, 657, 798
1945	2, 403	9, 214	38, 350	2, 505	----------	148, 269	1, 757, 488
1946	2, 271	9, 029	43, 253	5, 939	39, 667	96, 118	1, 778, 596
1947	2, 478	10, 082	47, 023	7, 067	52, 761	94, 508	1, 964, 717
1948	2, 348	11, 670	49, 962	8, 027	65, 508	90, 433	2, 088, 540
1949	2, 255	14, 427	49, 362	7, 826	68, 238	87, 201	2, 104, 094
1950	3, 238	15, 021	58, 677	6, 897	85, 505	87, 425	2, 356, 428
1951	3, 246	14, 481	66, 179	6, 095	101, 254	100, 211	2, 558, 240
1952	3, 443	13, 924	71, 007	6, 947	108, 304	99, 821	2, 658, 128
1953	3, 889	17, 599	72, 208	6, 610	118, 662	103, 941	2, 766, 680
1954	3, 927	19, 762	76, 643	7, 216	116, 630	104, 570	2, 818, 118

additional savings in operation through installation of automatic control equipment.

The price structure of petroleum products is so extremely complex that only a few general principles can be discussed here. There is, of course, the constant competition among the oil companies for a share of the market that will permit each to operate its facilities at their most economic rate. This places an effective ceiling on prices for all products because, in every area, companies of all types and sizes are competing for the available market.

In the broader field of competition among the various fuels, the availability and price of each fuel, plus other factors, such as convenience, govern prices. In recent years the increasing availability of natural gas because of the extension of gas-transmission lines to almost all sections of the country, the relatively low price of gas furnaces, and prices for the gas that are generally competitive with other fuels have caused rapid increases in use of this fuel. Sales of natural gas to industrial users usually are limited to those periods when transmission capacity exceeds household needs, and these sales are made at prices that are below those of competitive fuels when all factors are considered. In future, provision of large underground storage capacity near points of consumption will permit holding excess gas delivered during low-demand periods for use in the peak heating season. This development will

TABLE 15.—*Exports from the continental United States, 1935–54*
[Thousand barrels of 42 gallons]

Year	Crude petroleum	Motor fuel	Kerosine	Distillate fuel oil	Residual fuel oil	Lubricants	Other	Total
1935	51, 430	30, 613	6, 651	16, 249	12, 699	8, 499	2, 845	128, 986
1936	50, 313	28, 646	6, 936	20, 448	14, 435	8, 691	2, 525	131, 994
1937	67, 231	38, 293	8, 885	30, 128	15, 304	10, 975	2, 001	172, 817
1938	77, 254	50, 109	7, 504	29, 641	17, 920	9, 417	1, 883	193, 728
1939	72, 076	44, 638	8, 241	32, 020	17, 485	11, 881	2, 618	188, 959
1940	51, 496	25, 377	3, 374	19, 140	16, 109	10, 461	4, 509	130, 466
1941	33, 238	27, 083	3, 221	16, 925	14, 114	9, 924	4, 325	108, 830
1942	33, 834	35, 097	2, 576	21, 575	12, 095	8, 272	3, 458	116, 907
1943	41, 342	51, 577	4, 752	24, 957	14, 894	8, 863	3, 572	149, 957
1944	34, 238	100, 537	4, 888	43, 491	12, 536	8, 709	3, 217	207, 616
1945	32, 998	88, 059	6, 180	33, 496	11, 669	6, 575	4, 006	182, 983
1946	42, 436	45, 334	8, 637	29, 487	9, 188	11, 051	6, 990	153, 123
1947	46, 355	47, 449	7, 252	29, 877	10, 623	14, 871	8, 050	164, 477
1948	39, 736	37, 302	3, 495	21, 293	13, 011	13, 392	6, 445	134, 674
1949	33, 069	39, 347	2, 533	12, 295	12, 641	12, 912	6, 579	119, 376
1950	34, 823	24, 721	2, 078	12, 653	16, 228	14, 252	6, 551	111, 306
1951	28, 604	40, 136	6, 843	22, 555	28, 999	17, 429	9, 486	154, 052
1952	26, 696	36, 285	7, 821	33, 515	27, 701	16, 031	10, 139	158, 188
1953	19, 931	37, 925	7, 265	32, 328	25, 991	12, 999	10, 152	146, 591
1954	13, 564	34, 463	4, 903	24, 201	26, 856	15, 168	10, 866	130, 021

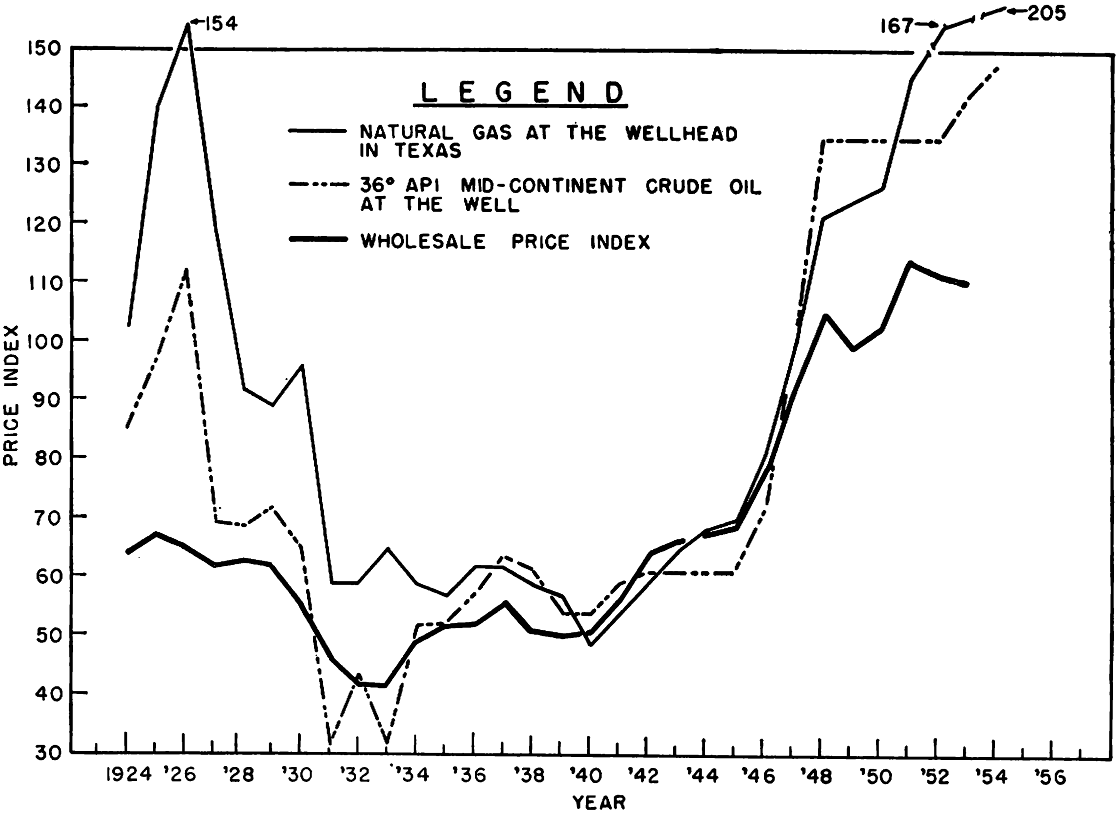

FIGURE 5.—Price Indexes of Crude Oil and Natural Gas (1947=100) and Wholesale Price Index (1947–49=100).

tend to stabilize availability and prices to both domestic and industrial users.

Prices for heating oils, both domestic and industrial, are influenced by prices for competitive fuels when convenience, labor involved, and cost of equipment are considered. However, the actual costs of producing and delivering each fuel must, to a large extent, govern its price. When the delivered price of any fuel becomes higher in comparison to the others than the consumer feels is justified, that fuel will command a diminishing share of the market.

For the future, it appears that prices for natural gas may still increase relative to other fuels without interfering with the normal expected increase in consumption of natural gas; however, there is no apparent reason for prices of petroleum products to move contrary to the general trend of commodity prices until such time as available supplies of crude oil—either domestic or imported—are inadequate to meet the Nation's needs. By the time that requirements for petroleum fuels outstrip supplies it is likely that substitutes, synthetic liquid fuels, or energy from sources now classified as "nonconventional" will supplement or replace petroleum fuels for many uses at comparable prices.

Table 16.—*Imports of petroleum into the continental United States, 1935–54* [1]

[Thousand barrels of 42 gallons]

Year	Crude oil	Residual	Other	Total
1935	32,330	19,909	19,929	72,168
1936	32,327	18,801	5,976	57,104
1937	27,484	22,887	6,781	57,152
1938	26,412	21,085	6,746	54,243
1939	33,095	15,680	10,285	59,060
1940	42,738	28,458	11,720	82,916
1941	50,606	37,369	9,167	97,142
1942	12,297	18,433	5,237	35,967
1943	13,833	27,210	22,369	63,412
1944	44,805	36,553	10,898	92,256
1945	74,337	31,648	7,634	113,619
1946	86,066	44,647	6,963	137,676
1947	97,532	54,244	7,613	159,389
1948	129,093	53,269	5,782	188,144
1949	154,719	75,330	6,307	236,356
1950	172,736	120,116	10,848	303,700
1951	177,356	119,166	7,502	304,024
1952	207,492	128,544	10,346	346,382
1953	233,212	136,100	9,304	378,616
1954	239,479	129,009	15,421	383,909

[1] Statistics on products from U. S. Department of Commerce.

TAXES

By its tax payments, the petroleum industry contributes very substantially to the cost of operating the various Government units of the United States. Total amounts are not available; but a study of 30 companies that supply 52 percent of the crude oil produced and 85 percent of the crude oil refined, shows that in 1952 these companies paid $1.7 billion in taxes. In addition, they collected, for Federal, State, and local governments, sales taxes amounting to $2.1 billion, while taxes on their purchases, payrolls, and dividends were estimated at $2.9 billion.

The taxes paid by or through the petroleum industry are multiple and varied. In addition to the income and property taxes to which all companies are subject, the oil industry pays or collects from its customers a tax of some kind each time a gallon of oil moves. Most oil-producing States impose a severance tax on each barrel of crude withdrawn from the ground. The Federal Government imposes a tax on the interstate movement of oil and refined products by pipeline. Many States collect inspection fees on petroleum products, even though the selling companies often are responsible for conducting the inspections. Finally, there are Federal, State, and some city taxes on gasoline, on diesel fuel used in highway transportation, and on lubricating oil.

TARIFF

Before 1932 oil imports into the United States were duty free. The Revenue Act of 1932, which became effective in June of that year, imposed an excise tax of 21 cents per barrel on

Table 17.—*Petroleum produced in the United States, 1918–36, by States*

[Thousand barrels of 42 gallons]

Year	Alabama	Arkansas	California	Colorado	Florida	Illinois	Indiana	Kansas	Kentucky
1918	----------	----------	97,532	143	----------	13,366	878	45,451	4,368
1919	----------	----------	101,183	121	----------	11,960	972	33,048	9,278
1920	----------	(1)	103,377	111	----------	10,774	945	39,005	8,738
1921	----------	10,473	112,600	108	----------	10,043	1,158	36,456	9,013
1922	----------	12,712	138,468	97	----------	9,383	1,087	31,766	8,973
1923	----------	36,610	262,876	86	----------	8,707	1,043	28,250	8,069
1924	----------	46,028	228,933	445	----------	8,081	935	28,836	7,407
1925	----------	77,398	232,492	1,226	----------	7,863	829	38,357	6,759
1926	----------	58,332	224,673	2,768	----------	7,760	808	41,498	6,274
1927	----------	40,005	231,196	2,831	----------	6,994	852	41,069	6,719
1928	----------	32,096	231,811	2,774	----------	6,462	1,052	38,596	7,359
1929	----------	24,917	292,534	2,358	----------	6,319	981	42,813	7,775
1930	----------	19,702	227,329	1,656	----------	5,736	994	41,638	7,389
1931	----------	14,791	188,830	1,545	----------	5,039	840	37,018	6,456
1932	----------	12,051	178,128	1,136	----------	4,673	806	34,848	6,287
1933	----------	11,686	172,010	919	----------	4,244	737	41,976	4,608
1934	----------	11,182	174,305	1,139	----------	4,479	838	46,482	4,860
1935	----------	11,008	207,832	1,560	----------	4,322	777	54,843	5,258
1936	----------	10,469	214,773	1,650	----------	4,475	822	58,317	5,633

For footnote see end of table.

TABLE 17.—*Petroleum produced in the United States, 1918–36, by States*—Continued

[Thousand barrels of 42 gallons]

Year	Louisiana	Michigan	Mississippi	Missouri	Montana	Nebraska	New Mexico	New York	Ohio	Oklahoma
1918	16, 043	(1)	---------	---------	69	---------	---------	809	7, 285	103, 347
1919	17, 188	(1)	---------	(1)	90	---------	(1)	851	7, 736	86, 911
1920	35, 714	---------	---------	(1)	340	---------	(1)	906	7, 400	106, 206
1921	27, 103	---------	---------	(1)	1, 509	---------	(1)	988	7, 335	114, 634
1922	35, 376	---------	---------	(1)	2, 449	---------	(1)	1, 000	6, 781	149, 571
1923	24, 919	---------	---------	(1)	2, 782	---------	(1)	1, 250	7, 085	160, 929
1924	21, 124	---------	---------	---------	2, 815	---------	98	1, 440	6, 811	173, 538
1925	20, 272	4	---------	---------	4, 091	---------	1, 060	1, 695	7, 212	176, 768
1926	23, 201	94	---------	---------	7, 727	---------	1, 666	1, 956	7, 272	179, 195
1927	22, 818	439	---------	---------	5, 058	---------	1, 226	2, 242	7, 593	277, 775
1928	21, 847	594	---------	---------	4, 015	---------	943	2, 603	7, 015	249, 857
1929	20, 554	4, 528	---------	---------	3, 980	---------	1, 830	3, 377	6, 743	255, 004
1930	23, 272	3, 911	---------	---------	3, 349	---------	10, 189	3, 647	6, 486	216, 486
1931	21, 804	3, 789	---------	---------	2, 830	---------	15, 227	3, 363	5, 327	180, 574
1932	21, 807	6, 910	---------	---------	2, 457	---------	12, 455	3, 508	4, 644	153, 244
1933	25, 168	7, 942	(1)	(1)	2, 273	---------	14, 116	3, 181	4, 235	182, 251
1934	32, 869	10, 603	(1)	(1)	3, 603	---------	16, 864	3, 804	4, 234	180, 107
1935	50, 330	15, 776	(1)	(1)	4, 603	---------	20, 483	4, 236	4, 082	185, 288
1936	80, 491	11, 928	---------	(1)	5, 868	---------	27, 223	4, 663	3, 847	206, 555

Year	Pennsylvania	Tennessee	Texas	Virginia	West Virginia	Wyoming	Other [2]	United States total		
								Quantity	Value at wells	
									Total (thousand dollars)	Average per barrel
1918	7, 408	---------	38, 750	---------	7, 867	12, 596	16	355, 928	703, 944	1. 98
1919	8, 137	---------	79, 366	---------	8, 327	13, 172	27	378, 367	760, 266	2. 01
1920	7, 438	---------	96, 868	---------	8, 249	16, 831	27	442, 929	1, 360, 745	3. 07
1921	7, 418	---------	106, 166	---------	7, 822	19, 333	24	472, 183	814, 745	1. 73
1922	7, 425	---------	118, 684	---------	7, 021	26, 715	23	557, 531	895, 111	1. 61
1923	7, 609	---------	131, 023	---------	6, 358	44, 785	26	732, 407	978, 430	1. 34
1924	7, 486	---------	134, 522	---------	5, 920	39, 498	23	713, 940	1, 022, 683	1. 43
1925	8, 097	---------	144, 648	---------	5, 763	29, 173	36	763, 743	1, 284, 960	1. 68
1926	8, 961	---------	166, 916	---------	5, 946	25, 776	51	770, 874	1, 447, 760	1. 88
1927	9, 526	---------	217, 389	---------	6, 023	21, 307	67	901, 129	1, 172, 830	1. 30
1928	9, 956	---------	257, 320	---------	5, 661	21, 461	52	901, 474	1, 054, 880	1. 17
1929	11, 820	---------	296, 876	---------	5, 574	19, 314	26	1, 007, 323	1, 280, 417	1. 27
1930	12, 803	---------	290, 457	---------	5, 071	17, 868	28	898, 011	1, 070, 200	1. 19
1931	11, 892	---------	332, 437	---------	4, 472	14, 834	13	851, 081	550, 630	. 65
1932	12, 412	---------	312, 478	---------	3, 876	13, 418	21	785, 159	680, 460	. 87
1933	12, 624	---------	402, 609	---------	3, 815	11, 227	35	905, 656	608, 000	. 67
1934	14, 478	---------	381, 516	---------	4, 095	12, 556	51	908, 065	904, 825	1. 00
1935	15, 810	(1)	392, 666	---------	3, 902	13, 755	65	996, 596	961, 440	. 96
1936	17, 070	(1)	427, 411	---------	3, 847	14, 582	63	1, 099, 687	1, 199, 820	1. 09

For footnotes, see end of table.

TABLE 17.—*Petroleum produced in the United States, 1937–54, by States*—Continued

[Thousand barrels of 42 gallons]

Year	Alabama	Arkansas	California	Colorado	Florida	Illinois	Indiana	Kansas	Kentucky
1937	----------	11, 764	238, 521	1, 605	----------	7, 499	844	70, 761	5, 484
1938	----------	18, 180	249, 749	1, 412	----------	24, 075	995	60, 064	5, 821
1939	----------	21, 238	224, 354	1, 404	----------	94, 912	1, 711	60, 703	5, 621
1940	----------	25, 775	223, 881	1, 626	----------	147, 647	4, 978	66, 139	5, 188
1941	----------	26, 327	230, 263	2, 150	----------	132, 393	7, 411	83, 242	4, 762
1942	----------	26, 628	248, 326	2, 199	----------	106, 391	6, 743	97, 636	4, 534
1943	----------	27, 600	284, 188	2, 320	4	82, 260	5, 283	106, 178	7, 883
1944	43	29, 418	311, 793	3, 083	12	77, 413	5, 118	98, 762	9, 621
1945	181	28, 613	326, 482	5, 036	30	75, 094	4, 868	96, 415	10, 325
1946	380	28, 375	314, 713	11, 856	57	75, 297	6, 726	97, 218	10, 578
1947	396	29, 948	333, 132	15, 702	259	66, 459	6, 095	105, 132	9, 397
1948	466	31, 682	340, 074	17, 862	290	64, 808	6, 974	110, 908	8, 801
1949	462	29, 986	332, 942	23, 587	441	64, 501	9, 696	101, 868	8, 803
1950	735	31, 108	327, 607	23, 303	487	62, 028	10, 699	107, 586	10, 381
1951	1, 020	29, 798	354, 561	27, 823	596	60, 243	11, 100	114, 522	11, 622
1952	1, 279	29, 440	359, 450	30, 381	591	60, 089	12, 037	114, 807	11, 918
1953	1, 694	29, 681	365, 085	36, 402	543	59, 026	12, 823	114, 566	11, 518
1954[3]	1, 584	29, 206	355, 779	41, 052	548	66, 998	11, 204	119, 317	13, 791

Year	Louisiana	Michigan	Mississippi	Missouri	Montana	Nebraska	New Mexico	New York	Ohio	Oklahoma
1937	90, 924	16, 628	----------	(1)	5, 805	----------	38, 854	5, 478	3, 559	228, 839
1938	95, 208	18, 745	----------	(1)	4, 946	----------	35, 759	5, 045	3, 298	174, 994
1939	93, 646	23, 462	107	40	5, 960	2	37, 637	5, 098	3, 156	159, 913
1940	103, 584	19, 753	4, 400	44	6, 728	276	39, 129	4, 999	3, 159	156, 164
1941	115, 908	16, 359	15, 327	47	7, 526	1, 898	39, 569	5, 185	3, 510	154, 702
1942	115, 785	21, 754	28, 833	36	8, 074	1, 237	31, 544	5, 421	3, 543	140, 690
1943	123, 592	20, 768	18, 807	36	7, 916	635	38, 896	5, 059	3, 322	123, 152
1944	129, 645	18, 490	16, 337	45	8, 647	417	39, 555	4, 697	2, 937	124, 616
1945	131, 051	17, 267	19, 062	45	8, 420	305	37, 351	4, 648	2, 828	139, 299
1946	143, 669	17, 074	24, 298	51	8, 825	293	36, 814	4, 863	2, 908	134, 794
1947	160, 128	16, 215	34, 925	55	8, 742	229	40, 926	4, 762	3, 108	141, 019
1948	181, 458	16, 871	45, 761	31	9, 382	215	47, 969	4, 621	3, 600	154, 455
1949	190, 826	16, 517	37, 966	49	9, 118	330	47, 645	4, 425	3, 483	151, 660
1950	208, 965	15, 826	38, 236	32	8, 109	1, 547	47, 367	4, 143	3, 383	164, 599
1951	232, 281	13, 927	37, 039	24	8, 958	2, 558	52, 719	4, 254	3, 140	186, 869
1952	243, 929	13, 251	36, 310	21	9, 606	2, 660	58, 681	4, 242	3, 350	190, 435
1953	256, 632	12, 285	35, 620	39	11, 920	6, 344	70, 441	3, 800	3, 610	202, 570
1954[3]	246, 093	12, 027	33, 591	78	13, 687	7, 423	75, 200	3, 257	3, 829	186, 349

Year	Pennsylvania	Tennessee	Texas	Virginia	West Virginia	Wyoming	Other [2]	U. S. total		
								Quantity	Value at wells	
									Total (thousand dollars)	Ave. per bbl.
1937	19, 189	(1)	510, 318	--------	3, 845	19, 166	77	1, 279, 160	1, 513, 340	1. 18
1938	17, 426	(1)	475, 850	--------	3, 684	19, 022	82	1, 214, 355	1, 373, 060	1. 13
1939	17, 382	50	483, 528	--------	3, 580	21, 454	4	1, 264, 962	1, 294, 470	1. 02
1940	17, 353	24	493, 209	--------	3, 444	25, 711	3	1, 353, 214	1, 385, 440	1. 02
1941	16, 750	12	505, 572	--------	3, 433	29, 878	4	1, 402, 228	1, 602, 000	1. 14
1942	17, 779	9	483, 097	--------	3, 574	32, 812	------	1, 386, 645	1, 643, 470	1. 19
1943	15, 757	10	594, 343	2	3, 349	34, 253	------	1, 505, 613	1, 809, 020	1. 20
1944	14, 118	9	746, 699	3	3, 070	33, 356	------	1, 677, 904	2, 032, 960	1. 21
1945	12, 515	8	754, 710	4	2, 879	36, 219	------	1, 713, 655	2, 094, 250	1. 22
1946	12, 996	10	760, 215	23	2, 929	38, 977	------	1, 733, 939	2, 442, 550	1. 41
1947	12, 690	8	820, 210	61	2, 617	44, 772	------	1, 856, 987	3, 577, 890	1. 93
1948	12, 667	19	903, 498	33	2, 692	55, 032	16	2, 020, 185	5, 245, 080	2. 60
1949	11, 374	18	744, 834	43	2, 839	47, 890	637	1, 841, 940	4, 674, 770	2. 54
1950	11, 859	12	829, 874	21	2, 808	61, 631	1, 228	1, 973, 574	4, 963, 380	2. 51
1951	11, 345	14	1, 010, 270	12	2, 757	68, 929	1, 330	2, 247, 711	5, 690, 410	2. 53
1952	11, 233	15	1, 022, 139	10	2, 602	68, 074	3, 286	2, 289, 836	5, 785, 230	2. 53
1953	10, 649	16	1, 019, 164	8	3, 038	82, 618	6, 990	2, 357, 082	6, 327, 100	2. 68
1954[3]	9, 107	17	981, 722	10	2, 902	93, 633	7, 919	2, 316, 323	6, 427, 140	2. 77

[1] Included under Other.

[2] Other includes Alaska, 1918–33; Arkansas, 1920; Michigan, 1918–19; Mississippi, 1933–35; Missouri, 1919–23, 1932–38; Nevada 1954; New Mexico, 1919–23; North Dakota, 1951–54; South Dakota, 1954 ;Tennessee, 1918–38; and Utah 1924–41, 1948–54.

[3] Preliminary figures.

crude and fuel oils, $1.05 per barrel on gasoline, and $1.68 per barrel on lubricating oils (*68*).

In 1939 a trade agreement with Venezuela reduced the tariff to 10.5 cents per barrel on a quota amount of crude and fuel oils. The quota was set at 5 percent of crude oil run to stills in the continental United States in the preceding year. Imports over the quota were taxed at the previous rate of 21 cents per barrel.

The General Agreement on Tariffs and Trade, effective January 1, 1948, reduced by one-half the excise-tax rates on gasoline and lubricating oils; the new rates were 52.5 and 84 cents per barrel, respectively. The change had no significance as no appreciable quantities of these products were imported either before or after it became effective.

From January 30, 1943, to January 1, 1951, an agreement with Mexico eliminated the quota system and allowed all crude and fuel oils to be imported into the United States under the 10.5 cents per barrel tariff rate.

The Mexican agreement was terminated January 1, 1951, and the preceding Venezuelan agreement again became effective. Upon protest by Venezuela of resumption of the quota system a new agreement was negotiated with Venezuela, effective October 11, 1952. This eliminated quotas and set an excise tax of 5.25 cents per barrel on crude- and fuel-oil imports below 25° API gravity and a tax of 10.5 cents per barrel for API gravity of 25° and above.

The tariff on petroleum follows the usual tariff structure of the United States; finished products are taxed at a higher rate than raw materials. This was intended to conserve the employment provided by the processing operations for American labor.

In 1953 and 1954 there have been a number of proposals to increase the tariff and to establish severe quotas on crude oil and on residual fuel oil. The proposals have been made by domestic crude-oil producers and by coal producers. Excess crude-oil productive capacity had caused the production allowables to be reduced in States with conservation controls, resulting in reduced income to domestic producers. The loss of market of coal to residual fuel oil caused the coal producers to seek to improve their competitive position by increasing the tariff or by imposing quotas on oil imports.

SELF-SUFFICIENCY

As recently as 1947 more crude petroleum and petroleum products were being exported than were being imported by the United States. Imports of crude oil, however, have exceeded crude exports since 1944. The spread between these two figures has grown wider in each succeeding year. In 1954 the United States net imports of all oils averaged 712,000 barrels per day. The country's reserve crude productive capacity at that time was estimated to exceed 2 million barrels per day (*48*), so that there is at present a self-sufficiency in crude-oil productive capacity and a substantial reserve for emergencies.

The development of foreign petroleum resources has been encouraged through the participation of American-controlled companies and through the moderate tariff policy of the United States. From a national point of view the drain upon our resources is reduced, and their life is extended. Also, by extending the area of search for new sources of petroleum, the more easily accessible oil will be developed first, thus diminishing the pressure for price increases that accompany the high exploration and production costs of supplying oil from less accessible sources in this country.

The United States does not import significant quantities of refined petroleum products, except residual fuel oil. Refinery capacity in this country has expanded with demand so that, given a supply of crude oil, domestic or imported, product needs can be satisfied. Residual fuel oil has been imported in increasing quantities into the east coast region of this country. This fuel, because of its availability at Caribbean refineries, competes strictly on a price basis with other fuels for heavy industrial and utility consumption along the seaboard.

Domestic refineries would not attempt to supply this quantity of residual fuel oil if the imports were stopped, and consumers would not have any great difficulty in obtaining substitute fuels although costs might be increased. Many consumers are able and quite often do convert to other fuels when price makes this action advantageous.

Natural gas marketed in the United States still is obtained principally from domestic sources. A special permit has allowed gas to be piped from Canada to one copper smelter in Montana since 1952. A number of proposals are being considered by agencies of this Government and of both Mexico and Canada regarding the importation of natural gas into this country. The additional available markets for natural gas and the rising field prices have spurred pipeline companies to investigate new sources of supply in all possible locations.

This country need not fear a shortage of energy sources in an orderly economy. Given time to make the conversion, this country's foreseeable needs for liquid or gaseous fuels could be supplied from large domestic reserves of coal and oil shale. The technology is being developed for converting these materials into fluid fuels. Only the economics of such conversion prevents their competing with conventional sources, and the known reserves of coal and oil shale are many times the proved reserves of crude oil and natural gas.

DEFENSE PROGRAM

Petroleum is unquestionably a necessity to both peacetime and wartime economy. In time of war suddenly expanded military requirements are added to necessary civilian requirements that must be satisfied at a level to insure an efficient "home front." In both World War II and the Korean emergency it proved practical to have Government petroleum matters centralized as much as possible in one agency. During World War II this agency was the Petroleum Administration for War (PAW) and during the Korean emergency the Petroleum Administration for Defense (PAD).

These agencies were responsible for determining the amount and location of available supplies of petroleum and how they could best be used to meet the needs of the United States and its allies.

The activities of PAW illustrate the problems such an agency must solve. Proved reserves of petroleum and the rate at which they could be produced were determined. A program was formulated that would get the most production from these reserves with minimum consumption of equipment and manpower. New reserves of petroleum had to be found to replace declining production in established fields and to increase output above the existing capacity to produce. Priorities had to be obtained for the oil-field supplies needed.

In World War II transportation was one of the most difficult problems. The diversion of tankers to military duty and the heavy toll taken by submarines of tankers on the gulf coast to east coast run caused a severe shortage of petroleum in the eastern part of the United States. The PAW initiated large-scale rail movements of both crude oil and products to the east coast and authorized a number of pipeline projects, including the Big Inch and Little Inch lines.

War machines require products not consumed in large volume in the civilian economy. During World War II this country's output of 100-octane aviation gasoline was expanded nearly tenfold. To meet the increased demand for this product, new processing plants had to be built. Also the processing at existing refineries had to be changed to produce the required quantities of high-cetane-number diesel fuel and of Navy Special fuel oil.

Disruption of the normal transportation system and the increased wartime demands required that a system of civilian rationing for gasoline and fuel oils be established. PAW recommended distribution plans that would provide equitable distribution of supplies and give no producer or jobber a larger share of the market than he previously enjoyed.

These are some of the problems a defense program must meet. The magnitude of the problems will be less if, at the outbreak of hostilities, there is excess crude-oil productive capacity and spare refining capacity. At the outbreak of World War II the United States had 750,000 barrels a day of excess productive capacity and an equal or larger excess of general refining capacity, but spare refining capacity capable of producing high-octane gasoline was not available. By the end of the war the crude-oil production rate had been increased 1 million barrels per day, and high-octane aviation-gasoline production had risen from 55,000 barrels a day to 520,000 barrels a day.

The magnitude of the defense problem also depends upon the size of the military operations, the military operations of allies, and the quantity of petroleum products provided for the civilian population of such allies. Total demand for all oils in the United States in 1941 averaged 4,060,000 barrels a day. In 1944, with World War II active in both Europe and the Pacific, the total domestic demand for crude oil had increased by 512,000 barrels a day. This increase was the net effect of expanded military consumption, supplies for allies, fully mobilized domestic industrial activity, and allocated domestic civilian consumption.

Under normal peacetime conditions, the Defense Department has the primary responsibility for plans and programs in anticipation of emergencies and war. Such planning is coordinated in large measure with other Government agencies and receives such support as is required from other qualified agencies and the industry. Industry staff for the reactivated Military Petroleum Advisory Board was provided in 1954 after disbandment of PAD.

IMPORT SECURITY

As was discussed under "Self-Sufficiency," this country now does not depend upon petroleum imports, and therefore interruption of these imports would not disrupt the economy. In 1954 approximately 9 percent of the crude-oil supply, or 658,000 barrels a day, was derived from foreign sources. In addition 405,000 barrels a day of products was imported. The reserve crude-oil productive capacity of the United States (about 2 million barrels a day) is considerably greater than the crude imports in 1954. As most of the product imports were residual fuel oil, some of such imports could be replaced by coal. The recent trend shows that imports are increasing gradually, and at some future time the total demand for petroleum may exceed the domestic productive capacity.

If foreign crude-oil production facilities were eliminated, the problem would be to supply the civilian and military needs of possible allies overseas now supplied by this foreign oil production, rather than to meet domestic demand.

About three-fourths of the combined crude-oil and product imports (about 275 million barrels) in 1954 originated in South America and the Caribbean area. These imports entered the United States almost exclusively through east coast ports. Also entering these ports in much larger volume are shipments of crude and products from United States ports on the gulf coast. If a blockade of the east coast were tight enough to prevent the entry of a near normal number of tankers from all sources, the disruption of gulf to east coast movements would be more serious than elimination of foreign oil imports.

Loss of tankers from submarine action and tanker transfer to military service in the early days of World War II drastically reduced the supply of petroleum to the east coast. In spite of tremendous efforts that raised tank-car shipments of oil to the east coast from virtually nothing to almost 1 million barrels daily, increased barge receipts from 20,000 barrels a day to as high as 160,000 barrels, and eventually the building of the "Big Inch" and "Little Big Inch" pipelines with a combined daily capacity of 560,000 barrels, the east coast supply was never adequate to meet combined military and civilian needs. As a result, it was necessary to ration both gasoline and heating oil for civilian use and to convert many industrial users of these products to alternative fuels. The war-built pipelines have been converted to gas service; if they were reconverted to transportation of petroleum or its products (which has been proposed) hardships would be imposed on the gas customers that they serve unless new pipelines replaced the old ones. The tank cars that were utilized for east coast service during the war have reverted to hauling crude oil and products for relatively short distances. It is notable that the number of tank cars available today about equals the number available in 1945, so that they could not supply an equal proportion of the much higher requirements for petroleum products that would result under similar circumstances today.

The major potential wartime problem thus is not possible loss of foreign crude oil and its products but possible loss of transportation facilities from domestic areas of production to the east coast.

STOCKPILING

Following World War II the United States became a net importer of petroleum. During 1954 petroleum imports exceeded exports by about 700,000 barrels a day. This condition exists because of world trade conditions and is not a situation caused by a physical shortage of domestic oil or lack of productive capacity to sustain a large enough flow to meet the peacetime needs.

Petroleum is a highly important strategic material and on a tonnage basis comprised about two-thirds of all material shipments to overseas forces during World War II. Total emergency and war conditions obviously increase the direct military demands for petroleum and also increase the requirements of those segments of the civilian economy that supply the armed forces. Such emergency demands added to essential civilian needs severely tax the capacity of the Nation to produce petroleum at rates consistent with good conservation practices.

One method of providing for an anticipated shortage during an emergency is stockpiling. To provide fully for such a situation of 4 or 5 years duration would require a tremendous storage capacity and large expenditures of money for facilities and oil. Surface tanks can be used, but they are expensive to build, are subject to earthquake, fire and wartime hazards, and some deterioration of the oil or products may occur in storage. Surface steel and concrete tanks are used to provide adequate storage facilities to meet the industry's normal requirements, but these are not large enough to provide space for stockpiling. Existing storage of this type has a volume of about 700 million barrels, and stocks on hand normally are maintained at about half this level. A 10-percent increase in the total storage capacity would add an appreciable volume of oil for short emergencies.

Abandoned mines and quarries, if they can be made leakproof, offer storage capacity at considerably lower cost than surface facilities. An abandoned feldspar mine in Sweden and an old slate quarry in Pennsylvania are now being used to store petroleum. Underground storage reservoirs have been prepared in impervious formations and salt strata for the storage of liquefied petroleum gases. In general, such storage facilities are safer from air attack than steel storage tanks and permit the withdrawal of oil at high rates. Because the volume requirements for a stockpile are large, depleted oil fields also have been considered as potential storage reservoirs. Nearly depleted reservoirs from which 100 or more million barrels of oil has been produced are of the size considered for preliminary studies. The injection of petroleum into a reservoir is a feasible process and has been accomplished in a few special cases. For stockpiling purposes old oilfields offer large reservoir volumes for the storage of oil that is not vulnerable to air attack but is limited in withdrawal rates by the physical characteristics of the reservoir rocks.

The stockpiling of petroleum for an emergency is feasible from a technologic standpoint, and additional technical knowledge is not required to institute such a program; however, the economic aspects do not point to stockpiling as a desirable solution for this national problem. To establish a stockpile of 1.46 billion barrels (1 million barrels a day for a 4-year emergency) would cost about $6 to $7 billion. This estimate includes cost of storage facilities, oil, handling and an interest charge of 2 percent for a 10-year period.

A stockpile of this magnitude, once established, would have a depressing influence on the petroleum price structure. Government policies and changes in those policies relating to such a large volume of stored oil would present problems that could not be evaluated properly by industry. These factors may act in such a way as to weaken the exploration and development programs of the petroleum industry. Such an effect would indeed be unfortunate, as it would in a measure defeat the purpose for which the stockpile was established.

Petroleum has not been stockpiled in excess of operating stocks to meet peak demands in America. Private enterprise has been encouraged, through competitive practices, to explore for and develop petroleum resources in this country. At the beginning of World War II the petroleum industry had a reserve productive capacity of about a million barrels per day. In the postwar period the industry has exerted great effort and has restored this reserve capacity plus additional reserve capacity to offset present net imports. In a severe emergency this reserve capacity to produce plus rationing of civilian supply of petroleum and continuing imports from certain Western Hemisphere sources, such as Canada, Mexico, and Venezuela, are the basis for the expectation that supplies will be sufficient to meet essential needs. Indications are that the current reserve productive capacity of the United States and Canada, alone, is adequate to meet contemplated emergency conditions that might involve this country. Additional transportation facilities will be needed to make available to consumers much of the oil that can be produced from present reserves, especially in Canada. However, this is an important subject involving the security of the Nation and should be reviewed at regular intervals in relation to world conditions, commitments of this Nation, estimate of emergency requirements, and the ability of the Nation's petroleum facilities to meet these requirements.

INDUSTRY CONCENTRATION

The areas of concentrated petroleum production in this country are along the gulf coast of Texas and Louisiana; in West Texas and eastern New Mexico; in the midcontinent area encompassing central Oklahoma, Kansas, and North Texas, in California around Los Angeles, and north of Los Angeles along the coast and in the San Joaquin Valley. In addition, gas not associated with oil is found in quantity in the Texas and Oklahoma Panhandles and in southwestern Kansas.

The location of petroleum refineries is determined primarily by the availability of a supply of crude oil, access to water transportation, and proximity to a concentrated market. Locations situated favorably with regard to at least two of these factors have become the refining centers of this country.

The Texas gulf coast with its prolific oil production and access to low-cost tanker transportation to the populous east coast has two refining areas—Corpus Christi, Tex., and the Houston, Tex.–Lake Charles, La., area. Crude oil for these refineries is generally received via pipeline from gulf coast, East Texas, North Texas, and West Texas-New Mexico fields. Farther east along the gulf coast there is another refining center in the vicinity of New Orleans, La. Here, in addition to ocean shipment via the gulf coast, barge shipment on the Mississippi River is convenient. This area is also served by pipelines that carry refined products throughout the Midwest and to Southeastern states.

The gulf ports also serve as shipping points for crude oil to supply the refining centers on the east coast. Baltimore, Philadelphia, and New York have deep-water harbors and are the centers of concentrated urban populations with a resultant large consumption of oil products. Because of these factors, large refinery capacities have developed around these cities.

Refineries have developed in the Tulsa and Oklahoma City area and in eastern Kansas. In addition to the availability of local oil production these refineries can now draw upon crude received via pipeline from Texas. Products from these refineries serve a sizable local market and also are shipped into the upper Midwest.

Refinery capacity has steadily increased around St. Louis, Chicago, and Toledo. The combination of large local markets and river or lake water transportation make these locations desirable for refineries. In addition, these refining centers have now been interlaced by a system of both crude-oil and products pipelines.

On the west coast, refining centers have developed near Los Angeles and San Francisco—the two major centers of population in the State; both are on the coast and have port facilities, and Los Angeles is in one of the largest crude-oil-producing areas in California. The Pacific Northwest had been supplied with products from California refineries until very recently. The construction of a crude-oil pipeline from Alberta, Canada, to the Canadian west coast and into Washington has motivated the construction of refineries around the Puget Sound area. The possible availability of Middle East crude oil by tanker shipment also was a factor in locating these refineries.

Natural-gas production usually is found in the same areas as crude-oil production. Long-distance large-diameter pipelines connect these producing areas with centers of population in virtually every area of the country except the Pacific Northwest, for which transmission lines are now being planned. Lines originating in the gulf coast, Texas, Louisiana, and Kansas areas extend north, northeast, and east to such centers as Chicago, Detroit, Pittsburgh, New York, New England, Washington, and some of the Southeastern States.

CONSERVATION

Conservation—the wise use of the Nation's oil and gas resources—is a dynamic force that gains strength with the growing store of technical knowledge. The visible waste of oil and gas in production operations and the burning of gasoline in early refining operations in obtaining kerosine and lubricating oil were among the first wasteful practices to be corrected. The waste of gasoline was eliminated by development of the internal-combustion engine and the accompanying demand for gasoline.

Conservation by preventing physical waste of petroleum in production has been difficult to attain because of slow development of knowledge of the technologic factors bearing thereon, coupled with the economic and legal aspects of the problem. However, Federal and State governments and industry, working together with the best available legal and technical experts, have accomplished much to prevent physical waste of these natural resources (*36*). There remains much room for improvement, such as reduction of evaporation losses and gas flaring, and increasing recovery of petroleum from reservoirs to the maximum consistent with related economic factors.

During the long struggle to prevent physical waste of petroleum and the legal battles for State regulation of production to meet market demands, petroleum engineers steadily developed the technical facts and concepts concerning the underground deposits of petroleum. Instead of the early concepts of underground rivers and pools of petroleum, science and technology developed the true concepts by showing the nature of the porous rocks that contain the oil and gas, the presence of the connate water in the rock pores, the nature of gas-oil solutions, and the significance of gas caps, water drives and gas-oil ratios in the production of oil and gas. To these has been added the concept of the natural energy, within the petroleum deposit, that is available for moving the oil through the porous rock to the well bore by (1) expansion of gas from either the gas cap or the oil-gas solution as a result of pressure reduction, (2) encroachment of edge water, and (3) gravity drainage.

Many of these factors, such as the character of the porous rocks, the original contents of oil, gas, and water in the deposit, and the characteristics of these fluids do not lend themselves to control by the oil operator; however, there are some factors that are controllable by the operator. These include the rate of production, more efficient utilization of the natural energy within the reservoir, and in some cases the augmentation of that energy by pressure maintenance through gas or water injection. The conservation or most efficient use of the natural energy and its supplementation by external means have resulted in ultimate yields that are 2 to 3 times greater than those obtained by earlier rule-of-thumb methods. Thus the earlier conservation measures for eliminating physical waste have been improved and carried forward to eliminate the unseen waste of energy within the reservoir. A survey of the Interstate Oil Compact Commission in 1949 showed that application of conservation methods such as controlled rates of production and efficient use of the energy within the reservoir for the movement of oil and gas had resulted in the production of 5 billion barrels of oil in addition to that which would have been produced by the older, less efficient methods. It was predicted that at least 5 billion barrels of oil would be produced in the future through the operation of these improved methods, in addition to the oil that would have been produced by the older methods.

Wise and efficient use of the natural energy within the petroleum deposit to increase oil yields or recoveries is certainly practical conservation. Great strides have been made in this direction during the past 20 years. The goal of perfection may never be attained. Much attention must be given to the problem of working out and adopting plans for operating more fields on a unit basis. Petroleum reservoirs are units created by nature, and for maximum yield of oil and gas they require unit operation to take full advantage of the natural energy within the reservoir for the production of oil.

While significant progress was made in conservation through the application of technologic knowledge to production practices, equal advances were being made in refining. Physical waste was curtailed early in the history of refining. Evaporation losses were reduced by use of light-color paints, floating roof tanks, pressure tanks, and better fire-protection methods. Satisfactory equipment is now available to reduce waste by evaporation to a negligible amount if the equipment is used generally.

Cracking of petroleum to produce lighter fractions has been a technologic development of great significance to conservation. With the rapid increase in the number of automobiles and in highway systems the demands for gasoline increased tremendously. Thermal cracking helped to meet this demand for gasoline with a much smaller throughput of crude oil than would have been required if the old straight-run-distillation method had been used. If the yield of gasoline from crude oil had not been increased by cracking processes in the years following 1925, the United States would have required about one-third more crude oil in 1954 to meet the gasoline needs. This estimate does not consider the improved quality of gasoline resulting from the cracking processes.

Through the years technological developments have been used to improve refining processes. These have been directed toward increasing the thermal efficiency in processing and the manufacture of better products. This has resulted in the development of processes such as catalytic cracking and the manufacture of high-octane gasolines. These modern processes produce gasoline of such quality today that 2 gallons does the work formerly done by 3 gallons (*46*).

Conservation through better refining processes and improved liquid fuels has been accompanied by important contributions from the automotive industry. The modern automobile engine is 50 percent more efficient than the early engines, and technology will make possible a further increase of 50 percent. However, attention must be directed to the fact that these important contributions to conservation by the petroleum and automotive industries have not been used entirely to obtain increased mileage per gallon. Much of the advantage gained through technologic improvements has been utilized by the American public to propel heavier cars with engines having faster acceleration and better performance characteristics, adding to the comfort and pleasure of the traveler. This use of technologic gains has been due largely to the high standard of living in this country and the comparative cheapness of gasoline in relation to that economy. Should the price of gasoline increase substantially in relation to prices in general, industry can produce lighter weight automobiles that will increase substantially the mileage per gallon from gasolines now being made by modern refineries.

The increased use of diesel engines has been an important contribution to conservation. This applies especially to trucking and railroading; the latter, by changing from steam to diesel locomotives has effected a fourfold increase in efficiency of fuel use. This has made it possible for the railroads to use diesel engines to replace coal- and oil-fired steam locomotives and operate with the same number of barrels of oil that were used in firing oil-burning steam locomotives alone. True, the fuel demands have changed from residual fuel oil to distillate or diesel fuel oil, but the refineries have been successful in adjusting their processing to meet this change in demand.

The use of LP-gas as a fuel in truck and tractor engines has been highly successful. Until recent years LP-gas was a waste product burned in flares. Now it is a high-grade fuel so good that a properly lubricated truck engine using this fuel will outwear the truck body. This is a good example of conservation through technologic improvements.

Mention has already been made of physical waste of natural gas. Thirty years ago the waste of natural gas was enormous in the fields of this country. The development of large-diameter, thin-wall, high-tensile-strength pipe and electric welding has made possible the construction of long pipeline systems to transport natural gas from distant producing areas to large metropolitan centers. This has created a large market for this fuel and has been directly responsible for eliminating much waste. Pipelines now receive one-third associated gas (which is gas produced in conjunction with oil operations) and two-thirds natural gas from gas wells. Wastage of gas has been reduced substantially, but 7.5 percent (*17, 1953 ed.*) or more of gas still is wasted in production operations. This may require further application of conservation measures.

Thus conservation of oil and gas has developed rapidly during the comparatively short history of the petroleum industry. The development of scientific and technologic knowledge about these resources and the application of that knowledge to improve operating methods have resulted in true conservation and wise use of these substances. Another important factor has been a favorable economic climate for the operation of free enterprise. This has produced a healthy industry with financial resources for constant modernization, expansion, and improvement of operations. The cooperation of Government and the petroleum industry in working together for such regulation of production as is necessary to attain good conservation practices has yielded rich returns. These are mighty forces working for conservation from which additional contributions to the wise use of the Nation's petroleum and natural-gas resources can be expected in the future.

RESEARCH

The history of the modern petroleum industry is, in a way, a monument to petroleum research. Colonel Drake drilled his well because of research on the properties of samples of Pennsylvania oil. Samples had been analyzed by Profs. Dixie Crosby and Benjamin Silliman, Jr. An extensive report by Silliman, made in 1855, indicated that the best illuminating oil (kerosine) yet known could be obtained from the Pennsylvania oil by distillation; about 50 percent of the crude could be converted to this valuable product. He also reported the availability of good lubricants in quantity and the presence of wax suitable for candles. The rapidly expanding demand for kerosine promised a ready and profitable market for oil if it could be made available in quantity. So the backers of the Drake well made their decision, and the oil industry came into being.

Research soon was called on to solve the first major industry problem when sulfur-containing crude oil was discovered in Ohio in the Lima field. When Herman Frasch demonstrated that Lima crude could be rid of the sulfur by distilling the oil in the presence of copper oxide, the market price of the Lima oil immediately increased sevenfold. Certainly this was a research project that paid off.

Today, the petroleum industry is noted for its dynamic and aggressive research programs. These programs have developed from an understanding of research benefits by industry, educational institutions, and Government. About 10 percent of the total number of scientists and engineers engaged in research and development in the United States are concerned with petroleum and natural-gas problems. The largest number of these are employed directly by the industry, with smaller numbers employed in universities, nonprofit institutions, and State and Federal Governments.

INDUSTRY

The ability of the petroleum industry to develop new and better methods for finding, producing, transporting, refining, and utilizing oil has been a potent factor in the Nation's strength in peace and war. There are no substitutes for liquid fuels in many applications. These new and better methods are the direct results of a highly organized and well-financed research and development program by the industry, taking full advantage of petroleum research conducted by Government, universities, and research institutions.

Some 17,000 scientists, engineers, and their assistants are engaged in petroleum research and development, with a combined annual expenditure of over $130 million. About 20 companies furnish 90 percent of both personnel and funds. The largest company probably furnishes 15 percent of both figures. Some of the smaller companies operate with research staffs of 10 people or less and budgets not exceeding $100,000 a year.

Industry research covers the entire range of petroleum operations. Efforts are directed to new and improved instruments and techniques for finding new oil-bearing reservoirs and oil deposits, better equipment for drilling wells and better fluids for drilling, better methods for completing wells, and more efficient methods for producing the oil from the oil reservoir. Research is concerned with corrosion problems associated with producing, refining, and storing petroleum and natural gas. Research applied to transportation has achieved significant results, and oil and gas move more economically than almost any other commodity. Research to develop new refinery equipment and methods has received far more publicity than the other types of research, with good reason. Research has solved each new problem as it arose, and the results have been easy to see. The United States now is freed from dependence on imported rubber, and the synthetic rubbers now are far better for major uses than the natural. Military-aviation supremacy in World War II was based on the high-octane aviation gasoline developed through research. Research developed processes for increasing the amount of gasoline obtainable from a barrel of crude oil from about 20 percent to over 75 percent, even though gasoline demand does not require over 50 percent. Increased octane number of motor gasoline enables 2 gallons of present fuel to do the work of 3 of the old lower octane gallons.

High-speed, high-output, internal-combustion engines are the result of research efforts in metals, engine design, and fuels and lubricants. Lubricants have received much attention from petroleum-research staffs and have kept pace with engine requirements. Research in both the petroleum and chemical industries has provided an amazing array of pleasing new products that last longer, cost less, and do a better job. Some oil companies now market as many as 2,000 different products, and their research is developing a possible 1,000 new items each year. Only a small number reach the market each year from each company, but the total becomes significant.

As in all industrial research, petroleum research usually is applied research and includes development. It is done with a practical end in view, because the prospect of success is good, and because, if successful, financial rewards will be forthcoming. Most of the research is short term and may be necessary because of advances in associated industries, such as new developments in motorcars or industrial engines. Much research is defensive, in that one oil company cannot stand still as its competitor goes forward with new products developed by research. Industrial research, in general, not only must be self-supporting but must also offer greater financial returns than some other form of capital investment the company might make.

As with other industries, the petroleum industry is well aware that applied research depends on basic or fundamental research for its ideas and foundations. Basic research is the source bed of facts from which applied research can begin. The industry has devoted about 5 percent of its research expenditures in the past to basic research, much of it sponsored through grants to universities and nonprofit research institutions. One company recently announced grants and fellowships of $400,000 for the current year.

Government support of industrial research is the major factor in much research today, with most of the huge research contracts devoted to military and atomic energy problems. In general, the petroleum industry has preferred not to accept such Government contracts and support, in order to retain patent and other rights in research accomplishments. Only a small percentage of industrial petroleum research is financed with Federal money. The industry does work very closely with the military establishment, and some companies have entered into atomic energy operations with the Government. The industry has been willing to devote the required effort on necessary military problems at its own expense.

INSTITUTIONS

Foreign, then domestic, universities were the historical sources of basic research findings and still are the major contributors to such knowledge. The early history of research on the composition of petroleum shows that the major results came from university laboratories. Of the total $3.75 billion research effort in 1954 by all sources, about 8 percent was allotted to university laboratories. Of this 8 percent, less than half can be identified as money for basic research, and only a small part of that can be credited to basic research related to petroleum. Even so, that small part is significant, and the universities are making real contributions to the knowledge of petroleum, its formation, occurrence, properties, and utilization.

In the last 20 years nonprofit research organizations have been organized to conduct sponsored research in all fields, and petroleum has been a major study on such programs. A small number of these institutions have been in existence for a long time, such as Battelle and Mellon Institutes. They have done excellent jobs, and their successes have led to the establishment of others. Some have been founded to promote the development of a particular section of the country through research. Among these are Stanford Research Institute, Midwest Research Institute, and Southwest Research Institute. Many universities, in part as the result of large contracts for Government military research during and after World War II, have set up separate organizations similar to the institutes named above to carry on sponsored research, both basic and applied. Probably 150 or more schools now have research contracts with some Government agency in amounts from several thousand dollars to several million dollars a year. With an increasing need for basic research and an increasing awareness of its usefulness, support of basic research in universities and nonprofit organizations should increase.

GOVERNMENT

Research in Government laboratories and by Government agencies differs markedly in its approach to the task from the approach used by industry and by universities. Some primary responsibilities of the Government include national defense, standards of weights and measures, regulation of interstate commerce, reclamation, and conservation of the public domain. These responsibilities include concern for the general welfare of the Nation. Through the years the Congress has interpreted these responsibilities to include scientific studies relating to many phases of the life of the country, including research in agriculture, minerals, mining, health, and education.

The Bureau of Mines, through its Division of Petroleum, is specifically directed to conduct research on the conservation, development, and better utilization of petroleum, natural gas, and their products. Such research was begun in the Geological Survey even before the Bureau of Mines was established in 1910 and has grown slowly in the intervening years to its present stature.

Unlike industry, Government petroleum research does not have to show a direct, immediate profit, even though such a profit may develop. Bureau research takes a broad nationwide or even worldwide view of the problems to be solved and anticipates needs 10, 20, or 50 years hence. Bureau research likewise is aimed at benefits to all the people, or at solutions of problems properly associated with its own functions or the petroleum functions of other Government agencies. The Bureau of Mines is the technical consultant to all Government groups on petroleum scientific matters. Research is devoted to problems that industry will not or cannot undertake or that will be of immediate or future benefit either to the industry or the public. Efforts can be devoted to solution of those problems that are of industrywide significance, yet their solutions would not be financially attractive to research efforts by a single company.

The broad objectives of the Bureau's petroleum research are to increase the recoveries of oil and gas from known deposits and to increase the efficiency of utilization of these resources after they are produced. These objectives are pursued through 4 research programs and 1 statistical program as follows: (1) The production of oil and gas, both by primary and secondary methods; (2) transportation and storage of oil, gas, and products; (3) chemistry and refining of oil and gas; (4) thermodynamic studies of petroleum hydrocarbons and related materials; and (5) compilation and publication of statistics on oil and gas production and consumption and production and use of petroleum products.

The research studies are conducted at 4 experiment stations and 2 field offices, at Bartlesville, Okla.; Laramie, Wyo.; Morgantown, W. Va.; San Francisco, Calif.; and Dallas and Wichita Falls, Tex. The statistical work is done in Washington, D. C., with a small suboffice in California. The Bartlesville, Okla., station is the largest, with a total personnel of 150. The 2 field offices operate with about 12 people. The total appropriated sum for these 5 programs was $1,250,000 for each of the 1954 and 1955 fiscal years.

The Bureau has cooperated through the years with other Government agencies, State groups, industry associations, and industry companies on problems of mutual interest. The State of Oklahoma has made financial contributions to operation of the Bartlesville station for many years. Work has been accomplished on requests from all of the military departments, and specific problems have been solved under agreements with various companies, to the benefit of the Bureau's program, the public, and the companies. Such organizations as the American Petroleum Institute and the American Gas Association have contributed funds for studies on industry-wide problems of major significance.

The statistical program has vital importance to the public, all segments of Government, all the petroleum industry and other related industries, and the orderly progress of the economy. It has been described as the most complete and reliable coverage of any of the major commodities and receives excellent support and cooperation from the industry.

OUTLOOK

SIGNIFICANT CURRENT TRENDS

The energy demand of the United States has been rising steadily through the first half of this century at an overall annual rate of 3 percent. From 1918 to 1947 the demand for coal was almost constant, and petroleum and natural gas supplied the energy required for expansion. From 1947 through 1954 the demand for coal has been declining, and petroleum and natural gas have furnished both the energy of the replaced coal and the increase in total demand for energy.

A major portion of the reduction in coal demand has been caused by the conversion of American railroads from coal-burning locomotives to diesel-powered trains. This conversion is almost complete, and the rate of growth of diesel fuel demand should therefore become more normal.

Distillate fuel oil for home heating, after displacing coal in both new and replacement installations for many years, has had very strong competition from natural gas in the years following World War II. Gas is now available in many new areas and has been favored for space heating in new home construction because of the lower initial equipment costs and because of the relatively low cost of natural gas as a fuel. The cost of natural gas to the consumer has been rising more rapidly than that of other fuels for several years and has lost some of its competitive advantage in some areas. It is expected that some energy growth that has been absorbed by natural gas for household heating may be taken over by fuel oils.

Gasoline has no competition among nonpetroleum fuels, and its use probably will expand to the extent that the standard of living of the growing population includes automobiles. The planned expansion of the highway systems will increase the usefulness of motor vehicles—including passenger cars, trucks, and buses—and this extension will tend to increase per vehicle consumption, but this may be offset by increasing traffic congestion in urban areas that may reduce use of vehicles.

The long-term trend for petroleum appears to be conversion of a higher proportion of the crude to light products and reduced use of natural gas for industrial purposes. The increasing share of the industrial market will be an area of competition between coal and imported residual fuel oil.

POSSIBILITIES FOR EXPANDED PRODUCTION

The petroleum industry has been expanding ever since its inception, and since World War II this expansion has averaged 4.7 percent a year, based on domestic crude-oil and hydrocarbon-liquid production. The industry has provided a reserve production capacity of approximately 1 million barrels a day of crude oil above current daily production plus net imports. Moreover, there is reserve refining capacity of about 1 million barrels a day. These reserve capacities have been established by the industry as a national security measure and are available for use whenever required by emergency or peacetime demands.

In only 1 year (1943) since 1936, when reliable comparable estimates of petroleum reserves were initiated, have new discoveries not exceeded production. War-imposed conditions were responsible for the single failure. Similarly, for natural gas, new discoveries have exceeded production in every year since 1945, when comparable estimates were started. There is no reason to expect that these trends will be reversed suddenly, and it can be expected that petroleum reserves and productive capacity will be able to meet needs for many years to come.

FORECAST OF SUPPLIES AND REQUIREMENTS

All forecasts of the future supplies of the country's petroleum and natural gas agree that there is a fixed and limited quantity of these resources in the ground and that they cannot indefinitely supply the needs for liquid or gaseous fuels. The great debate concerns the quantity ultimately available and the rate at which it will be available, at what future date the productive capacity will be unable to meet requirements, and the influence of these factors on prices.

Moreover, there is agreement that the consumption of liquid fuels will continue to increase in the foreseeable future. For small-size space-heating installations, as in private dwellings, and for most modes of vehicular transportation liquid or gaseous fuels offer so many advantages that present and anticipated competition of other fuels is not serious.

Between 1925 and 1950 consumption of petroleum products in this country tripled. The President's Materials Policy Commission predicted a doubling of consumption between 1950 and 1975. For natural gas, marketed production increased fivefold between 1925 and 1950. Future consumption and production will depend largely upon the success of future exploration in providing adequate supplies. This country's 1975 economy should easily find use for twice as much natural gas as was consumed in 1950.

New reserves of both petroleum and natural gas have been found in recent years at approximately twice the rate of consumption. To do this, however, wells have been drilled successively deeper, and expensive offshore exploration and development have been undertaken. At the beginning of 1954 proved reserves of petroleum and natural gas were equivalent to 13 and 23 times 1953 production, respectively. This ratio of reserves to consumption has no real significance, except to show the relative size of reserves, because the rate of consumption is increasing each year, new reserves are found each year, and proved reserves cannot be produced to extinction at a constant rate.

Forecasters do not agree on the rate of productive capacity or the size of reserves in the long-term future (over 10 years), but they do agree that changes in both will be gradual and that, with alertness, it will be possible to recognize an approaching scarcity in time to develop other resources to meet the need. The United States is now a net importer of petroleum, not because the domestic capacity cannot supply the demand but because large foreign production has made it advantageous and economically necessary to process imported oil.

As reserves and production diminish, dependence on foreign oil supplies will complicate defense problems. Such a position, while it would be new to this country, has been experienced and met by other countries in peacetime and in war. Great Britain, though almost entirely dependent on imported crude oil and products, is turning more and more to petroleum as a source of energy.

PROBLEMS

The basic petroleum and natural-gas problems are how to continue the expansion of the industry and how to increase efficiency to meet the demands of the people for these resources under the free-enterprise system that permits every customer freedom to select whatever he wants to purchase. Under this system consumption of both petroleum and natural gas will continue to increase, and some authorities forecast that the demand for petroleum will be about 15 million barrels a day and of natural gas approximately 19 trillion cubic feet a year in 1975. Such expansion will require large expenditures of money, huge quantities of equipment and supplies, big crews of workmen, capable managers, and a continuing supply of new technical knowledge from the Nation's research program. The long-range problems include the complete range of activities extending from exploration to utilization of the products in the customers' equipment. These problems, directed toward finding more oil and gas deposits, increasing yields or recoveries from producing properties, reducing waste, improving transportation means, and upgrading and making better products in refineries, must be solved by means that are economically feasible. The solutions of these long-range problems are bounded by limiting economic factors that are imposed by competitive and potentially competitive energy sources.

The objective of exploration is to find more "new" oil and gas, and forecasts of future demands indicate clearly that great effort will be expended on exploration. Some of the resulting problems may be solved by further refinement or improvement of present geological and geophysical methods and instruments. Information resulting from exploratory wells is a powerful tool, making possible a complete restudy of older data for an area; sometimes it leads to new discoveries. The development of new methods, instruments, and tools for exploration would open all possible oil- and gas-productive areas for investigation and reevaluation. Basic research on the origin and accumulation of petroleum, sedimentation, and regional geologic investigation needs to be accelerated. Basic research in physics and chemistry as these sciences join geology in earth studies must be stressed also, for this combined effort on basic scientific problems will furnish the techniques to make the exploratory program successful in finding the oil and gas needed to meet the future demand. Studies in geologic basins, which have not been investigated thoroughly, and drilling to deeper formations in many of the older producing areas are examples of some of the difficult problems facing the industry today.

Development follows discovery and requires large expenditures of money. If petroleum and natural gas are to keep their competitive position and meet anticipated demands in future, research must lead to better drilling equipment, increased drilling speeds, and improved well-completion methods, all to be accomplished with minimum increases in costs.

Considerable research and development have gone into the rotary method of drilling to bring it to its present stage of development. Many problems, including improvement of bits, development of new and better bits, improvements in steel for drill pipe and tool joints, building of more efficient machines for the application of power at the bit, improvements in drilling muds, and a host of other advances are recognized and will receive attention in the future. The great improvements in drilling muds in recent years exemplify the beneficial results of continued research and experimentation. For certain drilling conditions, industry also has developed the use of natural gas, air, and inert gases instead of water or oil-base muds as a drilling fluid.

Although cable-tool drilling largely has been displaced by the rotary method, cable-tool drilling still comprises approximately 10 percent of the total footage in this country. This method should be studied to make substantial improvements in drilling efficiency and convenience of operation for the drilling crew in handling the tools, casing, and other equipment required in drilling and completing wells. In the interest of economy, careful attention should be paid to the selection of areas where cable tools can be used to advantage.

In considering drilling problems, attention is called to the desirability of continuing, with increased effort, the research and development of novel devices for rotary- and cable-tool drilling and entirely new methods of drilling not heretofore developed. A few examples are combinations of rotary and percussion motions to drills, application of sonic and ultrasonic vibrations, and fusion or burning of holes through formations.

The completion of wells presents many problems, and the solution of these problems will result in better mechanical equipment and better productive capacities. The mechanical aspects of well problems include improvements in casing, liners, wellhead fittings, and cements, including the development and application of plastics for special "cement" jobs, plastic tubular goods, and plastic coating of steel pipe to combat corrosion.

Completion practices also need intensive study, with the objective of obtaining better or more productive wells, both from the standpoint of initial production rate and that of ultimate production of petroleum over the life of the well. This includes the problem of eliminating damaging filtration and plugging materials from drilling muds while drilling the oil- and gas-productive zone. Methods for increasing the productive capacity of wells, such as acidization, hydraulic fracturing with water, acid, or other fluids such as light oils, and shooting, present problems in refinements, increased application, and the development of new methods to accomplish better the objectives of these techniques.

Both well drilling and completion can benefit greatly through studies of a fundamental nature on the physical properties of sedimentary rocks. These studies should include determination of such characteristics as compressive strength and coefficients of elasticity.

The production of petroleum presents a very large number of complex long-range problems; their solution will result in better operating methods and increased recovery of oil and gas, thereby increasing the yield per well or per acre. Problems relating to production can be solved by both basic and applied research, by using the scientific data of the former to solve specific problems in applied research. A very broad basic research program is required to fill the needs if the long-range demands for natural gas and petroleum products are to be met. Examples include phase-relationship studies of constituents in reservoir fluids, surface and interfacial forces within the reservoir, rock characteristics such as porosity and permeability to single- and multiple-phase flow, tortuosity, clay and cementing substances, characteristics or properties of the oil-gas solutions, multiphase flow through porous mediums and in well bores, and transient flow conditions in oil and gas reservoirs under producing and depletion conditions. These examples are intended merely to illustrate the type of problems involved and do not, by far, constitute complete coverage of the problems in applied and basic research.

Complex problems exist in the field of applied production research and in production operations. Only a few examples will be given to illustrate their broad scope. These problems start with selection of an optimum well-spacing program for a new field and end with selection of ways and means for satisfactory plugging of the wells when the reservoir is finally depleted to the economic limit and abandoned. During the productive period there are problems of controlling gas-oil ratios to conserve reservoir energy, ascertaining the type of natural control or controls the reservoir is operating under, consideration of the application of pressure-maintenance, and if possible the operation of the reservoir as a unit for maximum recovery. Many problems remain to be solved on secondary-recovery projects employing gas injection and water-flooding technology for bringing the oil to the surface of the ground. These recovery methods will be improved further, and research should produce new and more efficient types of equipment. With oil being produced from greater depths, this becomes an increasingly difficult problem. Additional production problems include wax removal, treatment of emulsions, and efficient separation of oil and gas to conserve the highly volatile liquids in the crude oil.

Research is being conducted continuously on possibly the most difficult problem in the production end of the industry, that being to develop a method whereby the petroleum, or at least a part of it, that now remains in depleted oil-bearing rocks, even after secondary-recovery methods, will be recovered by improved production practices. Important strides have been made during the past 30 years in increasing efficiency in recovering oil—the yields having been improved in some instances by 2 or 3 times the earlier recoveries; however, there is still much room for improvement. Employment of the best modern production methods often results in 50 percent or more of the original oil in the reservoir being left when the economic limits of production are reached. The problem of recovering at least a part of this residual oil is indeed important and a challenge for the very best effort of both basic and applied research organizations.

Transportation of crude oil, petroleum products, and natural gas presents many problems involving the construction of pipelines, tankers, barges, and railroad tank cars. Each of these has many problems peculiar to that specific method of transportation. For instance, tankers have problems associated with the building of bigger and speedier ships. Pipelines have problems resulting from the use of larger diameter, thinner walled pipe and operation at higher pressures; improvements in pumps, compressors, and power plants are constantly being made. Communication systems for transportation by all methods are constantly being improved, and further progress in this field can be expected as a direct result of research in the electronics field. Transportation also presents opportunity for basic research in the flow of fluids in pipes.

Many important problems relating to storage have been solved during the past quarter of a century, but problems remain if petroleum, natural gas, and their products are to be conserved adequately after being produced and supplied to consumers at desired rates to meet their requirements. For instance, the underground storage of natural gas to meet peak demands, to allow for more efficient use of pipeline systems and to prevent waste of gas produced in association with oil presents many problems. Their solution will be a significant contribution to conservation. It is anticipated that underground storage chambers will be mined or formed by solution of underground salt deposits, old abandoned mines conditioned, and abandoned pits or quarries utilized in constructing large storage facilities especially for the storage of liquefied petroleum gases and other liquid petroleum products. Use of these types of storage facilities will be attended by many problems. Attention is also directed to the numerous problems remaining to be solved in connection with steel storage tanks, high-pressure steel storage facilities, and concrete tank storage. Problems involved in storing liquefied natural gas should be solved, so that these liquids may be transported, especially by water, over great distances and stored safely near cities. Solving these problems will permit great volumes of flare gas to be conserved in remote areas that are accessible to marine transportation and will provide distant cities with natural gas.

Fundamental research has an important place in supplying scientific data to solve many storage problems. Further research will make highly valuable contributions to the solution of problems of incompatibility, instability, gum formation, and other deterioration of fuels in storage. It will aid greatly in solving problems associated with the failure of steel in line pipe, tankers, and storage tanks; basic research on low-temperature effects on metals will contribute to the solution of storage problems.

The refining industry has a great many difficult problems that require solution if the anticipated long-range demands for petroleum products are to be met. These demands will undoubtedly be satisfied in future, as in the past, by employing technology to improve processing and efficient operation plus manufacture of better products so that fewer barrels of crude oil and hydrocarbon liquids will be needed to meet future demands. This will require upgrading raw materials and stocks to make better products that will "go farther" for consumers.

A highly useful approach in solving the numerous problems associated with these broad objectives in refining is basic or fundamental research to produce complete information on the multitude of constituents in the raw materials and stocks passing through the refining processes, plus similar information for the finished products. A wealth of scientific data of this kind has been produced in the past century by laboratories, but each new contribution adds other problems to the list for solution. In fact, much more work is needed on the nature of crude oil. This includes chemical and thermodynamic research to ascertain the compounds present in crude oils and their chemical, physical, and thermodynamic properties. Not all of these data have been obtained on the gasoline fractions, a large number need to be obtained on the middle distillates, and virtually nothing in the way of basic research of this type has been done on lubricating-oil fractions and asphalts. The latter two now are processed to meet service conditions largely by rule-of-thumb methods. It is evident that much pioneering research for basic scientific facts remains to be done. Equally important is basic research on utilization of petroleum products to guide research in solving processing problems. Examples are research on the flame-combustion processes under wide ranges of conditions and the phenomena involved in lubrication. All answers to such basic problems will be of great value in solving refining problems. They will help to determine what the raw materials are and what characteristics the products should have for optimum utilization.

From this fundamental research will come a great quantity of scientific data that will be of material assistance in solving problems in the petrochemical industry. Not all of these data can be produced in time to satisfy the needs of the processing and manufacturing operations and therefore the process-development and plant-operating units will be compelled to improvise, estimate, and use experience factors to some degree.

Undoubtedly refineries will continue to have the problem of upgrading raw materials and feed stocks for a long time because of the demand for more distillate fuels and gasoline. This will result in the production of less residual fuel oil and better utilization of the heavy gravity crude oils. The increased demand for distillate fuel oils and jet fuels will accentuate the so-called "middle-of-the-barrel" problems. Certainly the upward trend of octane numbers for motor gasoline will continue for some time, and this, too, presents problems to refiners.

To meet the increased demand for more specialized products the industry has constructed a large amount of equipment that operates in a specialized manner with rather restricted limits for optimum operation. This is highly desirable, but one must not overlook the equally important problem of flexibility in refining operations. The increased number of units required to meet the anticipated increased demand may offer some degree of flexibility in operation of the plants as a whole, and development of new and better processes may also offer a solution.

Obviously there is the basic problem of developing new and better processes, both for refining petroleum and for making petrochemicals. High-sulfur crude oils, trace metals, and nitrogen fouling of catalysts and the production of cheap, high-pressure hydrogen introduce problems that have direct bearing on process and equipment development.

Refineries and petrochemical plants in their natural development must increase efficiency and produce better products. In operating equipment of this type there is always the need for improving thermal efficiency. This never ceases. Also there is the equally important need for increasing efficiency from the reaction and thermodynamic standpoint. The latter is of special consideration in developing new processes and determining optimum conditions for operating units now developed.

The drilling, producing, transportation, storage, and refining departments of the industry have several difficult problems that are common to all. These are illustrated by problems such as prevention of corrosion, reduction in evaporation losses, prevention of waste in general, salvage of materials, and prevention of pollution of air and streams.

This concludes a discussion of the problems associated with the commodities petroleum and natural gas. No attempt has been made to list completely even the more important problems; all examples were mentioned for illustrative purposes only. The discussion has moreover been limited to scientific and technologic problems only, and the reader should not overlook the fact that a great many other difficult ones in the legal, economic, management, and other functional fields of activity require solution. Moreover, there is the manpower problem. From this discussion, it is evident that thousands of long-range problems and tens of thousands of day-by-day technical problems should be solved in connection with the industry's operation. Although tremendous progress has already been made in solving technical problems, those of today and of each succeeding day become more complex. Each solution adds to the fund of knowledge but the increasingly difficult problems impose more and more strain on the research organizations of corporations, universities, research foundations, and Government laboratories. Undoubtedly one of the most important to be solved is procurement of adequate technical manpower to solve the many worthwhile problems now awaiting solution.

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PHOSPHATE ROCK

By

E. Robert Ruhlman[1]

ALL LIFE requires phosphorus for its existence. Many soils are either naturally deficient in phosphorus for maximum plant growth or have become so under erosion, leaching, or intensive cropping. Many soils would be forced out of crop production within a short time without the addition of phosphorus-containing fertilizers.

Summary

Phosphate rock is a natural rock containing one or more phosphate minerals, usually calcium phosphate, of sufficient purity and quantity to permit their use, either directly or after concentration, in the manufacture of commercial phosphatic products.

Phosphate-rock deposits occur in many countries throughout the world, but the larger known occurrences and those now of greatest economic importance are in North America, Europe, and Northern Africa. In the United States, phosphate rock is being mined in Florida, Tennessee, Idaho, Montana, Utah, and Wyoming.

Phosphate rock has diverse agricultural and industrial uses. The quantity used domestically for agricultural purposes (about 60 percent in 1953) has always exceeded that used for all other purposes. The consumption of phosphate rock largely follows that of fertilizers, for which there has been a greatly accelerated demand in recent years. The continuation of this trend is expected, with a resultant increased demand for phosphate rock.

Phosphate rock is mined mainly by open-pit methods, although underground mining is becoming increasingly important in the Western States. Most of the Florida and Tennessee phosphate rock requires beneficiation before utilization by industry. In the Western States acid-grade rock occurs as workable beds and does not require beneficiation. There are three fundamental methods of decomposing phosphate rock to obtain products suitable for fertilizer and industrial purposes: (1) Acid treatment; (2) thermal reduction; and (3) thermal treatment without reduction.

The United States, with its huge reserves, is more than self-sufficient in phosphate rock. Productive capacity is believed to be adequate to meet any expected increased demand, which may reach 4.8 million long tons of P_2O_5 by 1960 and 8 million tons of P_2O_5 by 1975.

Phosphate rock is not on the Government's list of strategic and critical materials; but elemental phosphorus is an important war material, and phosphatic fertilizers are essential to maintain and expand the agricultural output of the United States.

Important problems confronting the phosphate industry include conservation, development of new and improvement of present mining methods and beneficiating techniques, more complete recovery and utilization of byproducts, and economic methods of producing more concentrated phosphates for fertilizers.

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[1] Commodity-industry analyst, Bureau of Mines

BACKGROUND

Phosphate rock is a natural rock containing one or more phosphate minerals, usually calcium phosphate, of sufficient purity and quantity to permit its use, either directly or after concentration, in manufacturing commercial products. The term "phosphate rock" includes phosphatized limestones, sandstones, shales, and igneous rocks. It occurs as nodular phosphates, residual weathered phosphatic limestones, vein phosphates, and consolidated and unconsolidated phosphatic sediments.

The phosphate-rock industry in the United States, which began with a production of about 2 long tons of P_2O_5 content in 1867 in South Carolina, has expanded from about 450,000 long tons of P_2O_5 in 1900 to nearly 4,000,000 long tons of P_2O_5 in 1953. Florida began to produce in 1888, Tennessee in 1894, and the Western States in 1906.

Because of the variable mineralogical makeup and the variable proportions of the minerals present, phosphate rock does not have a definite chemical composition. Most phosphate rock belongs to the apatite group of minerals and can be represented by the formula $Ca_{10}(PO_4CO_3)_6(F,Cl,OH)_2$. The PO_4 radical can be partly replaced by small quantities of VO_4, AsO_4, SiO_4, SO_4, and CO_3 or CO_4. The (F,Cl, OH) radical may be all fluorine, chlorine, or hydroxyl ion or any combination thereof. Also, small quantities of calcium may be replaced by magnesium, manganese, strontium, lead, sodium, uranium, cerium, yttrium, and other rare earths. All domestic phosphate rock contains 3 to 4 percent fluorine. The major impurities include iron as limonite, clay, aluminum phosphates, and silica as quartz sand. Chemical analyses usually are reported as CaO and P_2O_5 or tricalcium phosphate ($Ca_3(PO_4)_2$) also known as B. P. L. (bone phosphate of lime) (1.0 tricalcium phosphate (BPL) = 0.458 phosphoric acid (P_2O_5)) (*8*).[2]

GEOGRAPHIC DISTRIBUTION OF PHOSPHATE-ROCK INDUSTRY

Deposits of phosphate rock occur in many countries throughout the world, but the larger known occurrences—and those of most present economic importance—are in the Northern Hemisphere—in North America, Europe, and Northern Africa.

Phosphate-rock deposits have been reported in 23 States.

Deposits that have yielded phosphate rock in commercial quantity are in the following States: Alabama, Arkansas, Florida, Georgia, Idaho, Kentucky, Montana, New Jersey, New York, North Carolina, Pennsylvania, South Carolina, Tennessee, Utah, Virginia, and Wyoming. In several of these States mining has been intermittent and the tonnage recovered from each occurrence relatively unimportant. The phosphate-rock deposits now commercially important are in Florida, Tennessee, Idaho, Montana, Utah, and Wyoming.

GEOLOGY AND OCCURRENCE OF DOMESTIC DEPOSITS

Phosphatic materials were deposited in certain areas of several of the great continental seas that invaded the interior of what is now the United States in Paleozoic time and in the fluctuating Atlantic coastal zone in Cretaceous and later times, following the original invasion of that region from the east by the Atlantic Ocean during the Cretaceous.

Florida.—The commercially important hard-rock and land-pebble phosphate-rock deposits of the Atlantic Coastal Plain of Florida were formed from low-grade marine phosphatic limestone but by different geologic processes. The hard-rock deposits resulted from subaerial weathering and leaching of the phosphatic limestone and irregular deposition of the phosphatic materials, with replacement of the underlying limestone and coating of various rocks by calcium phosphate. The land-pebble deposits were formed by marine plantation with reconcentration of the phosphatic materials by wave action at various stages of subsidence or elevation.

Hard-rock phosphate occurs in Florida in localized deposits irregularly distributed in an ore of sand, clay, and soft phosphate up to 100 feet thick overlain by up to 50 feet of overburden. The deposits range in extent from small areas less than 100 feet across to those underlying several acres. High-grade masses range in size from less than 1 ton to several tons. In the workable deposits the hard-rock

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

phosphate forms only 25 to 30 percent of the material. The hard rock is generally light-colored and dense and has a phosphate content up to about 38 percent P_2O_5.

Farther south, the main body of commercial *land-pebble phosphate rock* lies in a shield-shaped area around the towns of Lakeland, Plant City, Bartow, and Fort Mead. This area is some 65 miles from north to south and 30 miles across. The unconsolidated phosphate-rock ore in the operating fields averages 30 feet in thickness, with an overburden 5 to 20 feet deep. The P_2O_5 content of the ore varies areally, averaging 15 percent. The phosphate rock recovered ranges in size from plus-14-mesh to pebbles about 1½ inches in diameter. Most of the phosphate rock, after beneficiation, contains 30 to 32 percent P_2O_5 and 1½ to 3 percent combined iron oxide and alumina, although some of it is lower grade.

Soft phosphate rock is a tan or gray colloidal phosphatic clay containing variable quantities of P_2O_5. This type of material commonly is associated with both the land-pebble and hard-rock phosphate-rock deposits.

Tennessee.—Three types of phosphate rock—brown, blue, and white—occur in Tennessee. At present only the brown-rock phosphate is commercially important. The brown rock is the product of the residual weathering and leaching of the outcrops of phosphatic limestones. The brown-rock deposits overlie the parent phosphatic limestones. In some places the brown-rock deposits are exposed at the surface, but elsewhere they are covered by an overburden of clay and soil, usually less than 20 feet thick. Brown rock occurs mostly as loose, rounded grains of calcium phosphate. Although designated as brown rock, the color may range from white to dark brown.

The brown-rock beds do not exceed 32 percent P_2O_5, averaging in general about 22 percent P_2O_5. Workable low-grade material ranges between 16 and 24 percent P_2O_5.

Western States.—The Western phosphate field includes the deposits in Idaho, Montana, Nevada, Utah, and Wyoming. In this region the phosphate rock lies in regular beds at 2 horizons, 1 in the Permian and the other in the Mississipian. Thus far most of the production has come from the Permian deposits.

The whole region has been intensely folded and faulted and then eroded, exposing the phosphate-bearing formations in narrow bands along the flanks of folds or along the borders of the faulted areas. The beds dip at any angle from nearly horizontal to vertical.

The currently economic Permian phosphate-bearing formation is of marine origin. The phosphate beds are contained in the phosphate shale member of the Phosphoria formation, which is 75 to 180 feet thick and consists of yellowish to brown phosphatic sandstones, dark brown to black phosphatic shales, brown to black limestone beds, and 1 to 3 economically valuable beds of gray, brown, or black oolitic phosphate rock cemented by minor quantities of the same phosphatic material. The richest beds of phosphate rock range from 3 to more than 7 feet thick with the phosphate content ranging from 26 to over 36 percent P_2O_5.

RESERVES

The most recent published domestic reserve data, by the Geological Survey, are shown in table 1.

Estimates of the world reserves of phosphate rock by K. D. Jacob are shown in table 2.

TABLE 1.—*Reserves of phosphate rock in the United States,*[1] *1953*

[Million long tons]

Source	Inferred reserves minable under present conditions		Additional inferred reserves minable under changed conditions	
	Marketable product	P_2O_5 content	Product	P_2O_5 content
Arkansas	---	---	20	5
Florida	2, 040	660	23, 350	4, 932
South Carolina	---	---	9	2. 4
Tennessee	85	15	5, 398	1, 129
Western field [2]	3, 000	870	20, 000	5, 800
Total United States (rounded)	5, 100	1, 500	49, 000	12, 000

[1] McKelvey, V. E., and others, Domestic Phosphate Deposits: Geol. Survey, Open File Rept., Mar. 30, 1953.

[2] Includes Idaho, Montana, Utah, and Wyoming.

TABLE 2.—*World reserves of phosphate rock and apatite (4)*

Country:	*Million metric tons*
French Morocco	21, 000
United States	13, 526
U. S. S. R.	7, 568
Tunisia	2, 000
Algeria	1, 000
Brazil	572
Islands of Pacific and Indian Oceans	182
Egypt	179
23 other countries	670
Total	46, 697

MINING

Open-pit mining of phosphate rock is carried on in all three producing fields—Florida, Tennessee, and the Western States. Underground mining is limited to the Western States.

In the land-pebble deposits of Florida stripping of the overburden and mining of the ore (matrix) are done by large, electric, dragline excavators equipped with buckets up to 26 cubic yards in capacity and booms over 200 feet long. The ore is dumped into a sump and pumped to the washing plant, in some instances several miles from the mines.

In the other Florida deposits, the Tennessee field, and the Western field, the ore is mined by smaller draglines or shovels and trucked to the plant. Top slicing, sublevel stoping, and open-stoping methods are used in the western underground mines.

WASHING AND BENEFICIATION

Much of the Florida and Tennessee phosphate rock ore must be treated before utilization by industry. Washing is accomplished by sizing screens, log washers, various types of classifiers, and mills to disintegrate the large clay balls. The fine slimes, usually minus-150- or 200-mesh, are discarded as tailings. In the Florida land-pebble field the plus-14-mesh material is dried and marketed as high-grade rock or blended with the fine granular material. The fine granular material, minus-14- plus-200-mesh, is treated in flotation cells, spirals, or cones.

In beneficiating phosphate rock, the losses in washer operations range from 40 percent of the P_2O_5 in the Florida land-pebble field to more than 50 percent in some areas of Tennessee. The wasted phosphate, in a particle size generally less than 200-mesh, not only entails the loss of a valuable resource but incurs a large expense to the industry in disposing of these slimes. Many hundreds of acres of land are occupied by old ponds, and additional pond areas are being acquired for current operations at higher cost. Washer slime, comprising about one-third of the mined tonnage, is discharged as a pulp with only 8 to 10 percent solids. In about 2 years this material settles to a mass containing about 25 percent solids. The solids contain 15 to 17 percent P_2O_5, nearly the same as in the original ore.

Most of the Western field phosphate-rock production is of suitable grade as it comes from the mine. However, two companies now are beneficiating part of their production.

SUPPLY AND DEMAND

United States.—The phosphate-rock industry in the United States has experienced three distinct periods, each with its own peculiar characteristics, reflecting the economic, financial, and political conditions of the world. The first period extended from the opening of the South Carolina deposits in 1867 to the outbreak of World War I. The second period included the years between World War I and World War II. The third period, which began with World War II, is still in progress.

The first period was one of rapid growth, with relatively minor annual fluctuations in the quantity of phosphate rock produced, reflecting a generally stable world economy. In the second period the world's economic position was unstable, and growth of the phosphate industry virtually stopped. The average trend of the production in this period was nearly horizontal, but wide annual fluctuations occurred, showing as much as 2 million long tons difference between the output of successive years. Domestic production followed closely the widely varying economic conditions of this period. Since the beginning of World War II, the marketed production has advanced at a rate of expansion slightly greater than that which prevailed during the pre-World War I period and has already reached a level over twice as high as that attained in the first 70 years of the domestic phosphate-rock industry. Most of this expansion represents an increase in domestic consumption.

World.—Four areas supply most of the phosphate rock produced in the world (see table 3). These are the United States, North Africa, Russia, and the islands of the Pacific and Indian Oceans.

TABLE 3.—*World production of phosphate rock, 1929–53*

[Metric tons]

	1929	1930	1931	1932	1933	1934	1935	1936	1937
North America:									
Canada	1,075	36	---	1,194	2,008	73	169	476	91
Curaçao	103,289	87,497	73,774	65,407	85,550	100,627	90,709	78,131	101,837
United States	3,883,372	4,014,970	2,619,033	1,725,488	2,397,625	2,944,900	3,210,193	3,518,589	4,330,025
South America:									
Brazil	---	---	---	---	---	---	---	---	---
Chile	---	---	---	---	---	---	---	---	---
Europe:									
Belgium	40,330	40,380	49,100	25,810	25,130	14,385	173,360	16,090	---
Estonia	8,352	4,850	4,580	1,133	8,950	10,610	11,642	11,408	10,112
France	189,000	159,800	107,980	82,700	76,650	66,800	49,600	55,000	103,600
Germany	---	---	---	---	---	735	180	1,060	3,314
Ireland	---	---	---	---	---	---	---	---	(2)
Spain	7,626	5,400	7,734	9,980	14,507	19,297	---	---	10,702
Sweden	---	---	---	---	---	---	2,960	6,140	4,917
U. S. S. R.	45,000	224,000	330,000	156,500	213,400	382,800	767,900	920,000	900,000
Asia:									
British Borneo	---	---	---	---	---	---	---	---	---
China	---	8,000	8,000	8,000	8,000	8,000	8,000	8,000	8,000
Christmas Is. (Indian Ocean)	119,756	121,858	66,906	85,548	92,745	129,780	149,341	157,564	154,378
India	22	308	111	123	38	60	104	130	169
Israel	---	---	---	---	---	---	---	---	---
Japan	14,573	27,713	21,148	18,707	34,739	56,500	91,248	113,102	100,000
Jordan	---	---	---	---	---	---	---	---	---
Philippines	1,492	---	260	830	3,097	20,406	1,309	497	750
Africa:									
Algeria	747,035	846,646	564,898	569,279	587,753	532,210	603,863	530,998	631,148
Angola	---	---	---	---	---	---	---	---	---
Egypt	215,311	313,478	257,011	349,780	440,632	437,933	473,896	531,039	517,002
French Morocco	1,608,249	1,779,008	900,731	987,317	1,107,333	1,266,796	1,303,182	1,257,796	1,501,767
French West Africa	---	---	---	---	---	---	---	---	---
Madagascar	13,441	11,150	8,000	7,100	13,100	8,340	6,000	5,349	4,290
Seychelles Islands	12,789	15,977	4,730	14,213	12,307	12,062	10,082	23,942	9,594
South-West Africa	---	---	---	---	---	---	---	---	---
Tanganyika Territory	---	---	---	---	---	208	194	---	104
Tunisia	2,511,000	3,326,000	2,148,000	1,678,000	1,810,000	1,766,000	1,500,000	1,488,000	1,771,439
Uganda	---	---	---	---	---	---	---	---	---
Union of South Africa	---	---	1,906	1,183	1,181	77	---	---	---
Oceania:									
Angaur Island	65,494	56,345	60,202	65,609	66,492	72,148	78,112	89,226	90,654
Australia	71	26	619	883	97	210	239	178	20
Makatea Island	242,990	176,075	111,422	120,650	79,045	77,470	130,353	122,936	166,726
Nauru Island Ocean Island	585,844	512,265	392,172	438,466	670,898	565,522	707,051	965,349	1,024,168
World total [1]	10,480,000	11,800,000	7,750,000	6,420,000	7,750,000	8,500,000	9,370,000	9,900,000	11,450,000

	1938	1939	1940	1941	1942	1943	1944	1945
North America:								
Canada	189	142	325	2,256	1,147	1,316	437	271
Curaçao	99,283	N. A.	6,047	111,995	24,000	24,000	7,813	8,770
United States	3,922,630	4,052,176	4,133,573	5,000,069	4,896,282	5,456,155	5,283,462	5,486,404
South America:								
Brazil	---	---	---	---	---	6,111	5,216	7,463
Chile	---	9,014	32,000	29,123	24,909	37,924	14,376	13,203
Europe:								
Belgium	---	---	280	36,660	103,300	108,900	52,270	17,990
Estonia	13,012	---	---	---	---	---	---	---
France	N. A.	---	24,260	98,660	113,400	131,020	92,966	75,459
Germany	3,221	1,523	1,548	1,043	829	585	1,000	500
Ireland	(2)	(2)	(2)	(2)	16,444	17,252	19,978	22,110
Spain	23,429	10,320	15,781	13,846	15,232	15,722	17,770	20,349
Sweden	6,192	6,267	7,479	31,116	79,395	144,265	160,847	171,127
U. S. S. R.	1,000,000	1,618,000	2,600,000	2,000,000	1,016,000	1,016,000	1,016,000	1,626,000
Asia:								
British Borneo	---	---	---	---	---	---	(2)	(2)
China	8,000	8,000	---	---	---	---	---	---
Christmas Is. (Indian Ocean)	162,425	177,972	241,826	---	---	---	---	6,096
India	23	185	119	81	337	1,215	232	532
Israel	---	---	---	---	---	---	---	---
Japan	---	---	1,400	1,300	1,400	201,178	52,835	(2)
Jordan	---	---	---	2	2,818	5,384	4,961	4,867
Philippines	---	---	---	---	---	---	---	---
Africa:								
Algeria	584,452	499,920	370,890	446,535	334,550	76,798	220,349	401,304
Angola	---	---	---	---	---	---	---	---
Egypt	458,404	547,538	183,464	111,708	328,440	315,566	318,185	349,374
French Morocco	1,447,544	1,702,973	687,310	493,060	715,410	806,171	1,444,902	1,654,120
French West Africa	---	---	---	---	---	---	---	---
Madagascar	5,699	7,250	495	---	---	---	---	---
Seychelles Islands	21,703	23,545	14,613	18,795	22,347	1,849	5,941	7,090
South-West Africa	---	---	869	---	59	164	964	27
Tanganyika Territory	69	132	9	33	25	267	28	9
Tunisia	1,934,200	1,627,912	1,187,823	1,076,118	879,743	334,678	501,990	706,404
Uganda	---	---	---	---	---	---	---	8,648
Union of South Africa	---	---	---	425	9	5,801	21,088	27,342
Oceania:								
Angaur Island	105,578	(2)	(2)	(2)	(2)	(2)	26,417	(2)
Australia	498	17	313	8,649	14,300	13,141	6,418	9,344
Makatea Island	102,941	160,680	164,170	192,235	176,065	188,385	203,300	259,000
Nauru Island Ocean Island	1,184,816	1,244,170	1,263,385	---	---	---	---	---
World total [1]	11,180,000	11,818,000	11,000,000	9,745,000	8,896,000	9,044,000	9,566,000	10,957,000

See footnotes at end of table.

TABLE 3.—*World production of phosphate rock, 1929–53*—Continued

[Metric tons]

	1946	1947	1948	1949	1950	1951	1952	1953
North America:								
Canada	52	---	---	18	117	5	---	---
Curaçao	78, 675	79, 229	58, 827	92, 784	104, 240	107, 144	106, 902	96, 035
United States	7, 283, 899	9, 257, 220	9, 538, 840	9, 019, 957	11, 292, 541	10, 947, 971	12, 224, 314	12, 704, 516
South America:								
Brazil	10, 421	5, 592	2, 667	4, 553	13, 850	(2)	17, 959	(2)
Chile	15, 210	13, 994	59, 529	49, 311	13, 437	37, 182	26, 417	50, 000
Europe:								
Belgium	69, 927	58, 045	68, 938	44, 643	50, 846	129, 065	58, 983	35, 896
Estonia	---	---	---	---	---	---	---	---
France	97, 285	104, 068	84, 580	59, 643	73, 752	110, 000	102, 000	73, 000
Germany	[3] 400	[3] 698	[3] 473	---	---	---	---	---
Ireland	12, 189	10, 780	---	---	29, 000	25, 000	(2)	(2)
Spain	18, 608	20, 204	23, 012	23, 093	24, 080	22, 830	23, 474	21, 862
Sweden	50, 730	7, 696	1, 441	1, 604	2, 044	9, 013	21, 422	(2)
U. S. S. R	1, 626, 000	2, 032, 000	2, 336, 915	2, 540, 125	2, 540, 125	2, 794, 000	3, 000, 000	3, 000, 000
Asia:								
British Borneo	(2)	(2)	427	508	653	659	707	642
China	---	---	20, 000	20, 000	20, 000	20, 000	20, 000	20, 000
Christmas Is. (Indian Ocean)	34, 444	106, 765	108, 311	255, 236	320, 423	338, 693	354, 762	284, 689
India	247	867	1, 132	588	3, 074	423	452	(2)
Israel	---	---	---	---	---	297	17, 200	23, 092
Japan	7, 985	6, 802	3, 590	684	258	143	---	---
Jordan	4, 024	6, 058	4, 000	---	---	6, 635	24, 941	28, 700
Philippines	---	---	---	10, 998	32, 606	4, 821	4, 231	640
Africa:								
Algeria	584, 827	713, 790	670, 591	648, 202	684, 657	776, 575	702, 587	602, 753
Angola	---	---	(2)	(2)	1, 033	943	---	---
Egypt	294, 046	371, 226	377, 005	350, 480	397, 207	499, 976	522, 214	484, 176
French Morocco	2, 783, 580	2, 960, 735	3, 226, 700	3, 693, 000	3, 872, 241	4, 716, 800	3, 953, 100	4, 156, 000
French West Africa	---	---	3, 965	5, 675	11, 035	23, 580	43, 150	52, 400
Madagascar	---	---	---	---	---	---	1, 305	1, 556
Seychelles Islands	21, 397	14, 516	21, 722	14, 171	10, 005	4, 547	11, 120	8, 859
South-West Africa	1, 665	2, 223	1, 038	957	581	785	1, 675	1, 604
Tanganyika Territory	279	220	313	157	468	459	169	151
Tunisia	1, 399, 880	1, 759, 236	1, 863, 710	1, 441, 918	1, 524, 800	1, 678, 905	2, 264, 641	1, 718, 530
Uganda	7, 213	7, 269	---	---	467	2, 242	5, 010	5, 448
Union of South Africa	37, 691	41, 831	39, 656	56, 471	51, 844	81, 840	96, 568	80, 125
Oceania:								
Angaur Island	59, 557	107, 898	76, 713	157, 049	137, 000	144, 843	83, 905	112, 524
Australia	20	5, 402	2, 170	11	1, 653	8, 056	5, 623	(2)
Makatea Island	241, 085	196, 430	187, 344	265, 082	270, 300	227, 858	213, 555	250, 511
Nauru Island	88, 244	390, 062	544, 298	802, 070	1, 070, 358	942, 945	1, 164, 038	1, 178, 364
Ocean Island			126, 854	265, 087	251, 218	256, 451	249, 542	286, 894
World total [1]	14, 878, 000	18, 348, 000	19, 500, 000	19, 850, 000	22, 800, 000	24, 000, 000	25, 500, 000	25, 500, 000

[1] Partly estimated and rounded. [2] Data not available, estimate included in total. [3] West Germany only.

STRUCTURE OF PHOSPHATE-ROCK INDUSTRY

There are three distinct phosphate-rock producing areas in the United States—Florida, Tennessee, and the Western States. The producing companies in 1953 in all three fields are listed in table 4.

PROCESSING (*3*)

Methods.—There are three fundamental methods of decomposing phosphate rock to obtain products suitable for fertilizer and industrial purposes:

1. Acid treatment.
2. Thermal reduction.
3. Thermal treatment without reduction.

1. In the *acid treatment*, phosphate rock is mixed with sulfuric, phosphoric, or nitric acid or a combination of the three.

Decomposing phosphate rock with sulfuric acid produces either ordinary superphosphate or phosphoric acid. When ordinary superphosphate is wanted, enough sulfuric acid is added to high-grade phosphate rock to convert the phosphate into water-soluble or available monocalcium phosphate. Calcium sulfate is also produced but is not separated from the mixture, which seldom contains over 20 percent available P_2O_5. To produce phosphoric acid, additional sulfuric acid is added, and the phosphoric acid is filtered from the insoluble calcium sulfate.

TABLE 4.—*Phosphate-rock producers in the United States, 1953, and type of operations*

Field and company	Mine			Beneficiation plant		Drying or sintering plant	Fertilizer or chemical plant
	Open pit	Underground	Location	Washing plant	Other		
Florida land pebble:							
American Agricultural Chemical Co., 50 Church St., New York, N. Y.	X	------	Pierce, Fla	X	X	X	X
American Cyanamid Co., 30 Rockefeller Plaza, New York, N. Y.	X	------	Brewster, Fla	X	X	X	X
Armour Fertilizer Works, Atlanta, Ga	X	------	Bartow, Fla	Under development			------
Coronet Division, Smith-Douglass Co., Norfolk, Va.	X	------	Plant City, Fla	X	X	X	------
Davison Chemical Corp., Baltimore, Md.	X	------	Bartow, Fla	X	X	X	X
International Minerals & Chemical Corp., 20 N. Wacker Drive, Chicago, Ill.	X	------	do	X	X	X	X
Swift & Co., Chicago, Ill	X	------	Fort Meade, Fla	X	X	X	X
Virginia-Carolina Chem. Corp., Richmond, Va.	X	------	Nichols, Fla	X	X	X	X
Florida hard rock: Kibler-Camp Phosphate Enterprise, Ocala, Fla.	X	------	Dunnellon, Fla	X	------	X	------
Florida soft rock:							
Kellogg Co., Hernando, Fla	X	------	Hernando, Fla	------	------	X	------
Knight & Bevis, Dunnellon, Fla	X	------	do	------	------	X	------
Loncala Phosphate Co., High Springs, Fla.	X	------	Fort White & Newberry, Fla.	------	------	X	------
Pedrick & Bernard, Morriston, Fla	X	------	Morriston, Fla	------	------	------	------
Sea Board Phosphate Co., Dunnellon, Fla.	X	------	Dunnellon, Fla	------	------	X	------
Soil Builders, Inc., Dunnellon, Fla	X	------	Hernando, Fla	------	------	X	------
Superior Phosphate Co., Dunnellon, Fla.	X	------	Dunnellon, Fla	------	------	------	------
Tennessee brown rock:							
Armour Fertilizer Works, Atlanta, Ga	X	------	Columbia, Tenn	X	------	X	X
Harsh Phosphate Co., Nashville, Tenn	X	------	Nashville, Tenn	------	------	X	------
Highland Mining Co., Centerville, Tenn.	X	------	Centerville, Tenn	X	------	X	------
International Minerals & Chemical Corp., 20 N. Wacker Drive, Chicago, Ill.	X	------	Wales and Mount Pleasant, Tenn.	X	X	X	X
Mine Equipment Co., Columbia, Tenn	X	------	Columbia, Tenn	------	------	------	------
Monsanto Chemical Co., St. Louis, Mo	X	------	do	X	------	X	X
Owens Agricultural Phosphate Co., Centerville, Tenn.	X	------	Centerville, Tenn	------	------	X	------
Tennessee Valley Authority, Wilson Dam, Ala.	X	------	Columbia, Tenn	------	------	X	X
Victor Chemical Works, 141 W. Jackson Blvd., Chicago, Ill.	X	------	Mount Pleasant, Tenn	X	X	X	X
Virginia-Carolina Chem. Corp., Richmond, Va.	X	------	do	X	X	X	X
Western rock:							
Anaconda Copper Mining Co., Anaconda, Mont.	X	X	Conda, Idaho	X	------	X	X
Garfield Chemical & Mfg. Corp., Salt Lake City, Utah.	X	------	Thistle, Utah	------	------	------	------
Monsanto Chemical Co., St. Louis, Mo	X	------	Soda Springs, Idaho	------	------	X	X
Montana Phosphate Products Co., Garrison, Mont.	------	X	Garrison, Mont	------	------	------	------
George Relyea, Garrison, Mont	------	X	do	------	------	------	------
San Francisco Chemical Co., Montpelier, Idaho	X	X	Montpelier, Idaho	------	X	------	------
	X	------	Leefe, Wyo				
	------	X	Woodruff, Utah				
J. R. Simplot Co., Pocatello, Idaho	X	------	Fort Hall, Idaho	------	------	------	------
	X	------	Randolph, Utah	------	------	------	X
Victor Chemical Works, Chicago Ill	------	X	Melrose, Mont	------	------	X	X

Acidulating phosphate rock with phosphoric acid produces triple superphosphate. Also, phosphoric acid may be combined with potash and/or ammonia to produce highly concentrated fertilizer salts, such as ammonium phosphates and potassium phosphates.

Using nitric acid or a combination of nitric and another acid for acidulation of phosphate rock has not been used extensively in the United States. The resulting product, because of the hygroscopic calcium nitrate content, must be further treated, usually by ammoniation, to produce a suitable fertilizer product. Further use of nitric acid processes will depend on the availability and cost of nitric and sulfuric acids.

2. The *thermal-reduction method* is based on the smelting of phosphate rock with carbon (coke) and a siliceous flux in electric or blast furnaces. The products are elemental phosphorus, ferrophosphorus, carbon monoxide, and calcium silicate. The phosphorus is volatilized, condensed, and collected under water as a heavy liquid. The calcium silicate is tapped from the furnace as a molten slag. Small quantities of ferrophosphorus, produced from the iron in the phosphate rock, are tapped from the furnace and marketed as a byproduct.

Phosphoric acid is produced by burning phosphorus and hydrating the P_2O_5 produced. The phosphoric acid may be converted into various fertilizer or industrial phosphorus compounds. Calcium metaphosphate is produced by burning elemental phosphorus in a chamber into which pulverized phosphate rock is blown. The phosphate rock and P_2O_5 react to yield molten calcium metaphosphate, which is tapped and cooled to form a glassy solid.

3. There are three variations of *thermal treatment without reduction*. These are: (*a*) Heating phosphate rock with an alkali salt (for example, Na_2CO_3) with or without substantial defluorination; (*b*) heating phosphate rock in the presence of water vapor to defluorinate the rock as completely as possible; and (*c*) fusion of phosphate rock mixed with magnesium silicate (serpentine or olivine). Following grinding these products do not require further treatment.

Choice of Processing Method.—The higher cost of electric power compared to sulfuric acid has retarded the use of elemental phosphorus or phosphoric acid derived from elemental phosphorus in the fertilizer industry. The food, chemical, and drug industries require acid of high purity, such as that made by the electric-furnace process, and utilize most of the electric-furnace product.

Ordinary superphosphate is the simplest phosphate product to manufacture and the most widely used. It is largely water soluble, and the phosphorus is readily available to crops when applied to acid, alkaline, or neutral soils. Ordinary superphosphate usually is manufactured close to the market and sometimes is hauled directly from factory to farm in bulk to reduce handling costs.

Processes for manufacturing triple superphosphate and other derivatives of phosphoric acid, such as ammonium phosphate, are more complex than the ordinary superphosphate process and require a larger investment. The products carry a far greater proportion of plant food, which may offset the higher production costs when long freight hauls are involved.

Elemental phosphorus is more concentrated than any of the phosphorus derivatives, resulting in lower freight cost per unit of P_2O_5. The thermal-reduction method requires a larger investment than the wet process but has the advantage of yielding highly concentrated products of high purity suitable for use in the food, chemical, and drug industries, as well as the fertilizer industry.

Calcium metaphosphate has a very high P_2O_5 content (about 61 percent) and desirable physical properties, but it is not water soluble and is of questionable suitability on alkaline soils.

Defluorinated phosphates are effective on acid soils but are less satisfactory when applied to alkaline soils. Much of the defluorinated phosphate rock is used as animal-feed supplement.

USES

Phosphate rock has diverse agricultural and industrial uses. The quantity used domestically for agricultural purposes (60 percent of the total sold or used in 1953) has always exceeded that used for all other purposes. By far the greater part of the phosphate rock for agricultural uses goes into the preparation of ordinary and triple superphosphates, the former containing 16 to 20 percent available P_2O_5 and the latter 42 to 50 percent available P_2O_5.

Elemental phosphorus and phosphoric acid are used to make inorganic and organic chemicals which are used as leavening agents, water softening and cleansing products, plasticizers and insecticides, and for many other purposes. Elemental phosphorus is used for military smoke screens and incendiary bombs, pesticides, and phosphorus alloys. Phosphoric acid also is used in the production of fertilizers, animal-feed supplement, beverages, ceramics, as a catalyst and oil-refining agent, in photography, and in dental and silicate cements.

CONSUMPTION

By 1953 the annual world consumption of phosphate rock exceeded 23 million metric tons. The 2 major markets—Europe and the United

States—absorb over 80 percent of the world consumption.

Europe draws its supplies principally from North Africa, with Russia and the United States furnishing considerable quantities. The United States is self-sufficient.

FOREIGN TRADE[3]

Exports.—Before World War I a substantial part of domestic production was exported. Since that time exports, because of foreign competition, especially from North Africa, have consumed only a minor portion of domestic production. The 1940–53 exports are shown in table 5.

TABLE 5.—*Exports of domestic phosphate rock, 1940–53*

(U. S. Department of Commerce)

Year:	*Long tons*
1940	751,495
1941	1,019,960
1942	528,720
1943	358,217
1944	439,265
1945	490,733
1946	633,340
1947	1,642,987
1948	1,141,364
1949	1,257,962
1950	1,832,048
1951	1,726,834
1952	1,429,284
1953	2,100,798

Imports.—Imports of phosphate rock into the United States have been relatively unimportant. Several thousand tons annually of low-fluorine phosphate rock are imported from Curaçao. However, threats of large-scale invasion of the domestic market by foreign phosphates have been made. In 1927 the first imports of high-grade phosphate rock came from French Morocco (10,641 long tons) and caused considerable concern among domestic producers, as did later imports of apatite from the deposits in Russia, but these attempts to enter the domestic market were unsuccessful.

SUBSTITUTES

Basic slag is a limited source of agricultural phosphorus in the United States. Domestic production comes from a single company smelting a phosphatic iron ore of the Birmingham, Ala., district. No figures on production are available. Consumption has ranged from 200,000 to 400,000 tons per year during the last 5 years.

SELF-SUFFICIENCY

The United States is self-sufficient in phosphate rock. It has large reserves and depends on foreign sources only for low-fluorine phosphate rock, now obtained from Curaçao. Defluorination of portions of the domestic production will enable it to meet the demand for low-fluorine phosphate rock for animal feed. A little phosphate rock has been brought into the Hawaiian Islands from Makatea in French Oceania.

STRATEGIC CONSIDERATIONS

Phosphate rock itself is not on the Government's list of strategic and critical materials; but elemental phosphorus is an important war material, phosphate chemicals are essential to many vital industries, and phosphatic fertilizers are essential to maintain and expand the agricultural output of the United States. Electric furnaces for the production of elemental phosphorus in the United States are meeting all demands for elemental phosphorus for military and industrial use.

PRICES

Prices for phosphate rock are based on the percentage of tricalcium phosphate, $Ca_3(PO_4)_2$ (B. P. L.) present. Maximum allowable iron and aluminum oxides are specified. Bonuses are paid and penalties assessed for variations above and below the base grade. Prices usually are quoted in dollars per long ton for all marketable phosphate rock.

The average value at the mines in 1953 by States and types of rock as calculated from reports to the Bureau of Mines is as follows:

State and type:	*Average value at mines per long ton*
Florida:	
Land pebble	$6.05
Soft rock	6.19
Hard rock	7.88
Total Florida	$6.07
Tennessee	7.55
Idaho	3.82
Montana, Utah, and Wyoming	7.06
Total United States	$6.12
Imports	25.16
Exports	6.43

The Idaho value is a composite of the value of high-grade rock and electric-furnace-grade phosphatic shale.

Quotations of Florida phosphate rock from the Oil, Paint and Drug Reporter of December 28, 1953, are given in table 6. The trade journals do not publish quotations for Tennessee or Western States phosphate rock.

TABLE 6.—*Prices of Florida phosphate rock*

[Dollars per long ton, unground, washed, and dried, in bulk, f. o. b. cars at mine, by grades]

	Prices
Florida land pebble:	
70/68 B. P. L	$4.69
72/70 B. P. L	5.69
75/74 B. P. L	6.69
78/76 B. P. L	7.69

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[3] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

CONSERVATION PRACTICES

In 1908, soon after phosphate-mining operations were begun in the Western States, about 7,000 square miles of phosphate-bearing public lands in that field was withdrawn from entry pending their examination and classification. In 1910 phosphate-bearing lands in Florida first were withdrawn for investigation and classification.

The Bureau of Land Management, United States Department of the Interior, administers leasing of phosphate public lands under Public Law 146—Act to Promote the Mining of Coal, Phosphate, Oil Shale, Gas, and Sodium on the Public Domain (Mineral Leasing Act, Feb. 25, 1920, 41 Stat., Ch. 85, sec. 9, pp. 440–441).

The Geological Survey is responsible for mapping the phosphate lands; for certification to the Bureau of Land Management of parcels of land known to contain economic phosphate deposits; and for supervising mining practices to prevent waste and to conserve currently uneconomic resources for future recovery.

The Bureau of Land Management, on application from a qualified citizen or corporate group, opens specified parcels of land to competitive bids for leases if the Geological Survey has certified them to be phosphate-bearing, or may accept a noncompetitive bid for a lease on parcels on which no phosphate is certified.

There are no standard provisions regarding phosphate leases. Rents, royalties, and minimum expenditure requirements are suggested by the Geological Survey, but royalties shall not be less than 2 percent of the gross value of output of phosphate rock from the mine. The minimum expenditure requirements are usually nominal to insure the good faith of the lessee, before production is begun.

In 1951 no phosphate rock was produced from public land in Florida. Less than 10 percent of the phosphate rock produced in the Western States in 1951 was from public land, excluding Indian lands.

In the Western States only a few good phosphate areas on private land remain, hence most new companies planning production from that area will seek leases on public land; however, most of the expansion of the next 10 years by existing operators will be on private land.

RESEARCH

The United States Department of Agriculture has been conducting fertilizer-technology research for over 40 years. The Section of Fertilizer and Agricultural Lime and its predecessor units have been responsible for the department's fertilizer-technology research since 1911, when this work was first placed on a continuing basis in the former Bureau of Soils. The section maintains close contact with Federal, State, and private agencies interested in fertilizer resources, technology, production, and use, as well as with industry associations, companies, and individuals. These contacts include studies and cooperative investigations. Since 1929 the Department of Agriculture's investigations have been directed chiefly toward (1) conversion of ammonia, phosphoric acid, and potash into high-analysis compounds for fertilizers; (2) improvement in processing phosphate and potash raw materials; (3) developing suitable high-analysis mixed fertilizers; (4) evaluation of fertilizer materials and mixtures as plant food; and (5) a continuing study of production, consumption of plant food and trends in fertilizer use (*2*, *5*).

In 1933 the Tennessee Valley Authority was established by act of Congress. In the interest of national soil conservation, and especially to improve the depleted lands of the Southeast, TVA immediately undertook a fertilizer research and production program. In planning the TVA program in 1933 there appeared to be ample opportunity to improve general fertilizer-manufacturing techniques, to lower manufacturing and processing costs, and to improve the grade of fertilizer generally sold, all with the purpose of lowering the cost per unit of plant food delivered to the consumer.

During late 1933 a chemical-research laboratory was established at Muscle Shoals, Ala., and shortly thereafter a chemical-engineering-research laboratory was developed. Most small-scale experimental work previously conducted at Knoxville, Tenn., was transferred to these laboratories during 1934, leaving at Knoxville the cooperative work with the Agricultural Experiment Station and eventually a small group working in cooperation with the Chemical Engineering Department of the University of Tennessee. The Chemical Engineering Division of TVA organized functioning units for research, development, design, production, mining, and other phases of the industry. The first large installation was an electric smelting furnace for producing elemental phosphorus in 1934. The present chemical plant includes six electric furnaces, phosphoric acid plant, superphosphate plant, triple superphosphate plant, calcium metaphosphate plant, synthetic ammonia plant, nitric acid plant, ammonium nitrate plant, and fused tricalcium phosphate plant. In addition to these installations, TVA also operates mines near Columbia, Tenn.

In 1951 the National Fertilizer Association and TVA formed cooperative technical committees to exchange data and coordinate TVA and industry research. Periodic meetings are held to discuss accomplishments and to plan future work.

Research is being conducted by TVA on fertilizer-manufacturing processes involving the use of nitric acid as a partial or complete replacement for sulfuric acid.

The Atomic Energy Commission, in cooperation with industry, has been conducting research on methods of recovering the uranium in phosphate rock, and uranium is being produced as a byproduct by several companies.

Utilization of the phosphate content and recovery of the uranium from the "leached zone" immediately overlying parts of the land-pebble ore in Florida have been investigated since 1953.

The Bureau of Mines has done preliminary work on mining-method studies and is attempting to develop a continuous mining machine adaptable to various types of underground deposits. Beneficiation studies also are being conducted by private companies and the Bureau of Mines to devise means of utilizing low-grade Western field phosphate rock. Methods for recovering vanadium and fluorine from phosphate rock have been developed.

The Minerals and Metals Advisory Board of the National Academy of Sciences studied the phosphate-slime problem in the Florida land-pebble field and on March 16, 1953, held a meeting of its Phosphate Panel at Lakeland, Fla. The recommendations resulting from this meeting were as follows:

1. Exploratory investigations on the general pattern of those previously conducted on leached zone material, but covering only a limited number of typical samples of phosphatic matrix. These investigations should include:
    (*a*) Sizing assay tests.
    (*b*) Mineral distribution grain counts of suitable size fractions.
2. Similar investigations of the chemical and mineral compositions of suitable size fractions derived from several typical samples of slime effluent from commercial washers.

OUTLOOK

For many years it has been necessary to fertilize increasingly larger areas of cropland and to increase applications on previously fertilized areas to maintain and increase food and fiber production. Fertilizer is needed to counteract the present downward trend of soil fertility. The farm-production trend, however, is upward, and this rising trend in crop production is expected to continue. Future agricultural requirements, because of the expanding national and world population and higher nutritional standards, will greatly exceed the present output. Phosphatic fertilizer consumption in the United States grew from about 6½ pounds of available P_2O_5 per capita in 1900 to 13.8 pounds in 1940 and then doubled to 27.4 pounds per capita in 1950. The available P_2O_5 applied per acre (2 pounds in 1900), was 6 pounds in 1940 and 12 pounds in 1950. This uptrend is expected to continue, although possibly not at the latest rate.

About 70 percent of the P_2O_5 consumed in phosphatic fertilizers is in the form of ordinary superphosphate, and the greater part of the balance is in triple superphosphate. Triple superphosphate has been manufactured continuously in the United States since 1907 with phosphoric acid made by the sulfuric acid process. Since 1947, when ordinary superphosphate production started to level, triple superphosphate has maintained its upward trend.

Interest in nitraphosphate manufacture in the United States accelerated during 1951–53, when a sulfur shortage threatened to reduce materially sulfuric acid supplies. There may be increased production of nitraphosphate in the future.

The trend in the production of phosphate rock in the Western States has been sharply upward and is expected to continue. New electric furnaces and new phosphatic fertilizer plants have been built.

Production of phosphate rock in North Africa and Europe will continue to supply the major part of European requirements, and the island deposits of the Pacific and Indian Oceans will continue to supply the eastern markets.

Early adverse reports on the value of direct application of raw phosphate rock as a fertilizer hindered the use of this material. In recent years the use of very finely ground phosphate rock for direct application to soils, mainly in the Midwest, has increased rapidly. The consumption of 106,292 long tons in 1940 was only 3 percent of total domestic sales, but by 1953 it had increased to 1,026,326 tons—8 percent of the total domestic sales.

In recent years progressively larger tonnages of domestic phosphate rock sales have been consumed in the manufacture of elemental phosphorus, which is used to manufacture chemicals for military and industrial purposes. This trend gives every indication of continuing.

Reserves of phosphate rock are adequate to supply many times any expected demand. It is believed also that mine productive capacity is adequate, and can be expanded to meet the expected increased demand, which may reach 4.8 million tons of P_2O_5 in 1960 and 8 million tons of P_2O_5 by 1975. Exploration and development, with improved beneficiation techniques, may be expected to maintain commercial reserves ahead of mining. In the Western States it is anticipated that underground mining will become increasingly important.

PROBLEMS

The important problems confronting the phosphate-rock industry are conservation, development of new and improved mining methods and beneficiating techniques, more complete recovery and utilization of byproducts, and improvement of commercial methods of producing more concentrated phosphates.

The general aspects of the Phosphoria formation in the Western States are known, but details of the economic phosphate-rock beds are incomplete. It has been difficult to evaluate drilling records because of poor core recovery and irregular spacing of diamond-drill holes. Studies are needed to correct these deficiencies.

Complex geological aspects and varying grade of the phosphate beds in the Western States hinder efficient mining operations. Many present and potential producers have stated the need for studies of mining methods to permit optimum recovery from the various types of surface and underground operations yet leave the low-grade beds in a condition to permit possible future entry.

The disposal and possible commercial utilization of the slimes from the washing of phosphate rock in Florida and Tennessee are problems that will become increasingly important in the future. No commercial methods have yet been developed to utilize the slimes, to recover the P_2O_5 from them, or to dispose of them compactly.

The lack of adequate beneficiation techniques for some types of rock is one of the factors that retards the growth of the western phosphate industry. Whereas some phosphatic shales (averaging 24 percent P_2O_5) are being used in producing elemental phosphorus in electric furnaces, western fertilizer operations are confined to the use of higher grade beds, with little or no regard to the potential value of the low-grade phosphate strata in the Phosphoria formation. Improvements in beneficiating techniques or further expansion of the thermal-reduction processes may allow more complete utilization of the beds, thus conserving reserves. Economical beneficiation methods would make it possible to mine a greater proportion of the Phosphoria formation.

The recovery of byproducts is becoming increasingly important. At present, ferrophosphorus, uranium, vanadium, fluorine, calcium silicate slag, and other byproducts are being recovered. Processes for extending the commercially feasible recovery of these and other byproducts have not been fully developed.

The Tennessee phosphate industry rapidly is depleting the economic reserves in that area. Processes that would permit mining lower grade phosphate beds and the phosphatic limestones would extend the life of the phosphate-rock-mining industry in Tennessee.

The fertilizer and phosphate industries are highly competitive; and, in attempting to make more attractive products, many processors include other elements in their mixed products. In addition to the major plant-food elements (nitrogen, phosphorus, and potassium), 15 or more other elements are necessary for plant life. These secondary and minor plant-food elements include calcium, magnesium, sulfur, iron, copper, manganese, zinc, boron, and others. The use of these secondary and minor plant-food elements will become increasingly important as agriculture depends more and more on fertilizers and the need for these elements is better recognized. Sources of the minerals and compounds that release the desired elements at appropriate rates in various types of soils should be sought.

With large parts of western phosphate rock on Federal land, the Government has heavy responsibilities in developing these deposits. Any change in the present regulations must be considered in the interest of conservation and utilization of this valuable national asset.

Technical problems include development of methods for production of phosphate fertilizers without the use of sulfuric acid, with less of it, or with nitric acid as a partial or complete substitute.

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PLATINUM-GROUP METALS

By

James E. Bell[1]

ALTHOUGH their advent into industry is comparatively recent, supplies are limited, and prices are high, the platinum metals have much industrial importance because of their unique physical and chemical properties. They are also highly esteemed for jewelry and decorative uses because of their rarity and unusual properties, including tarnish resistance and fine appearance. The United States consumes over half the world output of platinum metals but produces only a small part of its requirements.

Summary

The metals of the platinum group include platinum, palladium, iridium, osmium, rhodium, and ruthenium. Platinum is the most abundant and also the most important member of the group, but the others are growing in importance. All platinum metals except palladium command prices much higher than gold.

Platinum metals are sometimes found in native form in some combination with the member metals of the group. They also occur combined with other metals or metalloids.

Russia began mining native platinum around 1820 and was the chief producer for nearly a century. Colombia led for a few years following 1916, and Alaska became a producer of some importance in the mid-1930's. Recovery of platinum metals as byproducts from large nickel-copper mining operations has given Canada the lead since 1934. Development of lode deposits containing platinum metals has placed the Union of South Africa second only to Canada in output of these metals in recent years. Deposits of platinum metals are known in other areas in the world, but production has been minor. The domestic output forms only a few percent of the world total.

Based on their high melting points, resistance to chemical action, and workability, the platinum metals have specific and important uses in industry and in the arts and sciences. Platinum and iridium have important military uses and are among the strategic and critical metals being stockpiled. The domestic consumption of the platinum metals far exceeds domestic production, and continuance of imports from abroad is essential.

The world outlook for platinum-group metals is for greater production and consumption. The domestic outlook is for lower production and greater consumption, with perhaps development of substitutes in some uses.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

GENERAL

The platinum-group metals include platinum, palladium, iridium, osmium, rhodium, and ruthenium. Platinum is the most plentiful, best known, and most widely used member, but the other metals of the group are increasing in importance. The platinum metals are found in nature in the uncombined form, associated with each other and frequently with gold, copper, nickel, iron, or chromite; some metals of the group are also found in combination with sulfur, arsenic, or antimony.

Native platinum appears to have been discovered by the aborigines of Colombia, who used it for ornaments along with gold. Colombian platinum found its way to Europe early in the 18th century, though little use was made of it except for counterfeiting Spanish doubloons—striking spurious coins in crude platinum, and coating them with gold. Later in the 18th century work by scientists led to recognition of the valuable properties of platinum for laboratory utensils and to some commercial interest in platinum. Efforts to refine crude platinum led to identification and separation of the member metals of the platinum group early in the 19th century.

Native platinum was first noticed in the washings of gold placers in the Ural Mountains in Russia in 1819; and a few years later rich deposits were discovered in this area, which became the principal source of world platinum supply for nearly a century. During the early part of this period uses of platinum were so limited that the Russian Government supported the demand by authorizing platinum coinage. With the general rise of industrialism in the latter half of the 19th century, however, many uses for platinum metals and their alloys were found, and a strong platinum industry was firmly established.

The demand for platinum led to development of and growing production from Colombian placer fields early in the 20th century, and for a few years after 1916 Colombia was the chief producer. Demand had also led to discovery of placer platinum deposits or byproduct platinum metals in various parts of the world, including Australia, Belgian Congo, Borneo, Canada, Ethiopia, Japan, Papua, Sierra Leone, South Africa, Tasmania, and the United States (California and Oregon), with a significant output from some areas. The Goodnews Bay placer platinum field in southwestern Alaska became known in the 1920's and was developed into an important producer in the mid-1930's.

Production of nickel-copper ores in the Sudbury district of Canada commenced on a substantial scale in the early 1900's, but byproduct platinum-group metals known to be contained in the ores were not recovered as an important part of the operations until the mid-1930's; however, with expansion of facilities for producing nickel and copper and improvements in the processes for recovering byproducts, Canada became the leading world producer of platinum metals in 1934. The presence of osmiridium in small quantities in the ores of the Rand gold district, South Africa, was noted in 1892; and when preliminary concentration of gold values on corduroy was introduced in the Rand in the early 1920's in place of amalgamation, the recovery of osmiridium became an established practice, with regular annual production of a few thousand ounces. Platinum-group metals were discovered in economic quantities in certain horizons of the enormous Bushveld igneous complex in South Africa in 1925. After a slow start, because of technical and financial difficulties, the output from these deposits has increased steadily, with the result that South Africa is expected to lead substantially in the production of platinum-group metals in the near future. Nickel and copper are byproducts of South African platinum ores.

The general organization of the platinum-metals industry, as it relates to the United States, is as follows (based on data for 1952):

Canada leads in the current output of platinum metals, producing about 40 percent of the world total. Of the Canadian production, around 45 percent is platinum and the remainder mostly palladium. The principal producer is the International Nickel Co. of Canada, Ltd., which ships its crude platinum-bearing products to a subsidiary plant in England for reduction to refined metals. The greater portion of the output is returned to Canada, whence most is exported to the United States for fabrication and/or distribution by Baker & Co., Inc., Newark, N. J., and affiliated companies. A minor part of the Canadian output of platinum metals is produced by Falconbridge Nickel Mines, Ltd., and refined in Norway.

Second in production is the Union of South Africa, with 35 percent of the world total; the output averages about 60 percent platinum; the balance is largely palladium. The principal producing company is Rustenburg Platinum Mines, Ltd., which sends its crude products to Johnson, Matthey & Co., Ltd., in England for refining. The part of this metal imported into the United States is distributed by the Johnson, Matthey interests, and much is fabricated and distributed further by the affiliated J. Bishop & Co. Platinum Works, Malvern, Pa. The osmiridium byproduct of gold mining is shipped to refineries in various countries for refining; there are several in the United States.

Although no actual data are available, it is believed that the Soviet Union ranks third in the current output of platinum-group metals, producing about 15 percent of the world total. Large quantities of refined platinum metals were formerly imported into the United States from the Soviet Union, but during the years 1947–52 virtually palladium only was imported. However, in 1953 substantial quantities of platinum of Russian origin were sold on world markets, and a large part reached the United States.

Fourth in output of platinum metals is the United States (and Alaska), with 5 percent of the world total. This production includes crude platinum in small quantities recovered as a byproduct of gold placer mining, principally in California; byproduct platinum metals, largely palladium, present in very small quantities in some gold ores and copper ores, partly imported, and recovered in the final refining of gold bullion; and crude platinum dredged in the Goodnews Bay district in Alaska, which forms most of the domestic production. The operator in Alaska is the Goodnews Bay Mining Co.; most of the output is purchased by Johnson, Matthey & Co., Inc., refined by the J. Bishop Co. Platinum Works, and sold and distributed by Johnson, Matthey & Co., Inc.

The production of crude platinum in Colombia has trended downward in recent years and in 1952 formed only 5 percent of the world total. The bulk of the production results from dredging operations of the South American Gold & Platinum Co., which sells most of its output to refiners and dealers in the United States.

In addition to the Baker & Co. and J. Bishop & Co. plants, there are in the United States some several dozen platinum refiners, fabricators, and dealers, situated mostly in the Eastern States. Many platinum works handle scrap and do refining on toll; the larger fabricators do a considerable business in reworking plant equipment on toll.

TECHNOLOGY

Some properties of the platinum-group metals are listed in the table below (*7*):[2]

Metal	Specific gravity	Melting point, °C.	Tensile strength (annealed metal), p. s. i.	Brinell hardness (annealed metal)
Platinum	21.45	1,773	20,700	37
Palladium	11.76	1,554	30,000	37
Iridium	22.41	2,454		
Rhodium	12.40	1,966		
Osmium	22.48	[1] 2,700		
Ruthenium	12.20	[1] 2,450		

[1] Approximate.

In general, the addition of iridium, rhodium, osmium, or ruthenium to platinum or palladium increases their tensile strength and hardness. For example, annealed platinum with 5 percent iridium has a tensile strength of 40,000 pounds per square inch and a Brinell hardness of 90; annealed palladium with 5 percent ruthenium has a tensile strength 54,000 pounds per square inch and a Brinell hardness of 90. In both instances the tensile strength and hardness of unannealed materials are considerably higher. In general, also, the addition of iridium, rhodium, osmium, or ruthenium to platinum or palladium increases their resistance to chemical corrosion and tarnish.

The habitat of the platinum-group metals is in the ultrabasic rocks, where they have been concentrated by magmatic processes. There are two principal types of occurrence: Local concentrations of native metals in dunite, frequently associated with chromite; and disseminated in nickel-copper sulfide deposits in norite (*4*). Erosion of the former has yielded placer deposits like those in Alaska, Colombia, and the Soviet Union. The nickel-copper ores in the Sudbury district in Canada and certain horizons in the Bushveld igneous complex in the Union of South Africa are examples of the latter.

Crude or native platinum, whether in placers or lodes, never occurs as a pure metal but always as a natural alloy made up of varying proportions of members of the platinum-group metals; frequently it contains gold and iron as alloys or in mechanical mixture. Native platinum averages about 80 percent platinum, and native osmiridium about 35 percent osmium and 30 percent iridium, with other members of the platinum group accounting for most of the remainder. The principal minerals of the platinum-group metals found in the copper-nickel sulfide type of deposit are sperrylite ($PtAs_2$), cooperite (PtS), braggite ((Pt, Pd, Ni) S), laurite (RuS_2), and stibiopalladinite (Pd_3Sb). The average of the entire current production of platinum-group metals, both placer and lode, is around 60 percent platinum, 30 percent palladium, and 10 percent iridium, osmium, rhodium, and ruthenium together.

Crude placer platinum is recovered by methods and facilities that are essentially similar to those for recovering placer gold, except that the platinum-group metals do not respond to amalgamation, a method common in gold recovery (*2*). Most is recovered in large-scale operations by dredging, but small-scale hand methods are practiced in Colombia and probably in the Soviet Union, as well as in countries with minor output, like Ethiopia.

In Canada, at the operation of the International Nickel Co. of Canada, Ltd., sulfide ore averaging about 3 percent combined nickel and copper is mined on a large scale by underground methods, concentrated by flotation, and smelted; platinum-rich residues from the refining of nickel and copper are shipped to a subsidiary company in England for recovery of the platinum-group metals contained. Figures available indicate that the ore in place averages

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

about 0.025 ounce per ton in recoverable platinum metals.

Crude or native platinum, and also platinum scrap and byproduct platinum, are separated and refined into the metals of the platinum group they contain by a complicated chemical treatment based on solution in aqua regia and precipitation as double ammonium chlorides; in many instances the detailed procedures are closely guarded trade secrets among the various refineries (*3*). The precipitates are calcined, leaving a spongy mass of pure metal, which is compacted by melting in a high-temperature oxyacetylene or electric furnace. The platinum-group metals in platinum-rich matte are recovered by a combination of complex smelting, chemical, and electrorefining processes. Byproduct platinum metals from copper and gold ores are recovered by electrolysis in the final refining of gold bullion. The bulk of the output of refined metals has a purity of 99.95 to 99.99 percent; even higher purity is required for some purposes.

BYPRODUCTS, COPRODUCTS, AND SUBSTITUTES

Some byproduct gold is always recovered in placer platinum-mining operations, and in certain platinum placers gold is abundant enough to be a coproduct. Considerable nickel and copper and a small quantity of gold are byproducts of lode platinum mining in South Africa.

Gold-platinum alloy is substituted to some extent for platinum for spinnerets in rayon manufacture. White gold in a sense is a substitute for platinum in jewelry. Palladium is substituted for the more costly and (at times) scarcer platinum for jewelry and for various chemical applications. Rhodium and ruthenium are satisfactory substitutes for iridium for some uses.

SECONDARY SOURCES

Substantial quantities of platinum-group metals in the form of worn laboratory ware, outmoded jewelry, scrap, sweeps, etc., are sold to platinum plants for refining and resale to the market. Substantial quantities are also turned in for refining on toll and return of the metals to the owners. In addition, a large volume of platinum industrial equipment of various kinds is refined or reworked, mostly by the larger platinum plants on toll.

RESERVES

Figures available indicate that Canada has reserves of about 6.5 million ounces of platinum-group metals in copper-nickel ores, which will be produced during the next 20 years and that about 45 percent of this output will be platinum and the balance largely palladium.

Work to date indicates that the reserves of platinum metals in Africa may be very large. In the Union of South Africa the Bushveld igneous complex occupies an area of some 12,000 square miles, and platinum has been found at many widespread points, at which only the best-grade showings have been opened. Promising showings have also been found and prospected in the Great Dyke ultrabasic formation in Southern Rhodesia. The reserves of osmiridium in the gold ores of the Union of South Africa are considerable, and platinum placer deposits in Ethiopia are believed to have moderate potentialities. Recoverable reserves of 10 million ounces in Africa appear to be a reasonable expectancy. Projecting the ratios of the current production, the reserves would average around 60 percent platinum, 30 percent palladium, and the balance iridium, osmium, rhodium, and ruthenium together.

Despite a production of some 11 million ounces extending over more than a century, the reserves of platinum metals in the Soviet Union are estimated to amount to some 4 million ounces. Most of the reserves are in placers in the Ural Mountains.

The reserves of platinum in Colombian placers are believed to be substantial, but parts of the area in which they occur are difficult of access and of low productive potential. At times the political and economic situations are unfavorable to mining operations, also.

The known reserves in Alaska are comparatively small, and the placers in which they occur are scheduled to be worked out in a decade or less. The sources of gold ore and copper ore that yield byproduct platinum-group metals, largely palladium, in moderate quantities in the United States should continue to be active for several decades.

To summarize, a recoverable world reserve of some 25 million ounces of platinum-group metals appears to be within reason. This would be composed of around 15 million ounces of platinum, 7.5 million ounces of palladium, and 2.5 million ounces of iridium, osmium, rhodium, and ruthenium together.

USES

As pure metals, combined, clad, or alloyed with other metals, the platinum-group metals are used for jewelry, in the chemical and electrical industries, in dentistry, and for many miscellaneous purposes; however, certain peacetime markets for the platinum metals are restricted by their high cost. In ornamentation their uses are based on fine appearance, workability, and chemical inertness. Platinum is the most widely used member of the group, and palladium is next in quantity used, the other four members are employed mostly in alloying platinum or palladium.

Platinum hardened with iridium or ruthenium is widely accepted as the ideal metal in the jewelry and decorative arts, particularly for gem-set jewelry. Palladium alloyed with ruthenium is gaining in acceptance for jewelry, particularly in Europe. Both platinum and palladium are beaten into leaf for designs and decorations.

Platinum-gold and platinum-rhodium alloys are widely used in spinnerets (nozzles usually with several hundred fine-diameter holes) for making rayon fiber from viscose. Fiber glass in rapidly increasing quantities is produced by forcing molten glass through banks of platinum nozzles, whence it emerges in fine streams that are stretched to filaments of minute diameter. Pure platinum and platinum-iridium alloys are used as insoluble anodes in various electroplating processes; chemical laboratories have long used platinum utensils and equipment.

Many chemical applications of the platinum-group metals arise from their catalytic activity. Catalytic uses include the production of nitric and sulfuric acids, hydrogenation and dehydrogenation, the synthesis of hydrocarbons, and hydroxylation. A recent and expanding development is the use of platinum as a catalyst for producing high-octane gasoline from low-grade and natural gasoline.

The platinum-group metals have numerous electrical applications. Palladium is used widely in the contacts of telephone relays. Platinum, both pure or hardened with iridium or ruthenium, is used for contacts in voltage regulators, thermostats, relays and contacts in high-tension magnetos. Sparkplugs equipped with platinum-alloy electrodes have long life and resist fouling. Platinum and platinum alloys are employed in numerous delicate electrical and laboratory instruments and in electronics tubes.

Platinum and palladium alloys are used extensively in dentistry for dentures, pins, and anchorages. Rhodium electroplate is used for jewelry, reflectors, and corrosion- and wear-resistant surfaces in industrial applications. Ruthenium and osmium are employed in many highgrade hard alloys for tips of fountain pens and phonograph needles. Platinum and palladium are used in special photographic printing papers. Platinum has great military importance for use in sparkplugs and high-duty electrical contacts for magnetos in mechanized equipment, in control instruments, and as catalysts in many chemical production processes.

It is to be noted that most uses of the platinum-group metals are nondissipative and that a large pool of these metals has accumulated in the United States.

STATISTICAL CONSIDERATIONS

In the United States figures on platinum-group metals with respect to domestic mine production, domestic recovery of refined metals from both new and secondary sources, sales to consuming industries by metals and uses, and stocks are gathered and compiled by the Bureau of Mines through canvassing of the industry (*1*). Data on imports and exports are supplied to the Bureau of Mines by the United States Department of Commerce. Reports with statistics covering all these phases are issued by the Bureau of Mines quarterly and annually.

Statistical information regarding the consumption of platinum-group metals is not issued by any other country than the United States; however, Canada publishes reliable figures on its annual production of platinum separately and the other five platinum metals together. Corresponding data published by the Union of South Africa give the total annual production of the six platinum metals together and the annual export of each metal. No actual data on the production of platinum-group metals in the Soviet Union have been available for many years. Figures on the annual production of crude platinum are released by Colombia and Ethiopia but appear incomplete. Minor quantities of crude platinum probably are produced currently in several other countries, but data on such outputs usually are not available.

PRODUCTION, CONSUMPTION, FOREIGN TRADE, AND STOCKS

The total production of the world of platinum-group metals to 1953 is around 22 million ounces of which about one-half was produced, by Russia and one-fourth by Canada. Table 1 gives figures on the production of platinum metals, by countries, for 1941–52.

TABLE 1.—*World production of platinum-group metals, 1941–45 (average) and 1946–52*

[Troy ounces]
(Compiled by Berenice B. Mitchell)

	1941–45 [1] (average)	1946	1947	1948	1949	1950	1951	1952
Australia	137	95	99	92	39	46	41	51
Belgian Congo: Palladium from refineries	--------	--------	--------	209	106	--------	--------	--------
Canada:								
Platinum: Placer and from refining nickel-copper matte	[2] 199, 000	121, 771	94, 570	121, 404	153, 784	121, 100	153, 483	122, 317
Other platinum-group metals: From refining nickel-copper matte	[2] 189, 522	117, 566	110, 332	148, 343	182, 233	148, 342	164, 905	157, 407
Colombia: Placer platinum	36, 815	43, 835	41, 415	40, 047	20, 797	26, 445	[3] 32, 000	[3] 33, 700
Ethiopia: Placer platinum	788	[4] 140	[4] 1, 548	[4] 460	[4] 355	(5)	266	100
Sierra Leone: Placer platinum	4	105	431	109	38	(5)	--------	--------
Union of South Africa:								
Platinum-group metals from platinum ores	75, 277	21, 877	23, 332	22, 549	30, 500	144, 217	190, 898	232, 521
Concentrates (platinum-group metal content) from platinum ores		50, 575	55, 508	46, 374	56, 800			
Osmiridium from gold ores	6, 661	6, 100	6, 071	5, 520	6, 031	6, 449	6, 359	6, 141
U. S. S. R.: Platinum from placers and from refining nickel-copper ores (estimate)	125, 000	175, 000	150, 000	125, 000	100, 000	100, 000	100, 000	100, 000
United States: Platinum from placers and from domestic gold and copper refining	34, 984	26, 312	18, 406	19, 253	24, 807	37, 855	36, 951	34, 409
Miscellaneous	283	14	--------	31	104	210	283	580
Total (estimate)	[2] 675, 000	575, 000	500, 000	525, 000	575, 000	575, 000	675, 000	675, 000

[1] This table incorporates a number of revisions of data published in previous platinum tables.
[2] Includes certain adjustments in 1954, to account for metals produced in Canada in 1938–44 but not included in the statistics for those years.
[3] Estimate.
[4] Exports for year ended Sept. 10 of year stated.
[5] Data not available; estimate included in total.

TABLE 2.—*Annual averages for platinum-group metals, 1941–52* [1]

	Fine troy ounces
Domestic production of crude—	
Platinum plus byproduct	
Platinum metals	31, 122
Domestic refining—	
From new metals	139, 881
From secondary metals	80, 411
Imports of refined metals	249, 312
Domestic consumption as measured by sales:	
Platinum	245, 990
Palladium	154, 543
Iridium, osmium, rhodium, ruthenium	25, 791
Total	426, 324
Exports (unmanufactured)	32, 636
World production	623, 750
Stocks of refineries and dealers in the United States (Dec. 31, 1952)	282, 373

[1] Source: Bureau of Mines.

A breakdown of the domestic consumption by uses for the same period is as follows:

Metal	*Use*	*Percen*
Platinum	Chemical	33. 2
	Electrical	31. 4
	Dental and medical	7. 6
	Jewelry	26. 7
	Miscellaneous	1. 1
	Total	100. 0
Palladium	Chemical	9. 0
	Electrical	45. 3
	Dental and medical	19. 3
	Jewelry	24. 1
	Miscellaneous	2. 3
	Total	100. 0
Iridium, osmium, rhodium, and ruthenium together.	Chemical	26. 4
	Electrical	23. 6
	Dental and medical	. 9
	Jewelry	32. 7
	Miscellaneous	16. 4
	Total	100. 0

SELF-SUFFICIENCY AND STRATEGIC CONSIDERATIONS

During 1941–52, the domestic production of primary platinum-group metals averaged 7 percent of the domestic consumption (*1*). Unless new deposits are discovered, in the early foreseeable future this will decline to about 2 percent, with the annual output comprised of only a few hundred ounces of byproduct crude platinum from gold placer mining and a few thousand ounces of byproduct platinum metals from gold ores and copper ores. However, it is probable that, in extreme national need, platinum metals in various forms owned privately could be commandeered by the Government; potential supplies that might be obtained by such action are substantial.

Because of its military significance, sales of platinum for nonessential uses were prohibited during World War II; similar restrictions adopted early in 1951 were in effect to May 1953. Sales of iridium, osmium, and rhodium used for alloying platinum were controlled during the war; and restrictions were placed on the use of palladium in jewelry, with the object of releasing plant facilities and workers for war production. Platinum and iridium are among the strategic and critical metals being stockpiled. Platinum-bearing raw products produced in Canada were refined in the United States instead of in England during the war so that this output of platinum metals might escape the hazards of ocean transport.

DEFENSE PROGRAM

The defense program has led to a large increase in the demand for and consumption of platinum-group metals, beginning in the latter part of 1950. The chemical industry has bought platinum in extraordinary quantities for use in the manufacturing of fiber glass and as a catalyst for producing high-octane gasoline, both essentially nondissipative uses. Additional large quantities have been used for electrical applications in military equipment.

PRICES, TAXES, AND TARIFF

It is on record that crude platinum from Colombia was sold in Spain in 1788 at 14 cents per ounce (with gold valued at about $17 per ounce). The platinum coinage used in Russia from 1828 to 1845 had a value equivalent to $7.15 per ounce. By 1900 the value of platinum had increased to around $20 per ounce and has trended generally upward since that time, although subject to wide fluctuations. During or immediately after World War I, platinum reached a value of $155 an ounce, palladium $150, and iridium $400 to $450. During 1940 platinum ranged from $26 to $45 per ounce, palladium from $19 to $25, and iridium from $50 to $200. During World War II platinum was stabilized at a ceiling price of $35 per ounce, palladium at $24, iridium at $165, osmium at $50, rhodium at $125, and ruthenium at $35. Prices in 1948 were: Platinum $69 to $101 per ounce, palladium $24, iridium $80 to $120, osmium $100, rhodium $125, and ruthenium $69 to $101. A ceiling price of $93 per ounce for platinum was adopted by the Office of Price Stabilization in April 1952 and was in effect to May 1953. During 1953 the domestic retail prices of the platinum-group metals were as follows per fine troy ounce: Platinum $93 to $95, palladium $24, iridium $170 to $200, osmium $140 to $200, rhodium $125, and ruthenium $75 to $93. The price of crude platinum varies greatly, depending on the iridium content and on the quotations of refined metals.

Platinum mines and refineries are subject to property taxes, income taxes, and excess-profit taxes. Dividend payments to stockholders are subject to income taxes. Jewelry and other luxury items of platinum are subject to luxury taxes; in some states and cities they are subject to sales taxes also. The United States imposes no duties on imports of platinum metals in ores or in unmanufactured form, but platinum manufactured goods are subject to duties ranging from 25 to 55 percent ad valorem.

RESEARCH

The large fabricators of platinum-group metals carry on research leading to improvement of their products and to new uses of the metals. Some large industrial consumers carry on research leading to substitutes for platinum metals. It is to be noted that research in this entire field is highly specialized.

OUTLOOK

Consumption of platinum metals has trended upward almost without interruption for many years, particularly for industrial uses. During the past 20 years the annual consumption of platinum metals in the United States for decorative uses has increased roughly twofold and that for industrial uses fourfold.

The production trends are for declining output in the United States and Colombia, slightly greater output in Canada, and moderately greater output in the Union of South Africa.

The Great Dyke formation of basic rocks in Southern Rhodesia, with a strike length of some 350 miles and an average width of 4 miles, contains showings of platinum metals, some of which have been prospected, and are believed to have commercial possibilities (*2*). Development of the platinum deposits in Ethiopia is being encouraged by the Ethiopian Government, and this may lead to resumption of mining activity and a small, steady production.

International tension or strong industrial activity would have the effect of sustaining the demand for platinum metals at high levels.

It is to be noted that the future expansion of supplies of platinum metals in the United States depends mostly on continuance of imports from foreign countries; however, it may be said that to the extent the United States depends on sources abroad for its supply of platinum metals, the producers in foreign countries depend on this country for the greater part of their market.

PROBLEMS

The dependence of the United States on foreign sources for most of its supply of platinum-group metals imposes the necessity for stockpiling those metals of the group of military importance (platinum and iridium) for an adequate emergency supply. Research might develop substitutes in some uses. The uncertainty as to whether platinum-group metals produced in the Soviet Union will reach world markets and some lack of flexibility of output caused by large byproduct production contribute to problems of supply and price.

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POTASH

By

E. Robert Ruhlman [1]

ALTHOUGH the United States consumption of potash increased more than thirtyfold in the first half of the 20th century, agriculture still does not use enough potash to replace that lost from the soil.

Summary

Although only about 1 percent of the world's known main reserves of high-grade soluble-potash salts (principally potassium chloride) occur in the United States, domestic reserves are adequate for many years at the present rate of production. The remaining 99 percent is in western Eurasia, with 65 percent in Germany and Russia, and the remaining 34 percent distributed among Spain, France, Israel, and Jordan.

As 90 percent of domestic potash production comes from 1 area, the United States has grown in the last 25 years from an importer of potash to the second largest producer in the world and now exports sizable quantities.

The increased demand for potash in the United States in recent years has been met largely by expansion of the domestic industry. Future increased requirements might be supplied by domestic producers, but the possibility of large imports of potash is a factor that discourages further expansion.

The most important potash-producing area in the United States is in Eddy County, N. Mex., which has been active since 1931. Smaller quantities are produced in California, Utah, Michigan, and Maryland.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

The term "potash" was derived from pot ashes, which were originally obtained by evaporating, in iron pots, solutions leached from wood ashes. Present use of the term is the theoretical equivalent in potassium oxide (K_2O)—a compound never found in nature or manufactured but used as a common denominator for comparison of all potassium compounds.

HISTORY

Before World War I, potash supplies and prices were controlled by the German cartel, which in turn was controlled by the German Government.

Following World War I an international syndicate was formed, and the world market was divided between Germany and France (which, as a result of the war, gained control of many former German mines). New producing companies in Palestine, Poland, Russia, and Spain gained admittance to the syndicate and obtained a share of the world market. In 1927 the United States attempted to curb the syndicate's control over domestic sales by instigating an antitrust action. As a result, a Netherlands company was formed to sell potash, from the syndicate-member countries, in the United States. This arrangement continued until 1939.

The blockade of Germany by Great Britain during World War I cut off all foreign supplies of potash and caused intensive efforts in the United States to produce potash from the then known sources. By 1918 at least 128 plants were producing potash from kelp, wood ashes, lake brines, alunite, cement dust, beet-sugar waste, blast-furnace dust, and other miscellaneous sources. After the war ended, foreign supplies of potash again became available, and all but one domestic plant ceased operations. This plant, operated by the American Potash & Chemical Corp. and the largest producer outside of the Carlsbad, N. Mex., area, recovers potash from the saturated brines of Searles Lake in southeastern California.

Realizing the need for a domestic supply of potash, the Congress passed an act in 1911 providing funds to search for domestic sources and to perfect processes for extracting potash from materials of various types. The problem was originally assigned to the Bureau of Soils, United States Department of Agriculture, and the Geological Survey, United States Department of the Interior. In 1924 a bill was passed "authorizing investigation by the United States Geological Survey to determine location and extent of potash deposits in the United States." The bill was amended to include the Bureau of Soils and later again amended to substitute the Bureau of Mines for the Bureau of Soils. The Geological Survey did the geological work and located drill sites, and the Bureau of Mines supervised the drilling. Under this authority, which covered the period 1926–31, 24 holes were drilled under Federal direction, 10 in Texas, 13 in New Mexico, and 1 in Utah. Beds of potash salts described as of possible commercial interest were found in New Mexico and Texas; they varied in thickness from 1½ to over 8 feet and in K_2O content from 9 to 14 percent.

Much publicity was given this Federal project, and the occurrence of potash in the Permian Basin was repeatedly brought to the attention of the interested public and the oil drillers operating in the region. Much of the evidence obtained was the result of close observance by geologists of the brines and cuttings brought to the surface during oil drilling. Thus, it appears that it was more than a coincidence that in 1925, when drilling for oil east of Carlsbad, N. Mex., encountered unusually heavy beds of salines, the cuttings were examined for potash. The examination revealed significant traces of potash minerals. In 1926, the operating company put down a core-drill hole close to the first drill hole and obtained a vertical cross section of the saline strata. The core revealed a substantial thickness of potash minerals. Of major significance was the disclosure of a stratum of sylvinite—a mixture of sylvite (KCl) and halite (NaCl)—of high concentration and of such thickness and lying at such depth as to be commercially minable, if its lateral extensions could be shown to represent reserves large enough to warrant large-scale development. This required further and extensive exploration, participated in by both Federal and private agencies. The work revealed the presence, at a depth of about 1,000 feet, of a deposit of sylvinite equal in concentration of KCl and other essential characteristics to the better deposits of Europe and occurring in quantities warranting development of mines and refineries with assurance of ample reserves. During this period of exploration by deep drilling, 74 core-drill holes were put down in the Permian Basin by private interests, 71 in New Mexico, and 3 in Texas. Potash beds (uneconomic at present) were later found to underlie some 40,000 square miles, including 22 counties in Texas and New Mexico.

DOMESTIC INDUSTRY

The potash industry in the Carlsbad area began as a result of this drilling. The first company, United States Potash Co., began mine development in 1929 and production in 1931. The Potash Company of America was second, with development beginning in 1933 and production in 1934. The third producer, The International Minerals & Chemical Corp., began in 1936 as the Union Potash Co. and was in production in 1940. From 1940–51, these companies supplied the bulk of the domestic requirements and in 1951 had a combined productive capacity of about 1,350,000 short tons of K_2O a year.

Two new companies recently entered the New Mexico field—Duval Sulphur & Potash Co. in 1951 and Southwest Potash Corp. in 1952. Deposits east of the present operations are being studied by 2 separate groups, and in 1954 2 more companies announced plans to begin mine development. In 1953 the New Mexico area produced over 1.7 million tons of marketable K_2O equivalent.

The second most important potash-producing area in the United States is southeastern California, where the American Potash & Chemical Corp. recovers potash salts in addition to many other industrial and agricultural chemicals from the brine in the crystalline salt mass of Searles Lake. Operations were begun in 1916, when potash was first produced. The annual capacity of this plant was about 150,000 tons of K_2O in 1954.

These 2 areas supply about 97 percent of the domestic production; the remaining 3 percent comes from Salduro Marsh, near Wendover, Utah (Bonneville, Ltd.); the wells of the Dow Chemical Co. at Midland, Mich.; and various concerns that produce potash as a byproduct. The present potash industry in the United States can supply all the domestic requirements. In 1953 domestic potash production exceeded the apparent domestic consumption by about 100,000 tons of K_2O equivalent.

Potash deposits in the Texas-New Mexico region are irregular in shape, thickness, and quality. Indications of possible potash deposits on the basis of scattered wells have been found in approximately 3,000 square miles. The area now considered likely for intensive prospecting may increase or decrease as new wells are drilled. Some areas now discredited may develop valuable deposits after more intensive prospecting. Ten horizons have been identified as potential sources of potash. Operations have been limited to about 55 square miles. As this area is mined out and the producers have to search in unknown areas, new deposits undoubtedly will be found.

High-grade potash beds are known to exist in the Paradox Basin in eastern Utah. Because of the folded and faulted condition of the beds and the greater depth from the surface than the New Mexico deposits, little interest has been shown in these beds lately. Development of solution mining might make this an important producing area.

Additional supplies of potash may be available in North America from the Canadian deposits now being developed. The potash beds in the Province of Saskatchewan, Canada, have been drilled over a large area from Unity eastward to the Patience Lake area, disclosing beds up to 17 feet thick and assaying 11 to 24 percent K_2O. In 1952 shaft sinking was begun near Unity by the Western Potash Corp. to develop the deposit about 3,500 feet below the surface. The Potash Company of America was preparing to sink a shaft about 10 miles east of Saskatoon in 1954. Other companies are conducting exploration programs. Reserves have not yet been determined, but results so far indicate that the Canadian deposits rank among the world's largest.

Potash deposits in Canada may extend into the Williston Basin of North Dakota and Montana. Oil-well drilling in the basin has intersected salt beds, but no analyses have been made to determine the character of the salts. If potash deposits are found in this area, they will be at least 7,000 feet below the surface and will be difficult and expensive to recover by conventional mining methods. Solution mining, however, might be possible. The high thermal gradient reported for this area would help to maintain high solution temperature. With phosphate rock and nitrogen available, development of a potash industry in this area would favor local manufacture of complete fertilizers.

GEOLOGY

Twenty-five of the "potash minerals" are given in the accompanying table (table 1).

The occurrences of potassium in nature are as varied as its uses. It is found in many types of rocks and in the waters of lakes, rivers, and oceans. It has been estimated that 3.11 percent of the lithosphere is potash and that the waters of the oceans contain 0.04 percent. Much of the potash occurs in chemical compounds that have no present economic value; the most easily available and usable form is the mineral sylvite.

In the United States, central Europe, and Russia soluble potash salts are mined from the Permian series; in Alsace, France, from the Oligocene; in Poland from the Miocene; and in Spain from the Tertiary. The extent of the Permian Basin in the United States has not been clearly defined but includes parts of Colorado, Kansas, Oklahoma, Texas, and New Mexico.

In New Mexico sylvite, langbeinite, and carnallite have been found in important quantities only on the western flank of the basin, where it rises to an old erosion surface now buried under hundreds of feet of later sediments. Because gradual withdrawal of the Permian Sea was toward the southwest, the salt deposits in southeastern New Mexico are younger than those to the north, and their richness is attributed in part to concentration of the residual brines.

TABLE 1.—*The potash minerals*

Name	Composition	Equivalent K_2O content
Chlorides:		*Percent*
Sylvite	KCl	63.1
Carnallite	$KCl \cdot MgCL_2 \cdot 6H_2O$	17.0
Chloride-sulfates: Kainite	$MgSO_4 \cdot KCl \cdot 3H_2O$	18.9
Sulfates:		
Alunite	$K_2(Al(OH)_2(6)SO_4)_4$	11.4
Polyhalite	$K_2SO_4 \cdot MgSO_4 \cdot 2CaSO_4 \cdot 2H_2O$	15.5
Langbeinite	$K_2SO_4 \cdot MgSO_4$	22.6
Leonite	$K_2SO_4 \cdot MgSO_4 \cdot 4H_2O$	25.5
Syngenite	$K_2SO_4 \cdot CaSO_4 \cdot H_2O$	28.8
Krugite	$K_2SO_4 \cdot MgSO_4 \cdot 4CaSO_4 \cdot 2H_2O$	10.7
Apthitalite	$(K,Na)_2SO_4$	42.5
Picromerite	$K_2SO_4 \cdot MgSO_4 \cdot 6H_2O$	23.3
Kalinite	$K_2SO_4 \cdot Al_2(SO_4)_3 \cdot 24H_2O$	9.9
Nitrates: Niter	KNO_3	46.5
Silicates:		
Leucite	$KAl(SiO_3)_2$	21.4
Feldspars:		
Orthoclase	$KAlSi_3O_8$	16.8
Anorthoclase	$(NA,K)AlSi_3O_8$	2.4–12.1
Micas:		
Muscovite	$H_2KAl_3(SiO_4)_3$	11.8
Biotite	$(H,K)_2(Mg,Fe)_2Al_2(SiO_4)_3$	6.2–10.1
Phlogopite	$(H,K,Mg,F)_3Mg_3Al(SiO_4)_3$	7.8–10.4
Lepidolite	$K,Li(Al(OH,F_2)Al(SiO_4)_3$	10.7–12.3
Zinnwaldite	$H_2K_4Li_4Fe_3Al_8F_8SI_{14}O_{42}$	10.6
Roscoelite	$H_8K(Mg,Fe)(Al,V)_4(SiO_3)_{12}$	7.6–10.4
Glauconite	$KFeSi_2O_6 \cdot nH_2O$	2.3–8.6
Carnotite	$K_2O \cdot 2U_2O_3 \cdot V_2O_5 \cdot 3H_2O$	10.3–11.2
Nephelite	$K_2Na_6Al_8Si_9O_{34}$	.8–7.1

The general geology in the Carlsbad potash district is similar at all producing mines. The surface is covered with sand, caliche, and gypsum to a depth of about 80 feet; the Rustler formation, of Permian age, occurs below the surface for a depth of several hundred feet; the Salado (salt) formation (also Permian), which contains the potash deposits, underlies the Rustler. The maximum thickness of the Salado probably is about 1,600 feet. There are about 45 persistent sulfate beds (gypsum, anhydrite, and polyhalite) and as many halite beds. The potash deposits lie approximately halfway down from the top of the formation as originally laid down and are 650 to 1,500 feet beneath the surface. Throughout this 250-foot thickness are numerous thin beds rich in potash-bearing materials, chiefly sylvite, with langbeinite and some carnallite.

The Searles Lake brines in California occur in a spongelike salt deposit resulting from evaporation of inland water.

The Wendover salt flats brines in western Utah consist primarily of sodium, potassium, and magnesium chlorides.

RESERVES

The world reserves of soluble potash salts are large. The main deposits are in only a few countries—Germany, Russia, Israel, Jordan, Spain, France, and the United States. Estimates of the tonnage of these reserves vary widely. A reasonable figure would be 37 billion metric tons of K_2O, of which about 65 percent are in Germany and Russia. The remainder is broken down as follows: Israel-Jordan, 23 percent; France, 6; Spain, 5; and the United States, 1. Recent discoveries in Canada indicate large deposits of potash there. The Yorkshire deposit in the United Kingdom is estimated to contain 100 to 200 million tons of K_2O.

Many other countries have small- or low-grade deposits. Some of these that were producing in 1953 include: Australia, Chile, China, Eritrea, Ethiopia, India, Italy, Japan, and Korea.

The United States deposits are in New Mexico, California, and Utah. The California and Utah brines contain some 21 million tons of potash. Estimates of the reserves in New Mexico range from 58 to 80 million tons K_2O. With only a small part of the area prospected, this figure will undoubtedly be revised upward with further investigation. The estimates for New Mexico are based on a minimum bed thickness of 4 feet and a minimum K_2O content of 14 percent. If the minimum grade were lowered to 10 percent, as in the German fields, the reserves would be more than doubled; if lowered to 5 percent, as in the former Polish fields, the New Mexico field could still be worked 1,000 years at the current rate of production.

Other possible sources of potash are alunite, carnallite, polyhalite, wyomingite, greensands, porphyry copper tailings, and others. The potash reserves from these sources, if economic recovery methods were developed, would supply the United States for many hundreds of years.

Sea water contains about 0.04 percent K_2O. It has been estimated that each cubic mile contains the equivalent of 1,845,000 tons of K_2O. Methods have been developed that recover 70 percent of the potash, but the selling price would have to be increased to about $1.00 a unit to make this method economic. The 1954–55 domestic minimum price is about 36 cents per unit.

MINING METHODS

Potash is recovered in the United States from two main types of deposits—the large salt beds of the Salado formation in New Mexico and the brines of Searles Lake, Calif.; Wendover Salt Flats, Utah; and the Michigan brines. Mining methods used in the Carlsbad area have been described (7, *12*).[2] At the inception of mining operations, coal-mining methods and equipment were used and have proved suitable.

The room-and-pillar mining method has been used predominately, giving about 60-percent recovery of the ore. Recent experiments have indicated that recovery can be increased by subsequent pillar robbing. Two companies have reported up to 90-percent recovery, and further pillar robbing is planned. Four of the five producers now have plans for this type of cleanup mining. Mechanization has been keynoted through all mining. Continuous mining machines, diesel haulage equipment, and belt conveyors now are in use in several mines.

The mines are about 1,000 feet deep. All but the mines of International Minerals & Chemical Corp. produce from one level. The sylvinite beds in the operating mines range from 3 to 14 feet in thickness.

The recovery methods used by the American Potash & Chemical Corp. at Searles Lake, Calif., are an example of the method of working the second type of deposit. The brine is pumped from an upper and a lower horizon. Brine from the upper horizon is used in producing potash. The corporation has about 40 wells 70 to 120 feet deep and pumps about 3.7 million gallons of brine a day to the refinery. The liquid contains over one-third of its weight in dissolved salts. The approximate composition of the Searles Lake brine from the upper deposit is as follows:

	Percent by weight
Sodium chloride	16. 2
Sodium sulfate	6. 8
Potassium chloride	4. 9
Sodium carbonate	4. 7
Sodium tetraborate	3. 0
Other	. 3
Total salts	35. 9

MILLING AND REFINING

Two basic methods of ore treatment—flotation and fractional crystallization—are utilized in the Carlsbad, N. Mex., area to recover sylvite from its ore. Flotation is conducted in a potassium chloride saturated brine, and the sylvite either is floated from the impurities or depressed and the impurities floated. In the fractional crystallization method, the ore (potassium and sodium chlorides) is dissolved, and by temperature control the potassium chloride is crystallized. This method yields a high-grade product and is sometimes used to clean up flotation tailings.

Langbeinite is produced in the United States by International Minerals & Chemical Corp. This potash salt is separated from halite—its principal impurity—by selective dissolving of the halite.

The American Potash & Chemical Corp. uses evaporation followed by fractional crystallization. The sodium salts are first separated in triple-effected evaporators, leaving a hot liquor rich in potash and borax. Rapid cooling under vacuum causes the potassium salts to crystallize. They are removed by settling and centrifuging.

SUBSTITUTES

There is no substitute for potash in agriculture, and its importance as an essential plant food cannot be overemphasized. It is not only necessary for plant growth but regulates the intake, by plants, of the other minerals and elements necessary for normal plant growth.

BYPRODUCTS, COPRODUCTS, RELATIONSHIP TO OTHER COMMODITIES

In the Carlsbad area potash salt is the major product. Small quantities of sodium chloride are sold for animal feed. Magnesium oxide and hydrochloric acid are now being produced by one company. The potash recovered from brines, however, is only one of many chemicals produced. For example, the principal products of American Potash & Chemical Corp. are potash, borax, boric acid, soda ash, salt cake, bromine, and lithium carbonate.

Cesium and rubidium are known to occur in minor quantities in the Carlsbad potash ores, and other rare elements are associated with potash in this and other localities. Opportunities may develop in future that will encourage research for the recovery and utilization of some of these associated elements as byproducts of potash production.

The largest use of potash is in mixed fertilizer containing also phosphate and nitrogen. The

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

demand for and consumption of these 3 materials are closely related and have been increasing about 10 percent a year for the past 10 years.

OTHER POSSIBLE SOURCES

Potash occurs in many minerals, but the only economic sources at present are the soluble chlorides and sulfates. Many attempts have been made to produce potash from various silicates and other materials. Some companies were successful for short periods when world distribution from the usual sources was interrupted. Some of the more abundant potassium-bearing minerals are alunite, leucite, feldspar, and greensands.

Other sources of potash are organic and inorganic industrial wastes, such as Steffens' waste, blast-furnace dust, and cement dust. Although potash is being recovered from some of them, the quantity is not large, and these sources could never supply more than a fraction of the total domestic requirements.

From 1917–21 the Bureau of Soils investigated the possibility of recovering potash from kelp. A detailed account of this work was published in the Journal of Industrial Engineering Chemistry (vols. 11 to 16, inclusive, 1919–24). Upon the cessation of hostilities, foreign potash again became available in the United States, discouraging further investigation.

PRODUCTION AND FOREIGN TRADE[3]

The world production of potassium salts, showing the rapid growth of the domestic potash industry from only 20,777 short tons of K_2O equivalent in 1924 to over 1,900,000 tons in 1953, is given in table 2.

Table 3 gives the major sources of potash imported into this country and shows the resumption of imports after World War II.

Potash exports are shown in table 4. The major portion of all exports goes to countries in the Western Hemisphere.

USES

Approximately 95 percent of the potash consumed in the United States is used in agriculture—usually in mixed fertilizers. The remaining 5 percent is consumed by the chemical industry.

TABLE 2.—*World production of potassium salts, by countries, 1924–53*

[Short tons of K_2O equivalent]

Countries	1924	1925	1926	1927	1928	1929	1930	1931	1932	1933
United States	20, 777	25, 448	23, 365	43, 508	59, 909	61, 589	61, 269	63, 879	61, 989	143, 376
France	271, 624	343, 799	404, 180	408, 844	448, 240	542, 439	558, 172	405, 513	359, 901	365, 964
Germany	895, 600	1, 735, 945	1, 391, 752	1, 673, 332	1, 864, 130	1, 970, 664	1, 773, 065	1, 187, 884	960, 494	1, 131, 085
Poland	17, 981	41, 443	42, 926	48, 308	61, 300	70, 255	60, 869	57, 916	62, 880	85, 285
Spain	16	4, 464	12, 952	19, 070	26, 984	27, 034	32, 088	33, 737	60, 461	99, 909
U. S. S. R	----------	900	1, 047	1, 212	1, 328	1, 022	9, 000	18, 300	11, 900	46, 431
Israel-Jordan	----------	----------	----------	----------	----------	----------	1, 323	2, 866	5, 622	8, 264
Undistributed	5, 000	6, 000	5, 000	7, 000	6, 000	6, 000	6, 000	5, 000	7, 000	5, 000
Total world[1]	1, 211, 000	2, 158, 000	1, 881, 000	2, 201, 000	2, 468, 000	2, 679, 000	2, 502, 000	1, 775, 000	1, 530, 000	1, 885, 000
	1934	1935	1936	1937	1938	1939	1940	1941	1942	1943
United States	144, 340	192, 790	247, 336	284, 493	316, 947	312, 197	379, 673	524, 867	679, 195	739, 130
France	392, 529	382, 763	406, 616	539, 908	641, 307	649, 255	589, 000	741, 800	682, 324	732, 475
Germany	1, 465, 428	1, 760, 342	1, 788, 647	2, 169, 785	2, 051, 380	2, 272, 649	2, 276, 778	2, 393, 488	2, 291, 445	2, 300, 102
Poland	68, 318	80, 698	92, 522	110, 164	119, 436	N. A.	N. A.	N. A.	N. A.	N. A.
Spain	133, 381	133, 788	17, 492	8, 304	7, 927	28, 137	98, 825	128, 969	98, 826	96, 319
U. S. S. R	104, 719	190, 698	248, 000	293, 211	134, 500	N. A.	N. A.	N. A.	N. A.	N. A.
Israel-Jordan	7, 846	9, 481	12, 927	20, 099	32, 031	35, 013	49, 030	56, 040	57, 435	51, 703
Undistributed	4, 000	4, 000	6, 000	6, 000	96, 000	139, 000	120, 000	123, 000	120, 000	89, 000
Total world[1]	2, 322, 000	2, 755, 000	2, 820, 000	3, 431, 000	3, 400, 000	3, 436, 000	3, 513, 000	3, 968, 000	3, 929, 000	4, 009, 000
	1944	1945	1946	1947	1948	1949	1950	1951	1952	1953
United States	834, 554	874, 230	931, 797	1, 029, 859	1, 139, 863	1, 118, 395	1, 287, 704	1, 420, 323	1, 665, 113	1, 911, 891
France	514, 396	159, 504	633, 265	697, 584	761, 967	880, 198	994, 566	1, 088, 631	1, 162, 927	1, 136, 251
Germany:										
Federal Republic	2, 122, 512	936, 955	318, 077	377, 437	593, 596	825, 402	1, 206, 231	1, 459, 349	1, 712, 644	1, 675, 500
Soviet Zone			725, 975	794, 000	907, 000	1, 210, 000	1, 378, 977	1, 807, 000	1, 987, 000	2, 094, 000
Spain	214, 159	297, 395	150, 509	215, 931	166, 651	167, 045	178, 153	190, 555	199, 611	199, 909
Undistributed	44, 000	135, 000	230, 000	191, 000	290, 000	208, 000	247, 000	317, 000	327, 000	367, 000
Total world[1]	3, 730, 000	2, 403, 000	2, 980, 000	3, 307, 000	3, 860, 000	4, 409, 000	5, 290, 000	6, 283, 000	7, 055, 000	7, 400, 000

[1] Partly estimated.

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[3] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

TABLE 3.—*United States imports of potash materials by countries, 1930–53*

[Gross weight in short tons]

Countries	1930	1931	1932	1933	1934	1935	1936	1937
Belgium-Netherlands [1]	339, 713	188, 358	72, 402	124, 248	122, 683	174, 861	158, 571	156, 444
Chile	10, 601	9, 502	5, 080	16, 269	25, 399	31, 759	47, 707	61, 473
France	4, 072	7, 297	8, 488	8, 707	8, 050	13, 504	10, 205	30, 508
Germany	590, 920	333, 174	213, 646	249, 398	225, 992	316, 608	246, 847	535, 732
Israel-Jordan			500	3, 040	924			106
Spain	26, 277	30, 461	18, 640	67, 235	76, 690	43, 933	11, 606	58
U. S. S. R	1, 257			123	15, 013	10, 628		10, 693
Other countries	7, 244	8, 762	11, 763	10, 564	11, 416	12, 302	18, 740	15, 515
Total	980, 084	577, 554	330, 519	479, 584	486, 167	603, 595	493, 676	810, 529

Countries	1938	1939	1940	1941	1942	1943	1944	1945
Belgium-Netherlands [1]	73, 033	28, 944						
Chile	44, 648	55, 410	55, 571	23, 031	14, 514	19, 767	9, 439	81
France	66, 803	59, 170	186, 257	30				2, 848
Germany	220, 982	91, 184	1, 626	2				
Israel-Jordan	10, 256	10, 498						
Spain	15, 941		30, 882	16, 818	188		368	772
U. S. S. R	5, 206	163			1, 557	25, 212	4, 331	6, 861
Other countries	13, 518	10, 443	4, 701	6, 791	5, 169	449	5, 003	6, 650
Total	450, 387	255, 812	279, 037	46, 672	21, 428	45, 428	19, 141	17, 212

Countries	1946	1947	1948	1949	1950	1951	1952	1953
Chile	4, 687	2, 510	64	6, 970	20, 434	12, 305	27, 965	12, 557
France	5, 005	19, 040	17, 883	29, 602	63, 534	59, 193	73, 719	50, 346
Germany		11, 703	14, 130	660	188, 410	372, 409	210, 721	140, 350
Poland-Danzig	30			87	32, 554	11, 755		
Spain		38	95		49, 272	94, 878	40, 656	42, 791
U. S. S. R	3, 452	19	14, 275		17, 974			
Other countries	3, 025	17, 733	6, 453	6, 400	9, 312	23, 821	4, 376	7, 069
Total	16, 199	51, 043	52, 890	43, 719	381, 490	574, 361	357, 437	253, 113

[1] This country is not a producer; reported imports to United States from Belgium-Netherlands probably originated in France and Germany.

TABLE 4.—*United States exports of potash by type, 1930–53*

[Gross weight in short tons

	1930	1931	1932	1933	1934	1935	1936	1937
Potash fertilizer material	17, 042	32, 460	2, 034	28, 086	27, 988	75, 983	103, 031	103, 031
Potassium salts (chemical)	1, 256	1, 158	88	1, 275	2, 121	3, 641	2, 333	2. 094
Total exports	18, 298	33, 618	2, 921	29, 361	30, 109	79, 624	105, 364	105, 125

	1938	1939	1940	1941	1942	1943	1944	1945
Potash fertilizer material	84, 137	136, 750	93, 060	91, 950	84, 402	111, 541	110, 057	104, 687
Potassium salts (chemical)	2, 616	3, 579	14, 180	12, 883	13, 307	20, 133	15, 444	18. 966
Total exports	86, 753	140, 329	107, 240	104, 833	97, 709	131, 674	125, 401	123, 653

	1946	1947	1948	1949	1950	1951	1952	1953
Potash fertilizer material	96, 822	102. 939	104, 176	111, 156	107, 972	109, 139	94, 678	83, 412
Potassium salts (chemical)	23, 905	21, 970	23, 892	15, 601	9, 165	15, 112	6, 522	4. 796
Total exports	120, 727	124, 909	128, 068	126, 757	117, 137	124, 251	101, 200	88, 208

Potassium chloride is the most widely used form of potash in fertilizers, followed by the sulfate and the double salt, potassium-magnesium sulfate. Several grades of potassium chloride, from 62 percent minimum K_2O to 20 percent K_2O, standard and granular, are utilized in fertilizers. The higher grade is most widely used and was 90 percent of total United States marketable production in 1953. Potassium sulfate contains 49 to 51 percent K_2O minimum, and potassium-magnesium sulfate (sulfate of potassium-magnesia) contains 20 percent K_2O and 18 percent MgO.

A small part of the output goes directly to consumers for application as a single-element fertilizer. The major share, however, goes to fertilizer factories, where it is blended with other plant foods to produce a mixed fertilizer.

The nonfertilizer uses of potassium compounds derived from potassium chloride are based, in general, on caustic potash or potassium hydroxide (KOH) derived from the electrolysis of high-purity potassium chloride. Potassium compounds are used in manufacturing detergents, soap, glass, ceramics, textiles, dyes, drugs, matches, explosives, propellants, and other products. Potassium sulfate is used in manufacturing plaster-board and gypsum tile and as a raw material in the production of potash alums.

The demand for potash in the chemical industry increased from 74,000 tons of potash salts in 1940 to over 131,000 tons in 1951.

UNITED STATES TARIFF

All potash fertilizer materials enter the United States duty-free. The tariff on potassium chemicals ranges from three-fourths of a cent a pound for potassium carbonate (K_2CO_3) and potash alum (potassium-aluminum sulfate) to 25 cents a pound for potassium iodide. On other potassium chemicals, not separately classified, the rate is 12½ percent ad valorem.

PRICES

Before the growth of the domestic industry, potash was sold by importers at Atlantic and Gulf ports, on the basis of 60 percent K_2O, with an adjustment if the potash content were more or less. Importers equalized shipping costs at all basing points. In 1927 discounts for quantity, which earlier had been as high as 10 percent, were discontinued. Seasonal discounts (4 to 12 percent) for deliveries between July 1 and October 1 were continued.

In general, this price structure was in existence when the domestic industry began producing in the early 30's. In 1934 the 60-percent K_2O basis was increased to 62½ percent. To simplify accounting, the American Potash & Chemical Corp. initiated the 20-pound "unit" basis in place of the tonnage basis. Under this system the price per ton is figured by multiplying the percent of K_2O by the price per unit. This method of pricing was soon adopted by the other firms, although certain products still are sold by the ton. The split-discount system of pricing, suggested by the Thorp-Tupper Report, was used successfully for several years, with discounts as high as 16 percent in 1952–53 (*15*). The pricing system was changed in the 1954–55 price schedules. Under this system, different prices are in effect for various periods during the year, ranging from 36 to 43 cents per unit K_2O. This pricing system encourages early contracting, thus enabling the producing companies to schedule the output more evenly throughout the year. Port prices to specified Atlantic and Gulf ports were issued in the 1954–55 schedules.

Price schedules are issued annually by all domestic producers. All prices are now for minimum 40-ton lots, f. o. b. mine, per unit K_2O.

Prices for sulfate of potash-magnesia are still quoted per short ton.

STATISTICS

There are two Government agencies and one private organization which compile statistics on various phases of the potash industry. The Bureau of Mines collects statistics from the domestic producers and issues annual releases on domestic production and consumption and world production. The Bureau of the Census, Department of Commerce, collects import and export data which is furnished to the Bureau of Mines. The American Potash Institute, Inc., an association of most of the major domestic producers, issues quarterly releases on all the principal companies' deliveries in North America by destination.

TRANSPORTATION

Before World War II some potash was shipped from Carlsbad by rail to Gulf ports and then coastwise by water to Atlantic coast ports. At present, little or no domestic potash for domestic consumption is shipped by water, since low charter rates are not available. Table 5 gives the August 1954 freight rates from Carlsbad, Trona, and Wendover.

TABLE 5.—*Freight rates for muriate of potash and sulfate of potash from the major producing centers for domestic shipments, August 1954*[1]

	From Carlsbad[2]	From Trona	From Wendover
To:			
Texas City, Tex	$11.11	$14.55	$14.55
Albuquerque, N. Mex	8.78	13.60	14.55
Denver, Colo	12.02	14.55	12.02
Fargo, N. Dak			
Minneapolis, Minn			
Omaha, Nebr			
Sioux City, Iowa			
St. Louis, Mo			
Chicago, Ill	13.43	15.61	13.97
Los Angeles, Calif	13.60	8.05	[3] 21.80
Troy, Ala	14.30	17.02	16.48
Indianapolis, Ind	14.63	16.81	15.73
Cincinnati, Ohio	14.84	17.02	15.93
Columbus, Ohio			
Detroit, Mich			
Louisville, Ky			
Toledo, Ohio			
Albany, Ga	14.84	17.46	16.91
Jacksonville, Fla	15.39	18.11	17.57
Buffalo, N. Y	15.39	17.68	16.59
Rockingham, N. C	15.93	18.99	18.44
San Francisco, Calif	16.00	8.05	[4] 12.80
Portland, Oreg	17.00	13.60	[5] 12.00
Seattle, Wash	17.00	13.60	[5] 12.80
Boston, Mass	17.02	18.88	18.88
Portland, Maine	17.02	18.88	18.88

SOURCE: Panhandle and Santa Fe Railway Co. and Western Pacific Railway.

[1] Excluding 3 percent Federal excise tax.
[2] Includes manure salts from Carlsbad.
[3] Reduced to $11.80, fall of 1954.
[4] Muriate only; sulfate is $13.20.
[5] Minimum, 60,000 pounds.

GOVERNMENT CONTROL

A large proportion of the potash-bearing lands of New Mexico are owned either by the State or by the Federal Government and are leased from them by the individual firms. The salt lakes or marshes used as sources of potash by the American Potash & Chemical Corp. and Bonneville, Ltd., are partly federally owned. Under an act of Congress approved February 7, 1927, entitled "An Act to promote the mining of potash on the public domain" (44 Stat. 1057, 30 U. S. C. 281), the Secretary of the Interior is authorized to issue prospecting permits for a period not to exceed 2 years, subject to such rules and regulations as he may prescribe. The Code of Federal Regulations, Title 43.—Public Lands, Interior, Part 194. Potassium Permits and Leases, includes sections on Purpose of Regulations, Right Under Permit, Reward for Discovery, Statutory Authority, Royalties and Rentals, Leases for Potassium Deposits and Associated Minerals, Sale of Production, Operations, and Limitations on Holdings.

Since almost all of the major producing areas in the United States are on the public domain, the rules and regulations under which the Secretary of the Interior issues permits have a vital effect upon entry into the industry.

In addition to these controls, certain direct controls are exercised over potash production. The most important are those concerning safety and conservation used by mining inspectors.

Congressional hearings were held during 1953 regarding the threat of flooding the United States market with cheaper potash from Communist-dominated areas. In 1954 the Treasury Department initiated an investigation of imports to determine possible violations of antidumping legislation.

RESEARCH

As a result of the potash shortage during World War I, Government agencies, private research organizations, and individuals investigated all the then known sources of potash.

In 1931 the United States potash industry contributed only 4 percent of the world supply. In 1953 it produced over 25 percent. This immense expansion of domestic potash production was made possible by discovery of the large deposits in New Mexico.

New and improved mining and beneficiation techniques have been developed throughout the years by industry and Government. Mine recovery has been improved in all mines, and in one mine recovery up to 90 percent was reported. Ore recovery, previously by chemical means, now is accomplished mainly by physical ore-dressing methods (*3*). Private company research has further improved ore-dressing techniques.

Attempts have been made in the United States, Canada, and England to recover potash from underground deposits by solution mining. To date none of these has been successful.

In attempting to increase the supply of potash in the United States, the Bureau of Mines devised methods for recovering potassium salts from alunite, wyomingite, polyhalite, greensands, and porphyry-copper tailings. The methods were not perfected because the indicated cost was excessive.

OUTLOOK

To continue to feed and clothe the ever-growing population of the United States, which is increasing at the rate of 1½ million a year and will reach 190 million by 1975, according to the Bureau of the Census, farm production will have to be increased by such means as bringing more land into production, increasing the productivity of the soil, and introducing new disease-resistant crop varieties.

Although land is being improved in certain small areas, the average fertility of farmland in the United States is declining. It has been estimated that about 5 million tons of potash is removed from the soil annually by cropping and should be replaced to maintain productivity of the overall farm acreage. Additional quantities of potash are lost through leaching and erosion, and attempts to curb losses by these actions should be strengthened. A large additional tonnage of K_2O would be required for a number of years to restore the now depleted lands to maximum productivity.

The growth of the fertilizer industry in the United States—rapid in the last 50 years—will undoubtedly continue for many years. At the opening of the century, only 200,000 tons of commercial fertilizer was used in the United States, and in 1950 total consumption exceeded 18 million tons. This spectacular rate of increase will not be repeated in the next 50 years, but consumption will continue to increase.

Although mechanization has played a large part in decreasing the total labor cost of agriculture, the application of fertilizer has aided in lowering the costs and will continue to play an important part further to lessen labor costs through increasing land productivity.

The average plant-food content of mixed fertilizers in the United States has been increasing, reaching a peak of about 24 percent in 1953. The trend toward higher analysis materials will continue as problems of production are solved and consumers become better acquainted with the more concentrated fertilizers.

The proportion of domestic crop production resulting directly from the use of fertilizer has been steadily increasing—from about 14.5 percent in 1938 to 22 percent in 1948 and 25 percent in 1951. To produce enough food and fiber crops in the United States to satisfy the needs of the ever-growing population, a larger percentage of future crops will result from the use of fertilizer.

No new agricultural uses for potash are foreseen; however, the demand for potassium sulfate may increase as new-type fertilizers are developed.

New industrial uses of potassium compounds may be developed, such as new-type explosives and propellants, but the tonnage involved probably will not be large.

Since 1900 the domestic potash supply has changed significantly. Less than 2 percent of the potash consumed in 1900 was of domestic origin, whereas in 1953 over 90 percent of the domestic consumption was produced in the United States.

From 1931 (when mining of potash salts began in New Mexico) until 1951, the grade of ore decreased progressively. Originally the average K_2O content of the ore was about 25 percent; by 1951 it had dropped to 20.4 percent. In 1952 and 1953 the average grade increased slightly. Mining of progressively lower grade ores undoubtedly will continue as the higher grade beds are depleted and the demand for potash continues to increase.

Although the reserves are enormous, insoluble potash minerals will not contribute materially to the world supply in the foreseeable future.

New deposits may be found in the Texas-New Mexico area. In addition, other areas that may contribute to world potash supplies are the Paradox Basin in eastern Utah, Williston Basin of North Dakota and Montana, Saskatchewan deposits of Canada, and the Yorkshire deposits of England.

The expansion of the potash industry in France and Spain since World War II and the rebuilding and expansion of the industry in western Germany are again making large quantities of foreign potash available to the United States and other consumers.

PROBLEMS

The important problems confronting the potash industry are those of reserves, conservation, utilization of lower grade ores, and foreign competition. Because such a large part of the domestic potash deposits is on the public domain, the Government has a special responsibility for constant surveillance and review of the regulations needed to insure a healthy domestic industry under all economic conditions.

An important problem is to increase the recovery of potash through development of modified or new mining methods and new beneficiation methods that will permit utilization of lower grade sylvinite ores than being mined now. Because of such factors as the acreage limitations in the leasing regulations, companies abandon large quantities of ore and lower grade potash-bearing material. In the interest of conservation, these unrecovered reserves should be utilized, because, once an area has been abandoned, reentry would be dangerous and the cost probably prohibitive.

Means to permit utilization of the huge reserves of polyhalite in both Texas and New Mexico have long been sought. This mineral occurs in larger quantities than the present ores, and reserves have been estimated to exceed 250 million tons of K_2O.

The possibility of increased competition from European producers, both inside and outside the Russian-controlled area, must be faced by the domestic industry.

A long-range problem—to insure complete self-sufficiency of potash for the United States virtually forever—requires ways to utilize the potash contained in insoluble minerals, rocks, and sea water. Greensands, Georgia shales, wyomingite, alunite, and porphyry-copper tailings all contain appreciable quantities of potash and during previous shortages have been investigated as possible sources of potash.

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PUMICE AND PUMICITE

By

Henry P. Chandler[1]

OVER 2,000 years ago the Romans used pumice for construction purposes, but only in the last 20 years have its virtues as a lightweight aggregate earned it a substantial position in the construction industry of the United States.

Summary

From a domestic production of 23,271 tons in 1910 the output of pumice increased to 597,044 tons valued at $2,266,981 in 1952.

Pumice long has been used for certain abrasive and polishing purposes, but its wide application as a lightweight concrete aggregate is a recent development. In 1944, some 17,511 tons was used in connection with concrete construction compared to 553,899 tons in 1952. The growth of this market has led to development of many pumice and pumicite deposits in the Pacific Coast and Rocky Mountain States.

The bulk of the pumice consumed in the United States is produced from domestic mines; but some is imported, principally for use along the Atlantic seaboard.

As pumice and pumicite have numerous advantageous properties and relatively low average price, they are in a favorable position to compete in various markets. However, the industry has many problems, and there is need for maintaining and improving research and other fact-finding activities to serve as a basis for optimum development of the industry.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

The word "pumice" is applied to highly cellular, glassy lava. Its cellular structure is due to the expansion during cooling of steam and gases contained in the molten lava. Pumicite, or volcanic ash, is an accumulation of finely divided, glasslike particles blown from volcanoes during periods of eruption and subsequently more or less classified during transportation by winds. It is also called volcanic tuff. It may be wholly unconsolidated, or the particles may be more or less cemented or agglomerated.

OCCURRENCE

Although pumice and pumicite occurrences are common in areas of certain types of volcanic activity, commercial deposits are not so widespread in occurrence, for in many instances the material does not have the physical properties needed to make it commercially valuable. For example, block pumice of desirable grade is produced in very few localities. Pumicite of good grade, however, has a much wider distribution, and pumice suitable for making lightweight concrete aggregate occurs in many Western States.

Pumice is found near volcanoes, in solid blocks or fragments down to sand size or in mixtures of fines or moderately coarse pieces, and is generally white or light gray. Blocks of pumice, when dry, will float on water, and their apparent specific gravity is less than 1.

Pumicite occurs as a powder, varying from coarse to fine, usually nearly white or light gray; it is found in irregular bedded deposits. Commercial deposits usually average 6 to 25 feet in thickness. These deposits may be many miles from their volcanic source. In general, the coarsest grains are found nearest the source and the fine dust at the extreme edge of the zone.

Many deposits are erratic in shape and extent. For this reason, considerable exploration work is necessary to make certain that a given deposit is large enough to warrant extensive expenditures for development.

RESERVES

No estimates of the reserves of pumice and pumicite in the United States have been made, but they are thought to be adequate to maintain the present production for many years. No commercial deposits are known east of Kansas.

TECHNOLOGY

The physical properties of pumice or pumicite from different deposits vary widely. Variations in texture and composition affect the value of any particular deposit, and must be fully investigated when a new enterprise is being considered. Efficient abrasive material must be uniform in texture, and abrasive qualities depend to a considerable extent upon the characteristics of the individual particles. The presence of sand grains, or other foreign minerals, is harmful because they may scratch polished surfaces. A like hazard exists with lump pumice having nonuniform texture or cell structure.

A typical analysis of pumice and pumicite shows the following chemical composition:

	Percent
Silica (SiO_2)	72. 0
Alumina (Al_2O_3)	14. 0
Potash and soda (K_2O and Na_2O)	7. 0
Lime and magnesia (CaO and MgO)	2. 5
Iron oxide (Fe_2O_3 and FeO)	1. 0
Loss upon ignition	3. 5
	100. 0

Pumice and pumicite usually are mined by opencut methods, although there are some instances where systems of tunnels and drifts were used.

After mining, the crude pumice is crushed and dried (if necessary) and screened into commercial sizes, most of which are one-fourth inch and finer. Only a fraction of 1 percent of the pumice produced in the United States is used in lump form.

TABLE 1.—*Production by States, 1951 and 1952*

State	1951		1952	
	Short tons	Value	Short tons	Value
California	264, 411	$1, 228, 569	129, 780	$793, 716
Idaho	83, 528	133, 192	88, 085	141, 253
New Mexico	245, 564	884, 311	217, 482	755, 139
Oregon	47, 026	137, 136	59, 578	201, 809
Utah	9, 422	11, 478	(1)	(1)
Washington	5, 105	10, 832	3, 604	8, 089
Wyoming	1, 867	9, 141	2, 851	10, 918
Other States [2]	93, 019	338, 248	95, 664	356, 057
Total	749, 942	2, 752, 907	597, 044	2, 266, 981

[1] Included with "Other States" to avoid disclosure of individual company operations.

[2] Includes States indicated by footnote 1 and Alaska (1951), Arizona (1951–52), Colorado, Kansas, Montana (1951), Nebraska, Nevada, Oklahoma, and Texas (1951–52).

The consumption of pumice in the United States made relatively little progress until 1945, when its acceptance by the construction industry as lightweight aggregate increased its use many times. Over 60 firms now mine pumice and pumicite. The deposits of commercial importance are almost entirely in the Pacific Coast and Rocky Mountain States, with a few in western Kansas and western Nebraska. Most of the pumice imported—21,986 tons in 1952—was consumed on the Atlantic seaboard.

USES

Pumice for concrete aggregate is either used by the producers themselves or shipped in bulk to contractors or makers of concrete products. Since it is a low-price, bulky product, it cannot stand high transportation costs, and markets usually are limited to a fairly short shipping radius.

Pumice, when employed as a lightweight aggregate in precast masonry units or in poured concrete in place of sand and gravel, considerably reduces the weight of the product. Concrete, which commonly weighs 140 to 150 pounds a cubic foot when made with heavy aggregates, can be made to weigh as little as 60 pounds a cubic foot, thereby reducing the load on the steel framework in large buildings. The lighter units are easier to handle and install and cost less to transport, but strength of pumice concrete usually varies directly with the weight. Other advantages claimed for pumice concrete are heat and sound insulation, fire resistance, ease of cutting and shaping, and an ability to be penetrated by and to hold nails and screws.

Fairly coarse, screened pumice may be used as loose-fill heat insulation in the floors, walls, and ceilings of buildings and for acoustic tile and plaster.

Lump pumice is used as an abrasive; in silver plating; by lithographers; for rubbing down wood or metal surfaces and paint and varnish finishes; and for cleaning and polishing stone and glass. A type of pumice block is used for cleaning cooking utensils.

Ground pumice and pumicite are used in concrete admixtures; in cleaning, scouring, and polishing compounds; in tooth paste and powders; as an abrasive in rubber erasers; for abrading and polishing hard rubber and fiber board; in mechanics' hand soaps; and in metal polishes and in sweeping compounds.

PRICES

As reported in Oil, Paint and Drug Reporter, September 13, 1954, quotations on domestic and imported pumice: Domestic, coarse-ground: 0, ½, 1, 1½, 2 and 3, in bags, ton lots, New York, 3⅝ to 4¼ cents a pound; smaller lots, 3⅞ to 4½ cents. Imported, Italian, silk screened, fine, in bags, ton lots, 4 cents a pound, coarse 6½ cents sun-dried; fine or coarse, in bags, ton lots, 2½ to 4 cents. Pumice in barrels is ½ cent a pound higher.

The E&MJ Metal and Mineral Markets quoted pumice per pound, f. o. b. New York or Chicago, in barrels, powdered, 3 to 5 cents; lump, 6 to 8 cents.

Production and value of pumice and pumicite in the United States in 1952

	Short tons	Value	Average value
Crude material	230, 977	$508, 285	$2. 20
Prepared material	366, 067	1, 758, 696	4. 80
Total	597, 044	2, 266, 981	3. 80

TABLE 2.—*Pumice and pumicite sold or used by producers in the United States, 1951 and 1952*

Use	1951		1952	
	Short tons	Value	Short tons	Value
Abrasive:				
Cleansing and scouring compounds and hand soaps	8, 205	$124, 314	17, 308	$177, 609
Other abrasive uses	4, 485	318, 013	5, 121	248, 977
Acoustic plaster	3, 761	112, 518	3, 934	100, 097
Concrete admixture and concrete aggregate	720, 170	1, 988, 204	553, 899	1, 525, 331
Other uses [1]	13, 321	209, 858	16, 782	214, 967
Total	749, 942	2, 752, 907	597, 044	2, 266, 981

[1] Insecticides, insulation, brick manufacture, filtration, solvents, roads (surfacing and ice control), absorbents, soil conditioners, and miscellaneous uses.

Imports of pumice into the United States [1] during 1952

	Short tons	Value	Average value
Crude pumice	21, 986	$135, 305	$6. 17
Manufactured pumice	478	9, 792	20. 42
Total	22, 464	145, 097	6. 45

[1] These imports were from Italy and Greece.

TARIFF

The duty on imported pumice is: Unmanufactured, valued at less than $15 a short ton, $1.00 a ton. Valued at over $15 a short ton, ⅛ cent a pound. Manufactured pumice, ½ cent a pound. Articles made with pumice, 17½ percent ad valorem.

OUTLOOK

The trend in the building industry is toward the use of lightweight concrete, and in the Western States where pumice is available at competitive prices this material is expected to compete successfully. In the eastern United States competition with other lightweight aggregates and imported pumice is difficult or impossible. However, the rapidly expanding industrial areas in the West are good markets for increasing sales of pumice for construction uses.

The total sales of pumice and pumicite for the abrasive and other uses that they have long served have declined in the last few years; consequently, the current outlook in these markets is less encouraging.

PROBLEMS

Pumice producers have the economic and operating problems with which business ordinarily must cope and also some that are peculiar to the commodity. The phenomenal increase, during the recent building boom, of sales of pumice for use as lightweight aggregate tends to obscure the need for development in other markets. The low cost at which pumice can be produced gives it an opportunity to compete if the product can be offered with convincing technical justification. If the pumice and pumicite industry is to diversify its sales more successfully than at present, considerable effective research on the utility of these commodities in many fields of use will be needed.

Those firms that produce and use pumice as concrete aggregate have numerous economic and technical problems also. The competitive position of pumice aggregate is variable because of such factors as transportation cost, quality, concrete specifications, and availability of research facilities. The producers are relatively small, and research facilities in the industry are limited.

To foster optimum development of pumice production and use, the maintenance and improvement of the flow of information on deposits, chemical and physical properties, production and processing, uses, and economics are needed.

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QUARTZ CRYSTAL

(Electronic Grade)

By

Waldemar F. Dietrich [1]

QUARTZ, the commonest of all solid minerals, is used by the hundreds of millions of tons per year as sand and crushed rock, yet demand for a mere 1,000 tons a year in the form of quartz crystals, suitable for frequency control in electronic communications equipment, inspired one of the most intensive raw-materials searches of World War II. Successful resolution of the basic problems of electronic-grade quartz-crystal supply, first by expansion of natural crystal production and next by the development of processes for growing synthetic crystals, is an outstanding example of effective cooperation of Government and industry opening new vistas of future achievement in the electronic era.

Summary

Since 1921 quartz crystal has been the preferred material for accurate frequency control in radio transmitters and many other electronic devices. These applications utilize the piezoelectric properties of quartz, by which a mechanical force produces an electrical charge and vice versa.

This mineral suddenly became an important strategic material immediately before World War II, when quartz-crystal oscillators were adopted by the United States Army Signal Corps for frequency control in mobile radio communications units.

The peacetime industrial consumption of electronic-grade quartz crystal in 1953–54 was less than 100,000 pounds a year. The imports for military requirements in World War II reached a peak of more than 3,000,000 pounds in 1943 and averaged over 2,000,000 pounds a year throughout the war. Largely due to rejections on inspection, the quantities available for consumption were much less than imports. Actual consumption reached a peak of 1,858,000 pounds in 1944 and averaged nearly 1,300,000 pounds a year in 1942–45.

Brazil is the only significant producer of electronic-grade natural quartz crystal. Technical and financial assistance by the United States Government during World War II was successful in procuring enough quartz crystal to meet requirements.

Since World War II the United States Army Signal Corps has sponsored and subsidized a comprehensive synthetic-quartz-crystal program in cooperation with industry and private and Government research organizations. Parallel studies of substitute materials gradually declined in importance as it became increasingly apparent that quartz-crystal synthesis was destined for technical and commercial success.

With the ultimate assurance of adequate and dependable supplies of synthetic quartz, the long-range outlook envisions expansion of industrial applications and narrowing of the wide gap between peacetime and full mobilization requirements. Meanwhile, an adequate stockpile of quartz must be maintained.

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[1] Chief, Branch of Ceramic and Fertilizer Materials, Bureau of Mines.

BACKGROUND

HISTORY

Quartz (silica—SiO_2) is the commonest mineral other than water and comprises fully 12 percent of the earth's crust, but quartz crystals of suitable size and free from objectionable flaws for use in radio equipment are relatively scarce in economic concentrations.

Quartz crystal is one of the few natural materials that has piezoelectric properties suitable for practical applications. When a pressure (force) is applied to the surface of a quartz crystal, an electrical charge is produced that is proportional to the applied pressure. Conversely, quartz crystals expand or contract in certain directions in proportion to the applied voltage.

The piezoelectric properties of quartz were discovered by Pierre and Jacques Curie in 1880–81. Practical applications of these properties were confined largely to laboratory test work, until Professor Langevin, in 1917, initiated development of an ultrasonic device for the French Navy, for detecting submarines. Quartz-crystal plates in an oscillating electronic circuit were used to produce ultrasonic impulses. Although the ultrasonic submarine detector was not perfected until after the Armistice, its development vastly stimulated interest in practical applications of ultrasonics and related phenomena (*3*, *8*).[2]

In 1921 Prof. Walter G. Cady of Wesleyan University (Connecticut) discovered that crystalline quartz could be used to control the frequency of radio oscillator circuits. The noncrystal circuit-type oscillators then in use in radio and carrier telephony were altered by disturbing influences which caused frequencies to drift or wander from their nominal value. This made stabilization of the tuning of the receiving instrument difficult and required assignment of relatively broad frequency bands to avoid excessive interference. The difficulty largely is overcome by inserting in the circuit a properly dimensioned and mounted quartz crystal whose natural frequency of mechanical vibration corresponds to the frequency of the oscillating electrical circuit. The quartz thus dimensioned and mounted serves as a piezoelectric resonator with a fixed mechanical vibration frequency which reacts electrically to control the frequency of the driving circuit. In other applications, quartz or other piezoelectric crystals are used as transducers to convert electrical forces into mechanical forces, as in a loud speaker or in an ultrasonic generator wherein the mechanical expansion and contraction set up in the crystal by an oscillating current transfers sound waves to an acoustic medium. The converse effect is illustrated by a microphone, in which the oscillating pressure of a sound wave is converted into an oscillating electrical current of the same frequency.

Among elastic substances, quartz has the highest known efficiency of transfer of energy back and forth between strain and motion or from potential to kinetic energy. This is expressed by electronic and acoustic engineers as the "*Q*" factor, which may be defined as the ratio of energy stored to energy dissipated per cycle. The *Q* factor varies with the vibration mode that is excited. The higher the *Q* the lower the energy losses when the crystal is in operation. Quartz crystals used in electronic circuits commonly have *Q*'s in the hundred thousands, exceeding the *Q*'s of the best resonant electrical wire-circuits by more than a hundredfold. A quartz bell mounted to avoid transmission of its vibrations to its supports and housed in a vacuum so that it radiates no sound would vibrate for a longer time, once struck, than a bell made of any other known material for an equal quantity of energy received. Because of its high *Q* and low temperature coefficient of frequency, quartz-crystal oscillators are highly effective in holding the frequency of an oscillatory system to the frequency preferred by the quartz element. The temperature coefficient of frequency varies with orientation of the oscillator with the crystallographic axes of the crystal from which it is cut and approaches zero for limited temperature ranges in certain cuts.

The superior properties of quartz led to its adoption before World War II by the United States Army Signal Corps for frequency control in essentially all army communications equipment. This decision resulted in a peak wartime demand for quartz of the order of 20 times the normal civilian demand. Germany and Japan each entered World War II with substantial stockpiles of quartz but were unable to procure much additional supply during the war. Since 1940 the United States has taken about 85 percent of the free world's annual

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[2] Italicized figures in parentheses refer to items in the bibliography at end of this chapter.

production of quartz, most of which was produced in Brazil. Substantial deposits of electronic-grade quartz crystal are reported to occur in the Soviet Union. Soviet production since 1938 has been on a scale that indicates self-sufficiency in crystal.

The use of quartz-crystal plates in ultrasonics, where the vibrations of the quartz provide a source of sonic waves of set frequency, opened up a new field of science. It has found many applications, including radar, range finders, ultrasonic separators, and dewatering equipment (*4*).

INDUSTRY STRUCTURE

In 1954 electronic- and optical-grade quartz was imported by about 15 firms, all in or near New York, N. Y. Of these, 7 firms imported over 95 percent of the total.

At the peak of World War II demand, about 140 crystal unit plants were in operation. About 25 producers of quartz piezoelectric units supply the normal industrial demand. The manufacturing units are well distributed geographically. In 1953 production was reported to the Bureau of Mines from 52 producers in 19 States and the Territory of Hawaii. Pennsylvania, Kansas, California, New Jersey, and Illinois lead in the quantity of production, in the order given (table 1).

Brazil has been the source of nearly all of the imports of crystal, although sporadic shipments have been obtained from Madagascar, Portuguese West Africa, Australia, Guatemala, and Colombia.

TABLE 1.—*Consumption of electronic-grade quartz and production of piezoelectric units in the United States in 1953, by States*

State	Consumption of electronic-grade quartz		Production of piezoelectric units [1]	
	Number of consumers	Pounds consumed	Number of producers	Number of units produced
California	9	40, 400	10	919, 500
Connecticut and Massachusetts	3	3, 400	3	74, 400
Florida, Georgia, and North Carolina	3	12, 800	3	254, 000
Hawaii	1	1, 000	1	11, 000
Illinois	3	19, 800	3	365, 900
Iowa and Nebraska	3	18, 100	6	583, 800
Kansas	3	(2)	4	1, 060, 200
New Jersey	4	36, 300	4	496, 400
New York	4	14, 200	5	339, 500
Pennsylvania	5	155, 800	6	2, 748, 300
Virginia	1	6, 700	1	31, 500
Other States [3]	4	[4] 90, 700	6	333, 200
Total	43	399, 200	52	7, 217, 700

[1] For radio oscillators, telephone resonators, and filter plates.

[2] Included under "Other States" to avoid revealing individual company figures.

[3] Includes Maryland, Ohio, Texas, and a quantity unspecified by State.

[4] In addition to "Other States"; includes consumption of electronic-grade quartz in Kansas.

GEOLOGY

Commercial deposits of electronic-grade quartz crystal in the free world are almost entirely confined to Brazil, principally the States of Minas Gerais, Goiaz, and Bahiá and recently, Pará, which together have produced at least 95 percent of the total Brazilian quartz mined.

The quartz deposits of Brazil appear to form a mineral province, lying in four elongated, roughly parallel belts characterized by deep weathering. Most of the deposits in place are quartz veins or flat-lying blanket deposits of milky quartz, or a combination of the two types, in Paleozoic sedimentary rocks that are not highly folded. The clear crystal from which electronic-grade crystal is selected usually occurs in vugs or pockets, apparently of later origin than the milky quartz, which is the only significant accessory mineral. Very few single deposits of the large number worked produced more than 500 pounds of usable crystal a month during the World War II period of activity (*9, 11*).

Present data indicate that most of the typical characteristics of the Brazilian deposits are generally applicable to quartz-crystal deposits in place elsewhere in the world, except those in Australia, where usable crystals have been recovered from a deposit worked at times for molybdenum, tungsten, bismuth, gold, and silver.

In the United States an intensive search for electronic-grade quartz crystal during World War II failed to develop important sources of supply. A small production of usable crystal was obtained from Arkansas, California, western North Carolina, and southwestern Virginia. Most of the Arkansas deposits are cavity fillings associated with veins in Paleozoic sandstones, shales, and cherts exposed along the central belt of the Ouachita Mountains. In southwestern Virginia and western North Carolina most of the crystal was found in the weathered overburden of the Piedmont crystalline rocks, though some was recovered from veins. In California, buried placers in the Sierras yielded small quantities of crystals (*5*).

RESERVES

According to Johnston and Butler (*9*):

> The potential reserves of quartz in Brazil appear to be very great, despite the unprecedented volume of production during and immediately preceding the war. The actual (or visual) reserves of quartz crystal are low. The nature of the deposits is not conducive to measurement of reserves * * *.

Information available in 1954 supported the above views and inspired confidence that the World War II quartz production record of Brazil could be equaled or surpassed under sufficient incentive. New discoveries reported by importers have been made in northern Goiaz and Pará.

Known reserves of quartz crystal in the free world outside of Brazil are negligible. Quartz crystal reserves of the U. S. S. R. are believed to be substantial, but those of its satellites are believed negligible.

MINING

The scattered, irregular occurrence of quartz crystals and the need to avoid damage to usable crystals by heavy blasting or rough handling have restricted the application of mechanized mining practices. In Brazil there is little incentive to improve upon the classic methods of hand mining by individuals or small partnership groups with picks, shovels, hammers, moils, and wedges with nothing but the crudest man-powered pumping and hoisting equipment. Usually the mines do not lend themselves to mechanized mining or large-scale operations because the deposits are small, shallow, and scattered over a large area. The ratio of Brazilian mine-clean crystal to the total quantity of excavated material ranges between 1:1,000 and 1:100,000, with an overall average for the best mines of the order of 1:10,000.

During World War II improved mining methods were installed in Minas Gerais and Bahía through United States Government assistance. Scrapers and bulldozers were used to clear overburden and to remove old dumps that could not have been handled economically by hand. Compressors, machine drills, clay diggers, mine cars, buckets, hoists, and pumps were introduced. At least half a dozen deposits that were exploited by mechanized methods had a successful production and financial record for sustained periods. Many others, though financially unprofitable, contributed substantially to the expanded output. In many instances, mechanization was not feasible.

INSPECTION, GRADING, AND CLASSIFICATION

Electronic-grade quartz crystal must be free from certain unallowable defects but not necessarily colorless, because it is possible to use the smoky variety of quartz for oscillators; or, if necessary to permit inspection, the color may be removed by controlled heating to about 400° C.

When a quartz crystal is found in a deposit it is generally covered by clay or other minerals, which are removed to permit visual examination of the crystal. A skilled workman, known as a trimmer, then trims windows off opposite sides of the crystal to permit internal visual inspection.

The grading and classification of electronic- and optical-grade quartz crystal involve sorting according to size ranges, separation of faced and unfaced crystal, and estimation of the percentage of usable crystal by inspection and testing to detect and evaluate imperfections and flaws.

The undesirable defects in both optical- and electronic-grade quartz that are detectable by competent inspection are optical twinning, cracks, inclusions, veils, bubbles, needles, and ghosts or phantoms. An experienced inspector can detect all of these when the crystals are properly illuminated in an oil bath having the same refractive index as quartz. Monochromatic polarized light is used to detect optical twinning, and the other defects are detected with moderate arc-lamp illumination in the same oil bath. It is desirable that suitable facilities and acceptable inspection practices be established as near the primary source of supply of raw quartz as possible, to reduce transportation on material that would be rejected by processors. Electronic-grade quartz also must be relatively free of electrical twinning, since all electrically twinned portions are unusable for piezoelectric applications. Electrical twinning cannot be detected by any known optical means, although inspectors can infer its presence by surface irregularities in the crystal. It is normally detected by etching sawed surfaces with ammonium bifluoride or hydrofluoric acid. This is usually done at an early stage in manufacturing of piezoelectric units, and the electrically twinned portions are eliminated from further handling. Buyers of raw quartz for piezoelectric uses usually depend on visual- and oil-bath grading.

Current National Stockpile Material Purchase Specifications, P–43, September 12, 1951, provide for 12 weight groups, the smallest of which is No. 1, 100–200 grams. Other classification factors are percent of usability, character of defects, and whether faced or unfaced. Altogether, there are 133 different categories into which stockpiled crystals may fit.

Industry accepts, especially during periods of shortage, smaller crystals than are acceptable for stockpile purchases, and much of the industry consumption in 1950–54 was of crystals from 100 to 200 grams in weight, but there was a substantial consumption of quartz crystals from 50 to 100 grams in weight classed as "eye-clean" under illumination of moderate intensity.

MANUFACTURING

The manufacturing objective involving piezoelectric quartz crystals is to obtain as many units as possible per pound (*10*). A large proportion of these include quartz plates 1 millimeter or less in thickness and about one-half to three-quarter inch on a side if square or rectangular or one-half inch in diameter if round. These plates must be properly oriented with reference to the crystallographic axes to obtain optimum frequency-stabilization characteristics.

The crystal is reexamined in the oil bath by the consumer as an additional check. If of acceptable quality, one of several methods can then be used to detect electrical twinning. The standard practice is to cut the crystal and then etch it with hydrofluoric acid or preferably hot ammonium bifluoride. Electrical twinning will be indicated by distinctive etch patterns.

The inspected crystal is then prepared for sawing. The crystallographic axes of the crystal are alined in a diamond saw frame so that the usable portion of the crystal may be cut or sliced along a predetermined plane. To obtain a given performance characteristic, oscillator blanks or wafers must be cut in specific relationship to the crystallographic axes of the crystal. Lacking identifiable natural faces, orientation is determined by various methods based on physical, optical, piezoelectric, or X-ray diffraction phenomena.

About 80 percent of the present production of quartz-crystal units is of the thickness-mode type, in which the effective vibration frequency is a function of the thickness of the plate. Frequency is also controlled by the lateral dimension of the plates or by the lengths of the bars. The process of manufacturing quartz-crystal units described in the following two paragraphs applies specifically to thickness-mode units, but the basic principles are the same for the production of units with other modes.

The accuracy of the orientation during cutting or sawing stages usually is checked by X-ray, so that the plane of cut of the wafer can be controlled to a few minutes of arc. The cutting operation is accomplished with large diamond saws. Following the cutting operation, the wafers of quartz, having proper orientation, are etched so that optical and electrical twinning, with other defects such as flaws or cracks, may be marked out for removal in the following stage. The wafer is then "diced" into "blanks" of the desired dimensions. The "dicing" operation is performed by smaller diamond saws. The quartz blanks then progress through a succession of lapping stages where the thickness of the blank is reduced, maintaining both major surfaces flat and parallel with successively finer grits of abrasive powder until the desired thickness is reached. This lapping is normally accomplished on a group of crystals at one time. Tolerances of less than 0.0001 inch are maintained in lapping operations. The desired frequency, a function of the thickness of the blank, or disk, is obtained either by micrometer measurements or by actually testing the disk in a crystal oscillator circuit. When the desired frequency is reached for the lapping stage of manufacture, the crystals are again etched to a higher frequency to remove such quartz dust and abraded surface as may have been impregnated in the surface of the quartz oscillator plate during the preceding lapping operations.

Final thickness adjustments are made by etching to a thinner thickness than is desired, and the exact final frequency is obtained when metal electrodes for connection to radio circuits are plated on or attached to the crystal faces. The crystal, then properly assembled in a suitable holder, is completed and termed a quartz-crystal unit, which is ready for insertion in an electronic circuit. Care must be taken at all stages of manufacture to insure that the temperature does not reach the beta-quartz inversion temperature of 573°+C., as twinning generally results during inversion back to the alpha form on cooling.

BYPRODUCTS AND COPRODUCTS

A small proportion of the reported electronic-grade quartz-crystal imports is used for prisms, wedges, lenses, and other optical purposes. In general, only crystals of highest quality and in the medium to large size groups are selected for optical purposes. Electrical twinning is unimportant, but freedom from optical twinning, stains, inclusions, and other flaws is essential. Small quantities of fused quartz are known to have been employed in making optical prisms in Germany.

Quartz crystal is cut and polished as a semi-precious gem. Clear stones, free from inclusions and flaws, are required for this purpose, but either type of twinning may be present. No data are available as to the United States consumption of clear quartz for this purpose, but it is believed to be negligible, as most gemologists prefer the colored varieties, such as rose quartz and amethyst.

Fusing-grade quartz, known as lasca, is selected from small crystals and fragments as a byproduct of electronic-grade quartz-crystal mining. Fusing-grade quartz may be of any size and quality, provided that it is free from bubbles or chemical contaminants, either as adherent coatings or as inclusions. Although lasca is used principally to manufacture fused quartz, small quantities are also used for the production of small piezoelectric units. Lasca also is utilized as the nutrient in quartz-crystal synthesis. Almost the entire United States supply of fusing-grade quartz crystal is obtained by imports from Brazil, where the material is obtained from the rejects of electronic- and optical-grade quartz production by hand selection and hand trimming. The recovery of fusing-grade quartz from inspection rejects and factory wastes in piezoelectric unit manufacture in the United States usually is not feasible on account of labor, supervision costs, and the presence of oil or other contaminants.

Byproducts and coproducts are not significant as a source of revenue in the production and processing of electronic-grade quartz crystal.

SUBSTITUTES

Quartz crystal is far superior to any other piezoelectric material for frequency control in electronic circuits, but the critically short supply of World War II intensified the search for substitutes. Synthetic ammonium dihydrogen phosphate (ADP) has been used extensively for and is preferred in sonar work, and synthetic ethylene diamine tartrate (EDT) has been substituted successfully for quartz in telephone filters. However, it is the consensus of electronic specialists that the latter substitution is not completely satisfactory, and the use of quartz in telephone filters has been resumed. In radio and television transmission, quartz is by far the best material for frequency-control oscillators. Other substitutes for specific applications are dipotassium tartrate (DKT), usable for oscillators and filters, and lithium sulfate monohydrate for ultrasonic applications. The requirements for quartz crystal for frequency-control oscillators may be reduced greatly by the introduction of "crystal-saver" circuits in which a few quartz oscillators by the use of their harmonic frequencies, control the frequency of a multiplicity of wave channels.

In ultrasonic generators for industrial applications, quartz crystal does not hold the supremacy it enjoys in frequency-control devices. Ultrasonic sirens activated by compressed air are used widely in agglomerating and precipitating aerosols. Barium and other titanate ceramics show promise of improvement in place of quartz as electrostrictive generators where ultrasonic vibrations must be concentrated in relatively small areas, such as for high-speed dispersion or for inspection. The magnetostrictive generator, which uses a ferromagnetic material that is expanded and contracted by alternating magnetic forces, is limited largely to output frequencies of less than 1,000 kilocycles a second, whereas the electrostrictive generator may have output frequencies exceeding 500,000 kilocycles a second. Magnetostrictive generators are used principally in chemical-process work, for example, to increase the wetting action of solvents and show promise in degassing castings.

SYNTHETIC QUARTZ

The lack of domestic natural quartz crystal and the uncertainty of Brazilian supplies, coupled with the greatly superior frequency-control properties of quartz compared with any other known materials, led to some consideration of quartz synthesis even before 1942. This interest lagged during 1943 and 1944 with the meeting of military production demands by Brazilian quartz and the substitution of organic piezoelectrics (ADP) for certain applications. In early 1945, however, military advisors concluded that a continued high level of natural quartz-crystal supply was endangered by depletion of known Brazilian deposits without finding compensating new discoveries. The immediate effect of this conclusion was to intensify research on water-soluble piezoelectric crystals; but it was not until early 1946, after data on quartz-synthesis experiments by R. Nachen of Germany became available, that quartz-synthesis projects were activated by Signal Corps contracts with various industry and university research groups. In addition to a large number of contracts concerning various phases of quartz synthesis, the Signal Corps contracted for research on the synthesis of tourmaline, nepheline, and various other materials that could conceivably serve as substitutes for natural quartz in frequency-control applications. Annual Frequency Control Conferences also were sponsored at Asbury Park, N. J., for the free interchange of technical information among the various quartz-synthesis groups, oscillator and frequency-control manufacturers and related business, and those of similar technological and scientific interest.

By 1953 the Brush Development Co. and the Bell Telephone Laboratories, Inc., had each developed technically successful processes for hydrothermal synthesis of quartz in autoclaves operated at relatively high temperature and pressure. Both processes were conducted at pressures between 7,500–15,000 p. s. i. and temperatures between 350° and 410° C. in autoclaves of special alloy steel (*1*, *6*, *7*, *12*). In 1954 it was reported that the cost of producing synthetic quartz crystal on a finished unit basis by these processes was too high to compete with natural quartz crystal, in spite of a higher average yield of finished oscillator units per unit volume at least double that obtained from the natural material. There was substantial hope, however, that a successful synthetic-quartz process could be developed to operate at temperatures and pressures at which ordinary nonalloy-steel high-pressure steam-pipe fittings are suitable, thus reducing capital costs.

The relative advantages and disadvantages of synthetic v. natural quartz have not been evaluated fully. Among the advantages of synthetic quartz are that it is fully and accurately faced, it is relatively free of twinning and other irregularities, the handedness (that is, left- or right-handed) can be predetermined, and the optimum size and, to some extent, the optimum crystal habit can be chosen in accordance with the specific blanks to be made. Data are lacking on the stability of synthetic v. natural quartz crystal with age and with variations of temperature and atmospheric pressure in service.

SUPPLY

Only a few thousand pounds of electronic-grade natural quartz crystal have been produced in the United States; therefore imports represent almost the entire supply. Table 2 gives the quantity and value of uncut quartz crystal imported into the United States, 1932–53, as reported by the Bureau of the Census. Over 99 percent of the total imports have been from Brazil. Beginning in 1941 Brazilian Government statistics have been available from which estimates of the quantity and value of fusing-grade quartz imports into the United States can be made and deducted from total imports to give estimated data for electronic- and optical-grade imports.

TABLE 2.—*Imports of uncut quartz crystal, 1932–53, and estimated imports of electronic- and optical-grade quartz crystal, 1941–53, consumption of electronic-grade quartz, and production of piezoelectric units in the United States, 1940–53*

Year	Total imports of uncut quartz crystal [1]		Estimated imports of electronic- and optical-grade quartz crystal [2]			Consumption of electronic-grade quartz [3]	Piezoelectric units [4] [5]	
	Pounds	Value	Pounds	Value	Values per pound	Pounds	Production (number)	Number per pound
1932	374	$86	-----	-----	-----	-----	-----	-----
1933	9, 100	1, 000	-----	-----	-----	-----	-----	-----
1934	8, 219	2, 477	-----	-----	-----	-----	-----	-----
1935	7, 155	1, 722	-----	-----	-----	-----	-----	-----
1936	22, 766	9, 645	(6)	(6)	(6)	(6)	(6)	(6)
1937	32, 821	62, 076	-----	-----	-----	-----	-----	-----
1938	56, 171	144, 968	-----	-----	-----	-----	-----	-----
1939	67, 052	138, 695	-----	-----	-----	-----	-----	-----
1940	126, 521	264, 436	(6)	(6)	(6)	31, 000	(6)	(6)
1941	2, 237, 608	3, 830, 344	1, 674, 900	$3, 779, 700	$2. 26	59, 000	(6)	(6)
1942	2, 612, 106	8, 987, 108	2, 431, 400	8, 969, 000	3. 69	682, 000	6, 888, 000	10. 1
1943	3, 356, 000	11, 409, 803	3, 356, 000	11, 409, 800	3. 40	1, 588, 000	22, 575, 000	14. 2
1944	2, 300, 506	11, 178, 643	2, 118, 500	11, 115, 000	5. 25	1, 858, 000	29, 939, 000	16. 1
1945	1, 329, 798	6, 190, 621	1, 329, 800	6, 190, 600	4. 66	1, 040, 000	18, 918, 000	18. 2
1946	370, 556	2, 376, 598	216, 400	2, 328, 800	10. 76	172, 400	1, 744, 100	10. 1
1947	473, 788	1, 815, 468	264, 800	1, 782, 000	6. 73	68, 100	1, 052, 400	15. 5
1948	1, 238, 820	4, 209, 531	1, 224, 900	4, 205, 500	3. 43	61, 600	1, 225, 400	19. 9
1949	319, 631	1, 462, 018	306, 800	1, 460, 200	4. 76	46, 200	937, 100	20. 3
1950	310, 251	791, 412	241, 200	785, 900	3. 26	114, 300	1, 614, 000	14. 1
1951	1, 287, 398	2, 090, 061	843, 200	2, 045, 600	2. 43	282, 300	3, 290, 000	11. 7
1952	1, 576, 791	2, 885, 437	1, 049, 300	2, 881, 600	2. 75	502, 500	6, 181, 500	12. 3
1953	1, 320, 683	2, 254, 938	[7] 1, 119, 198	[7] 2, 240, 154	2. 00	399, 200	7, 217, 700	18. 1

[1] From U. S. Department of Commerce: Includes electronic-, optical-, ornamental-, and fusing-grade quartz crystal.
[2] Estimated from industry advices and Brazilian Government statistics.
[3] 1940–44, War Production Board.
[4] 1942–44, War Production Board.
[5] Includes oscillators, resonators, and other piezoelectric units.
[6] Data not available.
[7] Imports of Brazilian pebble, valued at $0.35 or more per pound.

The average value per pound, as shown in table 2, approximates the declared value at the export port. There is little correlation between the average declared value of imports into the United States from Brazil and resale prices in the United States because of variations in the distribution of sizes and qualities, fluctuating currency-exchange rates, changing trade practices, and lack of uniformity of grading between the United States and Brazil.

The import statistics from 1932 to 1941 reflect gradual increase in the peacetime demand for quartz oscillator plates for radio frequency control. During this period the grading of quartz crystal in Brazil by exporters depended entirely on visual inspection and varied widely among different exporters. Thus, much of the pre-World War II imports of uncut crystal into the United States was unsuitable for either optical or piezoelectric uses. Just before the United States entered the war, the polarizing oil tank for determining optical twinning and the arc-lamp beam for inspecting for inclusions were introduced extensively for inspection of all electronic-grade quartz. Progressive adoption of these devices and of other improved grading techniques by exporters soon raised the proportion of imported crystal usable for piezoelectric purposes.

In midsummer 1940 the Bureau of Mines made a survey showing that stocks of radio-grade quartz crystal in private hands were adequate for immediate needs but that they should be supplemented by a Government stockpile that had been accumulating gradually in consequence of purchases by the Procurement Division of the United States Treasury Department.

In 1941 the Governments of the United States and Great Britain agreed to buy all stocks of Brazilian crystals after Brazilian nationals had satisfied the needs of private industry. There was a sharp increase in the quantity and value of United States imports in 1941 compared with 1940. Selection methods still were unorganized, however, and a high proportion of the 1941 imports was rejected on further inspection.

In 1942 a more thorough, systematic check of material before export was begun in Brazil. American engineers were sent to advise regarding production and selection methods in various Brazilian mining areas, and inspection laboratories were set up in Rio de Janeiro.

On May 18, 1942, the War Production Board, under Order M–146, assumed control over the products for which quartz crystals might be used; and on August 4, 1942, under General Imports Order M–63, importers were required to obtain War Production Board authorization for disposal of quartz crystals in this country. On March 20, 1944, Order M–146 was amended to increase the number of permitted uses of quartz crystal, and the amended order was continued in force until August 20, 1945, when it was revoked.

In 1943 United States imports of uncut quartz crystals reached the alltime high of 3,356,000 pounds; a large proportion was of electronic grade. In addition, 5,968 pounds was produced in the United States. In 1943 most of the high-priority crystals imported into the United States were airborne, and only low-grade material was shipped by sea.

Although United States imports of electronic-grade quartz crystal declined in 1944 to about 2,118,500 pounds, the percentage of usable electronic-grade material increased compared with 1943; hence, the available supply of strategic material derived from imports was only about 10 percent less than in 1943, and lower losses in fabrication into piezoelectric units more than compensated for the decline of supply. The domestic production was 3,934 pounds valued at $23,769. Because attempts at domestic production were relatively unsuccessful, the Metals Reserve Company ceased purchasing domestic material at the end of 1944.

The end of the period of expanded quartz-crystal production was reached in 1945. Following cessation of hostilities, much of the remaining stock of crystal in processers' hands was turned over to the contracting Government agencies, which in turn assigned it to the War Assets Administration as surplus material. Metals Reserve Company stocks also were acquired by the War Assets Administration as surplus. The bulk of the electronic-grade material was transferred to the National Stockpile under Public Law 520, 79th Congress. Additional purchases for the stockpile under Public Law 520 were made by Emergency Procurement Service, General Services Administration. EPS purchases were made under authorization of the Munitions Board up to June 12, 1953, after which the stockpiling responsibility was transferred to the Office of Defense Mobilization, which continued the delegation of procurement functions to EPS. In June 1951 EPS was also designated as the central procurement agency for raw-quartz requirements for the Army, Navy, and Air Force.

CONSUMPTION

United States consumption of electronic-grade quartz crystal, 1940–53, is given in table 2. During the 4 World War II years, 1942–45, the peak consumption was 1,858,000 pounds in 1944 and averaged 1,292,000 pounds a year. This was about 3 to 4 times greater than the defense-inflated peak civilian level reached in 1940 or during the postwar (1947–49), period, which is an approximate measure of current peacetime demand. In 1953 consumption totaled 399,200 pounds. About 40 percent of the 1953 consumption of raw quartz crystal was in Pennsylvania, although the largest number of companies was in California (table 1).

Consumption statistics do not permit close correlation with the supply of raw material. The excess of imports over consumption (attributable to material rejected after inspection, to stocks, and to stockpile purchases) fluctuates broadly from year to year. Furthermore, there has been a progressive improvement in the selection and grading of uncut crystal at the source, as well as a decrease in the preparation losses in converting the crystal into piezoelectric units. In opposition to these factors, there has been a trend toward more rigid specifications for piezoelectric units for military consumption which tends to decrease the number of oscillator units recovered per pound of raw crystal. This is partly, if not wholly, offset by expanding industrial use of crystal that does not meet military (stockpile) specifications.

Analysis of the factors that influenced consumption trends in 1950–54 shows that, in general, the industrial base of quartz-crystal demand expanded because of new uses for which quartz crystal, if not the only acceptable material, was at least the preferred material. A substantial increase in industrial demand began in 1947 and extended into 1953 because of Federal Communications Commission regulations requiring closer frequency control for all new communications transmitters installed after January 1, 1950 and for conversion of all older units by January 1, 1953. The changeover of older equipment accounted for an indeterminate, extra, nonrecurrent demand for quartz oscillator units.

There is no fixed relationship between the consumption of electronic-grade crystals and the number of piezoelectric units produced. During the war years improved cutting efficiency, the recovery of imperfect plates, and the use of smaller sizes are credited with increasing the number of plates per pound from 10 in 1942 to 18 in 1945. Subsequent statistics do not reveal a consistent relationship, possibly due mainly to postwar readjustments of stocks. Design changes and new applications of piezoelectrics will continually change the consumption pattern, though an overall trend toward smaller units may result not only in a larger number of units per pound of crystal but also in expanded utilization of smaller sizes of uncut crystal.

A partial breakdown of the domestic production of piezoelectric devices, by uses, in 1942–44 is shown in table 3.

TABLE 3.—*United States production of piezoelectric units by uses, 1942–44* [1]

Use	1942	1943	1944
Radio oscillators:			
Small (½–1 in.)	5, 202, 808	14, 011, 577	(2)
Large (1 in. and larger)	1, 576, 782	8, 474, 523	(2)
Total	6, 779, 590	22, 486, 100	29, 764, 185
Telephone resonators	91, 173	64, 597	153, 058
Other	17, 050	23, 255	21, 506
Total production	6, 887, 813	22, 573, 952	29, 958, 749

[1] War Production Board.
[2] Not available.

PRICES

Electronic-grade quartz crystal is priced in accordance with weight, percentage of usability, type and distribution of flaws, and whether faced or unfaced. The Brazilian Government maintains a schedule of prices, known as "Tabela," for exports, based on declared minimum values at Rio de Janeiro, Brazil (table 4). The average for 1953 was $2 a pound compared with $2.75 a pound in 1952, due to an increased proportion of smaller crystals and lower quality. Resale prices in the United States are negotiated between buyer and importer and bear no fixed relation either to the Tabela price schedule or to the actual declared value. Brazilian quality classes differ from United States practice, and American inspection techniques are more efficient. Re-sorting of crystal receipts usually results in two or more lots of more uniform quality than the original lot. The best crystals may bring prices up to three times the Tabela schedule; others may have to be sold at a loss. For example, selected 201- to 300-gram, commercial class 1 crystal in 1953 sold for about $12 a pound in the United States compared

TABLE 4.—*Minimum allowable declared value of quartz-crystal exports from Brazil in 1952* [1]

[Dollars per pound] [2]

Weight group (grams)	Quality class [3]		
	1	2	3
200 or less			1. 25
201–300	4. 50	2. 50	1. 25
301–500	7. 50	3. 75	1. 88
501–700	10. 00	5. 00	2. 50
701–1,000	15. 00	6. 25	3. 12
1,001–2,000	20. 50	7. 50	3. 75
2,001–3,000	24. 75	8. 75	4. 38
3,001–4,000	28. 75	10. 00	5. 00
4,001–5,000	32. 50	11. 25	5. 62

[1] Established by the National Department of Mineral Resources of the Brazilian Ministry of Agriculture.
[2] 1 cruziero per kilogram—$0.025 per pound (approx.)
[3] The first 2 classes cover first-quality crystals exceeding 200 grams in weight; crystals in class 1 have over 60 percent usability; and crystals in class 2 have 30 to 60 percent usability. Class 3 includes crystals up to 200 grams in weight with over 60 percent usability; and crystals weighing over 200 grams with a minimum of 30 percent usability.

with the Tabela schedule of $4.50 a pound, and 301- to 500-gram class 1 crystals brought about $17 per pound versus the Tabela schedule of $7.50. Crystals in the weight group of 201 to 300 grams, class 3, were sold in the United States for $2 to $3 a pound compared with the Tabela value of $1.25.

Up to 1954 electronic-grade quartz crystal has entered the United States duty free. A very small quantity has been processed in Brazil, but the industry has not shown indications of becoming a significant processor, so that a tariff on manufactured quartz elements has not received serious consideration.

OUTLOOK

The outlook for electronic-grade quartz crystal would be toward decreasing reliance upon natural crystal if improved production techniques and expanded production at lower cost of synthetic crystal are achieved. Stabilization of the Brazilian natural-quartz-crystal industry or establishment of a synthetic-quartz-crystal industry on a substantial scale at comparable prices would favor adoption by industry of quartz crystal as the preferred material for many uses now inhibited by the uncertainties of the quartz crystal supply. The extended use of "crystal-saver" circuits also would narrow the gap between emergency and peacetime requirements, but new uses in such fields as ultrasonics (18,000 to 100,000 cycles) and supersonics (over 100,000 cycles) may create requirements far in excess of present levels. Hence, the peacetime consumption pattern may be large enough so that the industry can meet the peak load of a military mobilization without serious disruption or diversion of critical manpower and materials for a major expansion of production and fabrication facilities.

PROBLEMS

The problems of the quartz-crystal industry are concerned chiefly with assurance of adequate supplies of raw material and, of more importance, at least partial alleviation of the feast-or-famine volume of business that has beset the industry from its inception.

Supplies of natural crystal to the United States in peacetime seem assured. The problems of supply in a full mobilization and the optimum size of the National Stockpile involve evaluation of the following factors: Military defensibility of Brazil; transportation of crystal from Brazilian mines to the United States; continued political amity with Brazil; expandability of Brazilian production to meet mobilization requirements; the time, United States technical assistance, and United States equipment required to achieve the desired production level in Brazil; and finally, estimates of the time, skilled labor, and critical metals needed to expand synthetic-crystal production to close the gap, if any, between requirements and stockpiled material plus wartime supply.

Another aspect of the supply problem involves continued research and pilot-plant production of synthetic quartz crystal which are needed to develop improved processes that can compete in cost with natural crystal. At least limited research is also indicated on the synthesis of alternative materials.

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RADIUM

By

John E. Crawford[1]

RADIUM, which before utilization of nuclear fission enjoyed an enviable and unique position, now must endure competition from the man-made radioisotopes. The material's one distinct advantage over the artificial radioisotopes is its long half life.

Summary

The first pure salt of radium was prepared by Marie Curie in 1902, and in 1910 Mme. Curie and A. Debierne isolated the metal.

Radium is a radioactive element with a half life of 1,600 years. There is a constant ratio between radium and uranium—1 part radium in 3,400,000 parts uranium in all sufficiently old minerals in which radioactive equilibrium has not been disturbed by weathering or leaching.

Radium was discovered in the mineral pitchblende, an oxide of uranium. The most extensive pitchblende deposits have been found at Katanga in the Belgian Congo and at Great Bear Lake in Canada. Second in importance as a source of radium is carnotite, a double vanadate of uranium and potassium, found principally as a yellowish impregnation of sandstone in the western United States.

Since 1943, when uranium became important as a source of fissionable uranium-235 and plutonium, production of radium-bearing material has substantially increased as a byproduct of the treatment of uraniferous ores. Most of the radium used for civilian requirements in the United States is imported, usually as radium salts.

All uses of radium are based on its radioactivity. The most important of these have been in medicine, principally to treat malignant growths. For about 20 years radium has been employed for industrial radiography purposes. Radium is mixed, in very small concentrations, with zinc sulfide to make luminescent paints for watch, clock, and meter dials and signs. An intimate mixture of radium with beryllium is a moderately intense source of neutrons; and a radium foil is used in certain types of static elimination equipment.

Radium supplies are ample to satisfy known requirements, and radium-production capacity probably exceeds the current demand for the metal and its compounds by a significant quantity. New applications for the metal and its compounds should be considered.

Economical radioisotopes are competing with radium in many fields of use; although radioisotopes do not have the long half life of radium, they are usually cheaper and provide a greater variety of radiation potentials.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

GENERAL REVIEW

Radium (*9, 11, 12*)[2] is one of the heavier of the natural-occurring elements. It is a member of the alkaline-earth group, with an atomic number of 88 and an atomic weight of 226.05. The radioactive element was discovered by three French scientists, Pierre Curie, Marie Curie, and G. Bemont, in 1898 at the Sorbonne in Paris. By chemically separating the constituents of the radioactive mineral pitchblende, two highly radioactive, unique elements were determined. The elements were named polonium and radium. Four years later, in 1902, Mme. Curie isolated the pure salt of radium, and in 1910 Mme. Curie and A. Debierne produced radium metal.

Of all the radioactive isotopes of radium, the only radioisotope stable enough to be produced in recognizable amounts is radium–226. The half life of radium–226 is 1,600 years, and its disintegration is caused largely by the emission of alpha particles. The element is a brilliant white metal (specific gravity of nearly 6.0); it melts at about 700° C. and volatilizes appreciably at that temperature.

OCCURRENCE

Radium's (*9, 12*) 1,600-year half life is short compared with the earth's geological time scale—a yardstick measured in hundreds of millions of years; however, radium will continually be generated by the disintegration of uranium, with which it is intimately associated in nature. As already stated where radioactive equilibrium has not been altered by weathering or leaching, there is a nearly constant ratio between uranium and radium—1 part radium to 3,400,000 parts of uranium—in all rocks of sufficient age. There are minute quantities of radium in springs, rivers, and sea water, as well as in the various layers of the ocean sediments. In the earth's crust the average content of radium including both igneous and sedimentary rock, is about 1 part per million million. The richest uranium ores contain less than one-quarter gram of radium per ton, and some uranium deposits containing as little as 1 gram of radium in 300 tons have been worked for radium.

The famous pitchblende deposits at Katanga in the Belgian Congo and at Great Bear Lake in Canada have been outstanding sources of radium for many years. The carnotite occurrences of the Colorado Plateau area of the western United States also have been a commercial source of radium. Pitchblende is a rich uranium oxide mineral and carnotite a double vanadate of uranium and potassium. Radium-bearing uranium deposits have been found in other localities, including Australia, England, Germany, Madagascar, Portugal, Russia, and South Africa. The increased importance of uranium as a source of nuclear energy has resulted in intensive prospecting and the discovery of many more radium-bearing minerals.

METALLURGY

The general method Mme. Curie developed for recoverying radium is still used by the industry today, with minor modifications. The outline of a radium-recovery process follows:

The milled radium-bearing raw material is digested in nitric-sulfuric acid mixture. Barium is added to act as a carrier for the radium. The uranium and certain other constituents contained in the ore dissolve, but the insoluble radium, barium, and lead sulfates and a siliceous gangue are left as residues. After digestion of the ore the residues are filtered out and boiled with sodium hydroxide to remove some of the lead sulfate and silica. The radium-bearing residues are next autoclaved with a sodium carbonate solution; as a result, the barium and radium sulfates are converted to carbonates. The barium and radium carbonates are dissolved in hydrochloric acid and the remaining silica filtered off, after which the barium and radium are once more precipitated as sulfates and converted to carbonates, for the purpose of purification. Hydrobromic acid is added to dissolve the purified carbonates and transform them into bromides. The following step constitutes an involved fractional crystallization process, the success of which depends upon the fact that radium bromide is less soluble than barium bromide. Crystals formed are enriched in radium. After 10 recrystallizations there are a few parts of radium per thousand parts of barium as compared with a few parts of radium per million parts of barium in the original crystallization. Next the bromides from several batches are consolidated and repurified, and further crystallizations made until radium salts of the desired purity are obtained. The purity of the salts may range from 95 to 99 percent

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[2] Italicized figures in parentheses refer to items in the bibliography at he end of this chapter.

radium. They are usually sealed in small tubes to be measured and sold (*9*).

Certain types of ore may require modifications of the process described above.

The pitchblende of the Great Bear Lake deposit in Canada contains native silver which has been recovered at the refinery by (1) roasting the ore with sodium chloride to convert the native silver to silver chloride, (2) digesting the ore with sulfuric acid, (3) adding hydrochloric acid to precipitate the silver and barium chloride to remove the radium, (4) extracting the silver from the radium-bearing residues with sodium hyposulfite, and (5) precipitating the silver with sodium sulfide. The radium is then removed from the residues in a fashion similar to that described above (*8*).

Probably the most economical method for recovering radium from carnotite ores of the Colorado Plateau area of the United States was the modified nitric acid process. The ore is leached with nitric acid and the solution partly neutralized with caustic soda, after which barium chloride and sulfuric acid are added (*4*). Subsequent steps for removing the radium from the residues are about the same as those indicated under the general method. The carnotite deposits, however, are now worked primarily for their uranium content, and the radium is left in tailings from which it is doubtful that much is being recovered.

A recently developed method for separating radium and barium employs synthetic organic cation exchange.

Another procedure, perfected in 1953, recovers radium from a prepared radium-barium chromate concentrate and is known as the precipitation from homogeneous solution method (*5*).

PRODUCTION AND CONSUMPTION

The total world production of radium, in grams, was approximately 50 by 1916, 300 by 1925, and 1,000 by 1940. Production was greatly accelerated during world War II (*9*). Total world stocks of radium in 1952 were estimated at about 2,000 grams.

The important source of radium for some years after its discovery was the Joachimsthal mines in Bohemia, now a province of western Czechoslovakia. Some radium was produced in other European countries in following years. The United States was the first major producer from 1911 to 1923. Carnotite ores from the western United States were processed to recover radium, principally at Denver, Colo., and Pittsburgh, Pa. The rich Belgian Congo pitchblende deposits at Katanga were discovered in 1913. A refinery at Oolen, Belgium, began producing radium from imported Congo pitchblende in 1922 and immediately dominated the world market. The Canadian Great Bear Lake pitchblende deposits were discovered in 1930; and the refinery at Port Hope, Ontario, started producing radium in 1933. From about 1936 through 1940 the world market was divided between the Belgian and Canadian producers on a 60:40 basis by a cartel agreement. During World War II the African ores were shipped to the United States and Canada for refining (*2*, *12*). After the war ended, Belgium once more became the leading producer of radium.

Since 1943, when uranium became important as a source of fissionable uranium–235 and plutonium, production of radium-bearing material has substantially increased as a byproduct of the treatment of uraniferous ores. Whether or not the increase in the supply of radium thereby made available finds application in the atomic energy programs of the countries involved has not been disclosed.

A relatively small quantity of the new radium (exclusive of that reclaimed) purchased annually by consumers in the United States is derived from domestic ores—probably no more than a few percent of the total consumption. The majority of the radium used for civilian requirements in the United States is imported, usually as radium salts; a record of such imports from United States Department of Commerce statistics for 1947 to 1954 is presented in table 1. A large part of the imports in recent years consists of radium stocks produced in Belgium and transferred to the United States for safekeeping rather than to satisfy any increase in domestic demand for the metal.

TABLE 1.—*Radium salts imported for consumption in the United States, 1947–54*

Year	Quantity (grams)	Total value	Average value per gram
1947	76. 681	$1, 505, 814	$19, 600
1948	77. 018	1, 385, 337	17, 900
1949	98. 032	1, 719, 656	17, 500
1950	80. 969	1, 245, 511	15, 300
1951	85. 805	1, 225, 564	13, 600
1952	173. 711	2, 873, 688	16, 500
1953	96. 750	1, 662, 106	17, 200
1954 [1]	56. 801	851, 141	14, 985

[1] 1954 estimated on the basis of 11 months.

Consumption of radium in the United States was estimated in 1954 at 25 to 30 grams a year, with about 200 additional grams on lease.

The principal suppliers of radium and radium compounds for civilian requirements in the United States are the Radium Chemical Co. (sales representative of Union Minière du Haut Katanga), New York, N. Y.; U. S. Radium Corp., New York, N. Y.; and Canadian Radium & Uranium Corp., New York, N. Y. (sole domestic producer) with a refinery at Mount Kisco, N. Y. (*2*).

PRICES

For some time after its discovery, radium was exchanged freely among scientists for research purposes. Its useful therapeutic applications, however, soon made the scarce radioactive material a much-sought-after commercial item, and it became costly. In post-World War II years new sources of radium-bearing uranium were developed and the demand for radium decreased, with the result that the price of radium declined steadily during the period.

The price of radium per milligram rose steadily to $135 during World War I, then began to decline, and reached a level of $70 in 1923, when radium from the Belgian Congo became available; this influx of cheaper radium caused production in the United States from carnotite to cease abruptly. Soon after 1930, the collapse of world buying power resulted in a drop to $50 a milligram, which continued as the nominal quotation until early in 1936, when it was lowered to $40. Shortly thereafter, the two outstanding producing companies—the Union Minière du Haut Katanga (operating in the Belgian Congo) and Eldorado Gold Mines, Ltd. (Canada)—agreed to share the world radium market because their combined output potential far exceeded demand for the metal; under this cartel action the price of radium was maintained at $40 until 1940. The cartel dissolved about 1940, and from 1940 through 1950 the price ranged from $25 to $30. Beginning with 1951, radium became available at $16 to $21.50 as stocks increased (*2*). Throughout 1954 E&MJ Metal and Mineral Markets quoted the price of radium at $16 to $21.50 per milligram of the element in purified salts, depending on quantity.

The price of radium is quoted in terms of the actual weight of the element in a purified radium salt. Radium preparations of all kinds are usually sold on the basis of Government certification as to radium content as determined by gamma-ray measurements. In the United States this work is carried out by the National Bureau of Standards, Washington, D. C.

USES

All uses of radium result from its radiations. The most important of these have been in medicine, principally in treating cancer. For this purpose the element may be used directly, sealed in tubes and needles; or its gaseous decay product, radon, may be compressed into a small capsule, in which form it may be used until the radon itself has decayed to a low intensity (*11*).

Radium is employed for industrial radiography, particularly in inspecting metal castings (*3*, *4*, *9*) and is mixed in very small concentrations with zinc sulfide to make a luminescent paint for watch, clock, and meter dials and signs that must be read in the dark. An intimate mixture of radium with beryllium is a moderately intense source of neutrons. Such neutron sources have been used extensively in scientific work and have found practical applications in searching for oil deposits (*2*).

There is some use of radium foil as an ionization agent in static elimination equipment for industrial grinding operations and in processing metal powders (*1*, *14*).

For most uses of radium, substitutes have become available in recent years. Cobalt–60 and other radioisotope products of nuclear reactors, as well as high-voltage X-ray generators are competing with radium in radiotherapy and radiography. Fluorescent paints activated by ultraviolet radiation have, in some instances, replaced radium paints in airplane-panel meters. Cyclotrons and nuclear chain reactors provide much more intense sources of neutrons.

OUTLOOK

Supplies of radium appear ample to satisfy both present and future known requirements. Radium production capacity exceeds and will probably continue to exceed the current demand for the metal and its compounds by a significant amount.

PROBLEMS

Economical radioactive isotopes are competing with radium in many fields of use; they are cheaper and provide a greater variety of radiation potentials. The half-life of most of the useful radioactive isotopes is short compared to that of radium, and sometimes a greater quantity of radioisotopes is required than radium to produce a desired effect. Nevertheless, the competitive position of the radioisotopes is firmly established, and radium—which enjoyed unique fields of use before World War II—now must live with its new neighbors from the nuclear reactors.

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RARE-EARTH METALS

By

Frank D. Lamb[1]

RARE-EARTH metals, a family of 15 elements, excite the interest of research metallurgists as a fascinating field for investigation, both as to their properties and their applications in industry. Only within recent years has it been possible to isolate the rarer of these elements in sufficient quantities to permit undertaking such a research program; even at present, the production methods for some of them are not applicable to large-scale operations.

Summary

The rare-earths are a closely related family of 15 metals (lanthanum, cerium, praseodymium, neodymium, promethium, samarium, europium, gadolinium, terbium, dysprosium, holmium, erbium, thulium, ytterbium, and lutetium). The principal commercial source of these metals has been the mineral monazite, a rare-earth phosphate containing thorium; but the discovery in 1949 in California of a very large deposit of bastnaesite—a fluocarbonate of the rare earths—added another important source to the United States supply (*15*).[2] The rare-earth silicate minerals, cerite and allanite, have contributed very little to domestic production and are not considered to be commercial sources. Over 200 other minerals are known to contain appreciable quantities of rare-earths, and it may be expected that some production will come from such minerals as euxenite, pyrochlore, samarskite, fergusonite, and gadolinite in the future.

There are large reserves of monazite in India, Brazil, and South Africa, and the world need for rare-earth metals could be supplied from these sources for many years; however, because of the thorium content of monazite and its potential use as a source of atomic energy, the Governments of India and Brazil have placed restrictions on monazite exports. Known placer deposits of monazite in the United States are numerous; but most of them are limited in size, and few can be mined for monazite alone.

The deposit of bastnaesite in California is extensive; in consequence, it is capable of supplying all of the foreseeable raw-material requirements of the United States for the production of rare-earth metals and compounds. Future exploitation of this deposit and similar deposits in New Mexico may depend to a large degree upon the availability of rare-earths as coproducts in the production of thorium from monazite. If the demand for thorium for atomic energy increases substantially, a surplus of coproduct rare-earths will result unless their consumption in new uses increases accordingly.

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[1] Commodity-industry analyst, Bureau of Mines.

[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

Rare-earth metals and compounds are consumed principally for the following purposes: Glass industry, 30 percent; carbon-arc electrode cores, 25 percent; misch metal and ferrocerium, 20 percent; and miscellaneous, including additives in the iron and steel industry, 25 percent. Misch metal, the mixture of all the rare-earth elements in metallic form, contains 45 to 55 percent cerium and lower percentages of the other rare-earth metals. The mixture is used in producing lighter flints and to some extent in producing alloy steels and magnesium alloys. Recently the use of rare-earth compounds derived from California bastnaesite has promised to create better alloy steels and to increase the yield of certain types of low-carbon steels that are difficult to work.

The future of the rare-earth industry will be strongly influenced by the results of the research activities conducted both by industry and Government during the next few years. Important problems regarding methods of extraction and isolation of the individual metals, determination of the properties of the less common elements, and development of new uses for the group as a whole will have to be worked out if the industry is to expand appreciably.

BACKGROUND

HISTORY

In 1794 the Finnish chemist, Johann Gadolin, found a new metallic oxide in a mineral that Arrhenius had discovered at Ytterby, Sweden, 6 years before. In 1797 Ekeberg confirmed the discovery and suggested that the oxide be named Yttria for the village in which it had been found and that the mineral be called gadolinite in honor of the chemist. Both Gadolin and Ekeberg believed that the new oxide was that of a single metal; they did not know that it was a mixture of 15 or more different elements.

The names of the elements of the rare-earth group have been derived from a variety of sources. Yttrium, erbium, terbium, and ytterbium are named after Ytterby, the Swedish village from which gadolinite was first obtained. Europium is a geographical name. Holmium was derived from Stockholm; lutetium from Lutetia, an ancient name for Paris; and thulium from Thule, an old name for Scandinavia. Samarium received its name from samarskite, named in honor of a Russian mine officer, Samarski; and cerium was named for the asteroid Ceres. Didymium means the twin element and was formerly supposed to be a single element. When didymium was found to consist of 2 elements, 1 was called neodidymium, the new twin, later shortened to neodymium; the other, praseodymium, signifying a green twin. Lanthanum is derived from the Greek verb meaning to lie hidden, and dysprosium means difficult of access.

From 1893 to 1910 the United States was one of the principal producers of monazite concentrates, although it ran a poor second to Brazil. The production—about 300 tons per year—came from North Carolina and South Carolina. The total domestic output from 1910–44, was only about 50 tons. Since 1948 monazite has been recovered in Florida as a coproduct with zircon, ilmenite, and rutile. During 1948 Rare Earths, Inc., of McCall, Idaho, produced a small tonnage from black-sand concentrates, previously removed from gold-bearing gravels of west-central Idaho. The production of monazite from dredges in Idaho accounted for the major domestic supply of the mineral in 1953. Bastnaesite production was begun in May 1952 by the Molybdenum Corp. of America from its property near Mountain Pass, Calif.

GEOGRAPHIC DISTRIBUTION OF KNOWN DEPOSITS

Large productive deposits of monazite occur in the States of Travancore, India; Rio de Janeiro, Espirito Santo, and Bahía in Brazil; and the van Rhynsdorp district of Cape Province, Union of South Africa. Smaller but significant deposits are known in Australia, Ceylon, United States of Indonesia, Korea, and the Malayan Union. In the United States, large reserves of monazite are present in the placer gravels of central Idaho, along the flanks of the Idaho batholith. Other deposits are known in Montana, Florida, North Carolina, South Carolina, and California. Deposits of commercial quantities of bastnaesite occur in the Belgian Congo (Ruanda-Urundi), California, and New Mexico. Small production of allanite and cerite has been reported from Sweden and the United States.

The possibility of eventually using thorium as a source of atomic energy prompted India, in 1946, to restrict exports of monazite concentrates. The restrictions were still in effect during 1954, effectively cutting off one of the most important raw-material sources for the United States industry. Brazil increased its production and from 1946 until about August 1950 was virtually the sole source of United States imports. Brazil suspended issuance of export licenses for monazite on January 1, 1951. The Governments of India and Brazil were receptive to offers from foreign commercial firms who would agree to construct and operate monazite-processing plants within the confines of the monazite-producing country. Under this type of agreement, the processor markets the rare-earth content, leaving the thorium for stockpiling in the country of origin. A French firm, the Société de Produits Chimiques des Terres Rares, now operates a monazite-processing plant of this type in India. Another French company, probably affiliated with the first, has discussed the possibility of a similar arrangement with the Government of Brazil. Exportation of monazite from Australia to nonempire destinations was forbidden in 1944. A few tons of monazite were produced in 1954 from a deposit in Ceylon and a similar quantity is produced annually from sands of the Nile delta in Egypt.

PRINCIPAL SOURCES OF DOMESTIC PRODUCTION

California.—The Molybdenum Corp. of America owns most of the claims on the massive barite-carbonate-bastnaesite lode in the Mountain Pass area of San Bernardino County, Calif. This deposit of bastnaesite, a fluocarbonate of the rare-earth metals, was discovered in April 1949. The main ore body is about 2,500 feet long and 500 feet wide. A tremendous tonnage is available, with a grade of about 10 percent total rare-earth oxides. It is mined by open-pit methods at relatively low cost and milled in a flotation and roasting plant capable of treating 5,000 tons monthly. The intimate association of the barite and bastnaesite, combined with the tendency of the ore to slime upon grinding has contributed to recovery difficulties. Present practice involves flotation of the bastnaesite from the barite in a pulp which is kept at a boiling temperature. Concentrates assaying 65 to 70 percent rare-earth oxides are produced with a recovery of about 70 percent of the total rare-earth oxides.

Florida.—Monazite occurs with other heavy minerals (mainly ilmenite, rutile, and zircon) in the sands of Florida beaches and dune areas. These sands occur along both coasts and in numerous inland areas of Florida, but only where they are locally concentrated do they have economic importance. Deposits near Jacksonville, Starke, and Melbourne, Fla. were mined in 1954 and produced modest quantities of monazite as a coproduct with titanium-bearing and other heavy minerals.

Idaho.—The earliest attempt to produce monazite in Idaho was in 1909, when a small plant was built near Centerville in the Boise Basin. The plant was abandoned after a few months, as it could not compete with low-cost monazite from India. Between 1946 and 1948 the gold-dredge operators in the Boise Basin saved some of the monazite-bearing black sands but made no attempt to separate the rare-earth minerals.

In 1949 the Bureau of Mines and others began a series of churn-drill investigations to ascertain the monazite tenor of the alluvial deposits along the flanks of the granitic mass called the Idaho batholith (*5*). The first work was conducted in areas where monazite was known to occur, such as Boise Basin. Preliminary studies indicated that higher concentrations of monazite existed in the alluvials in areas not previously reported to contain the mineral.

The Big Creek deposit near Cascade, Idaho, was among the many alluvial deposits examined. A gold dredge, with minor flowsheet alterations for saving the black sands and monazite, was placed on the Big Creek deposit in December 1950. This dredge, owned by Baumhoff-Marshall, Inc., began mining the alluvials early in January 1951. Monazite is the sole source of revenue, as only a trace of gold is present in the gravels, and the other black sands have no value in Idaho. Late in 1951 two more converted gold dredges were moved to the Big Creek area. In 1953 the three dredges produced more than half of the domestic needs for monazite.

Late in 1954 a dredge was under construction in Bear Valley, Idaho, for mining a placer deposit containing monazite and euxenite, a niobate and titanate of rare-earth elements and uranium. The presence of uranium and niobium minerals in the Bear Valley placer promises to make this an economic source of all these useful metals.

South Carolina.—Near Aiken, S. C., a placer deposit was being prepared for dredging late in 1954 to produce monazite, rutile, ilmenite, and zircon. Located on the Atlantic Coastal Plain of South Carolina, below the Piedmont Plateau, this deposit is typical in composition but larger than most of the many similar deposits known to exist in the North Carolina–Georgia area. Early development of other Coastal Plain deposits depends to a large degree upon the success of this operation.

INDUSTRY

The following firms are the principal processors of monazite and bastnaesite:

Domestic

Lindsay Chemical Co., West Chicago, Ill.
Molybdenum Corp. of America, New York, N. Y.
Maywood Chemical Works, Maywood, N. J.
Rare Earths, Inc., Pompton Plains, N. J.

Foreign

Thorium, Ltd., London, England.
Société de Produits Chimiques des Terres Rares, Paris, France.
Treibacher Chemische Werke, Treibach, Austria.
Oriquima do Brazil, São Paulo, Brazil.
Indian Rare Earths, Ltd., Alwaye, Travancore, India.

The following firms are producers of misch metal and ferrocerium:

Domestic

Cerium Metals Corp., New York, N. Y.
New Process Metals Corp., Newark, N. J., (subsidiary of Ronson Lighter Co.).
General Cerium Corp., Edgewater, N. J.
Mallinckrodt Chemical Works, St. Louis, Mo.
American Metallurgical Products Co., Pittsburgh, Pa.

Foreign

Shawinigan Chemicals, Ltd., Shawinigan Falls, Quebec, Canada.
Belgo Canadian Mfg. Co., Ltd., Montreal, Canada.
Imperial Chemical Industries, Ltd., Liverpool, England.
Sibor Hard Metals, Ltd., London, England.
Chemo Metals, Ltd., London, England.
British Flint and Cerium Mfg., Ltd., Tonbridge, Kent, England.
Société de Produits Chimiques des Terres Rares, Paris, France.
Treibacher Chemische Werke, Treibach, Austria.
Promethus, Kempten, Germany.
Société Anonyme de Pont-Brule, Brussels, Belgium.
Monacerium, Ltd., Lisbon, Portugal.

TECHNOLOGY

The term "rare-earth metals" is used to designate a group of elements which are closely related chemically. They were originally called "earths" because their oxides resemble those of the alkaline earths and "rare" because they were thought to be scarce. Now many of the elements are known to exist in considerable quantities, but the term "rare" has been retained by habit in identifying the group.

Monazite, 1 of the 2 principal commercial sources of these metals, is mined by placer-mining methods, employing dredges similar to those used for mining gold in many areas of the world. Gravity-concentration devices, such as jigs, tables, and spiral separators, are used to obtain mixed concentrates of the heavy minerals. In addition to monazite, these concentrates usually contain magnetite, ilmenite, rutile, zircon, garnet, and staurolite. They may contain numerous other minerals, including kyanite, columbite, tantalite, euxenite, samarskite, gold, and platinum. These complex mixtures require intensive treatment to separate the marketable products from each other. For this purpose, magnetic and electrostatic separation methods are generally used, sometimes preceded or followed by wet or dry gravity concentration. Chemical treatment may be required when certain refractory minerals occur as major constituents of the heavy concentrates.

The methods for processing monazite vary considerably, and the details usually are closely guarded commercial secrets, but the essential principles have changed little since the beginning of the century. Monazite concentrates are finely ground and digested with concentrated sulfuric acid in cast-iron pots until all monazite is completely decomposed. The mixture, containing soluble rare-earth sulfates, is poured into cold water in a lead-lined vat, stirred until solution is complete, and then allowed to settle. The insoluble minerals, such as rutile, quartz, and zircon will settle out, and the rare-earth-containing solution is decanted off. Magnesium hydroxide is added to neutralize the free acid and causes the rare-earth salts to precipitate fractionally. Thorium phosphate is concentrated in the first fraction, being less soluble than the phosphates of the rare-earth metals. The thorium-rich precipitate is filtered and dissolved in acid, and the fractional precipitation is repeated. The precipitate is purified further by boiling with oxalic acid. Thorium oxalate is less soluble in mineral acids than the rare-earth oxalates; therefore, the thorium can be removed at this stage. After the bulk of the thorium has been removed from the original sulfuric acid solution, sodium bisulfate is added to precipitate the double sulfates of the cerium group. This precipitate is dissolved in hydrochloric acid; the sodium, phosphorus, manganese, and chromium compounds are removed, and the acid solution is evaporated in an atmosphere of hydrogen chloride gas. The chlorides are fused in a cast-iron pot, and the melt is boiled to expel volatile impurities. Electrolytic methods are used to reduce the mixed chlorides to metal. Direct electrolysis of fused chloride at 900° C. with graphite anodes and a current density of 6 to 7 amperes per square inch at 10 volts will produce finely divided metal.

Bastnaesite is mined in California by open-pit methods and processed in a flotation mill and roasting plant. Although the operation was considered to be in an experimental stage in 1954 the details of processing were said to have been completely worked out, and any quantity of bastnaesite likely to be required could be produced.

For those uses requiring rare-earth compounds other than oxides, bastnaesite concentrates can be produced containing over 50 percent rare-earth oxides with a high yield. Chemical treatment of this product is required to produce rare-earth chlorides, fluorides, nitrates, etc., as desired. The impurities are mainly lime, which may be removed by leaching with dilute hydrochloric acid, and barium sulfate, which remains as a residue after the rare-earth content is dissolved with concentrated hydrochloric acid. Any desired rare-earth compound can be produced from the resulting rare-earth chlorides.

Some new uses being developed for rare-earth elements in the iron and steel industry employ the oxides of these metals (*8*). For this purpose, high-grade bastnaesite concentrates can be produced, at some sacrifice of recovery, and leached with dilute acid to eliminate lime. The leached concentrates are then calcined to remove fluorine and carbon dioxide. Practice in 1954 was to produce only 1 grade of concentrates assaying 65 to 70 percent rare-earth oxides, with a recovery of about 70 percent. These all-purpose concentrates are usable for the production of rare-earth chemicals or, after roasting, as additives in the iron and steel industry.

Most commercial uses of the rare-earth metals employ misch metal because separation and purification procedures of the individual elements, for the most part, have been too complex and expensive. Recent development of separation and purification techniques, however, has made a number of high-purity rare-earth compounds available. Pure lanthanum, cerium, neodymium, and praseodymium compounds have been produced for some time; and these same metals, as well as the rarer members of the group, have been prepared in spectrographically pure form at the Ames (Iowa) Laboratory of the Atomic Energy Commission at Iowa State College (*18*). Although enough background information is not available to predict many of the possible uses of pure rare-earth compounds and metals, separation of these elements in large quantities will affect many industries.

USES

The metals derived from processing the rare-earth minerals are consumed principally for the following purposes: Glass industry, 30 percent; carbon-arc-electrode cores, 25 percent; misch metal and ferrocerium, 20 percent; and miscellaneous, 25 percent (*6*).

Rare-earth fluorides and oxides are used as cores for high-luminosity carbon electrodes. Ordinary carbon arc lamps, although of high intensity, dissipate much of their intensity in the infrared region of the spectrum. Addition of the rare-earth compounds results in a brilliant white light because of the intense emission in the visible region. High-speed photography has been made possible by the use of high-luminosity electrodes.

Lanthanum in the pure metallic form is ductile and soft and has a melting point of about 1,600° F. It oxidizes in air and is attacked slowly by cold water and rapidly by hot water. The oxide has found application in the optical industry because it yields a special noncorroding optical glass of high refractive index for cameras and other instruments.

Cerium, the most abundant member of the rare-earth group, is an iron-gray metal. It is soft and ductile and melts between 1,418° and 1,472° F. In the form of wire it burns brilliantly when heated. The metal is a powerful reducing agent and is used effectively in alloys. It is the active ingredient in lighter flints. Cerium oxide is used as an opacifier in porcelain coatings for signs and, based on physical polishing properties, as an abrasive for polishing lenses and mirrors. The oxidizing effects of cerium compounds have led to their use in photography, as oxidizing catalysts in organic preparations, and as an ingredient in driers. Cerium salts also play an important role in leather tanning. The oxalate is used for treatment of seasickness and nervous disorders.

The salts of praseodymium and neodymium are used in the ceramic industry for coloring glass and glazes. A mixture of the salts is used in manufacturing lenses for better-grade sun glasses to provide a sharp cutoff of ultraviolet light.

Samarium metal is extremely difficult to isolate in the pure form. Samarium compounds have been used as sensitizers for phosphors excited in the infrared region.

Some properties of europium, gadolinium, terbium, dysprosium, holmium, erbium, thulium, ytterbium, and lutetium have been determined, but only small quantities of the metals have been made.

Other separated rare-earth compounds have the following uses: Ceric ammonium nitrate, used as a scavenger in the production of explosives; praseodymium oxalate, used in the fabrication of high-temperature crucibles; lanthanum oxide, used as an ingredient in aerial photographic lenses; didymium carbonate, used in radio condensers and as a glass decolorizer; cerium hydrate, used as an ingredient in optical glass for atomic-energy piles; cerium oxide of very high purity, used in windows to view atomic reactor proceedings.

Rare-earths are employed in the glass industry for coloring, decolorizing, and the production of glass with special physical and optical properties. The usual coloring agents lower the light permeability and index of refraction of glasses, but the use of neodymium and praseodymium does not affect these properties. Pure neodymium yields a changing color tone between red and lilac, and praseodymium, a color tone between green and yellow. Didymium (a mixture of neodymium and praseodymium) is used to make a neutral-gray glass. Cerium oxide is not a powerful colorant but, with titanium, imparts a reddish-yellow color. Neophane glass, containing neodymium, is ideal for spectacle lenses. Glasses containing 2 to 4 percent cerium oxide absorb ultraviolet light waves. The use of cerium dioxide as a decolorizing agent in place of manganese reduces the coloration of the glass by exposure to X-ray radiation.

Misch metal, the mixture of all the rare-earth elements in metallic form, contains 45 to 55 percent cerium. Because of the predominating constituent, the mixture is often referred to as cerium. Misch metal has been alloyed with copper, nickel, zirconium, zinc, aluminum, and magnesium.

An aluminum alloy with 11 percent misch metal has stress-rupture properties superior to present commercial alloys in the range 700° to 800° F. (*10*). Additions of misch metal improve the creep resistance of extruded magnesium in the range 400° to 600° F. (*9*). This alloy has many highly important applications in the construction of aircraft parts exposed to intermediate operating temperatures, with a consequent significant decrease in dead weight and increased load capacity, speed, and range of operations. These alloys are being used in jet engines and gas turbines. Addition of rare-earth alloying materials to hot-dip aluminum baths gives smooth coatings on mild-steel parts without use of a flux.

An increasing use for rare-earth metals is in the manufacture of steel and alloy steels (*7*). Additions of small quantities of rare-earths, preferably to the ladle, result in very fine grained strong steel. The action of the rare earths is believed to be to form compounds with nonmetallic or gaseous material within the melt and form many nuclei for crystallization of the main mass of metal. The sulfur content of steel to which rare-earths have been added is effectively reduced by the formation of relatively stable sulfides, which rise through the molten metal to form a slag. These additions to steels increase resistance to low-temperature oxidation and corrosion, impart better hot- and cold-working properties to stainless varieties; improve workability in tool steels; and improve electrical properties of silicon electrical-grade steels.

Rare-earth metals added to cast iron act as powerful deoxidizers and aid in removing sulfur from the molten metal. Additions to malleable-iron melts act as a carbide stabilizer. Resistance to scaling at higher temperatures and to certain corrosive atmospheres results from additions of rare earths to cast iron.

The rare-earth metals are superior to aluminum and silicon and equal to magnesium as deoxidizing agents for copper and copper alloys. Small additions of rare-earths to a copper alloy containing nickel and silicon reportedly make them heat hardenable.

Nickel alloys are made more resistant to oxidation by additions of the rare-earth metals. Appreciable grain refinement has been observed in alloys containing nickel and copper, and the castability of the alloys is improved.

The beneficial effects of rare-earths in certain heat-treatable aluminum alloys have been known for some time (*10*). An aluminum alloy containing 1.5 percent manganese and 4.5 percent rare-earth metals can be forged more easily than Dural. Other aluminum-rare-earth alloys are suitable for use in pistons and other high-temperature applications.

Magnesium-rare-earth-zirconium alloys have a combination of good casting qualities and mechanical properties that makes them attractive for some important lightweight stressed components of aircraft engines (*12*).

Miscellaneous uses include: Waterproofing, mildewproofing, weighting and dyeing of fabrics, and compounding printing inks and phosphors (*6*).

SUBSTITUTES

Most of the carbon-arc electrodes containing rare-earth cores are consumed by the motion-picture industry as a source of light in filming and projecting motion pictures; a substantial quantity of misch metal is used in the form of ferrocerium alloy for flints in mechanical lighters; and a large portion of the rare-earths consumed in glass manufacture is used for decolorizing glass batches. In each instance, substitute materials are available which are comparable and cost less, but they are greatly inferior to the rare-earth metals. In the case of carbon-arc electrodes substitution is accompanied by a loss in light-emission efficiency. Iron and calcium fluorides may be used, but with a 30-percent reduction in light intensity. These substitute electrodes are objectionable for projecting black and white pictures, and the light is not white enough for projecting color pictures.

Zirconium-impregnated lead flints might possibly replace ferrocerium during an emergency. The substitute flint would, however, be more expensive and less satisfactory.

White arsenic is a satisfactory substitute for rare-earth compounds used in decolorizing glass batches, but rare-earth compounds have no known substitute in the production of special glasses possessing ultra-violet absorptive properties or the unusual optical characteristics that make them desirable for strategic purposes. Rare-earth oxides could be replaced by rouge, a less expensive and less strategic compound, for polishing glass and metal. However, the rare-earth oxides are considered greatly superior for this application and are especially desirable because of the savings in man-hours and machines that their use affords.

PRICES

E&MJ Metal and Mineral Markets (Mar. 3, 1955) quoted monazite, c. i. f. United States ports, total rare-earth oxides, including thorium, massive, 55-percent grade, 13 cents per pound. Monazite sand, 55-percent grade; 18 cents per pound; 66 percent, 20 cents per pound; 68 percent, 22 cents per pound. Individual contracts are negotiated between domestic monazite producers and buyers, and the price depends upon the quantity and grade of material delivered. In 1954 these prices were $300 to $400 per short ton.

There is at present no fixed price schedule for bastnaesite ores or concentrates. Compounds made from roasted bastnaesite concentrates (essentially rare-earth oxides) are sold at $1 per pound of compound.

Prices quoted in March 1955 for misch metal and ferrocerium were $4.50 and $8.00 per pound, respectively. High-purity cerium metal was quoted at $18.00 per pound. Rare-earth chlorides were sold at 40 to 45 cents per pound; rare-earth fluorides at $1.00 per pound; and lanthanum oxide at $10.00 per pound. Prices of other individual rare-earth metals and compounds depend upon the quantity and quality of material purchased.

OUTLOOK

There is an increasing need for rare-earth minerals, partly because of the need for the thorium obtained from monazite for experimental use as a source material for atomic energy and partly because of the increasing use of rare-earths by industry. Interest in developing uses for these metals and compounds is probably greater today than at any time in the past; and it may be expected that, within a few years, world requirements for rare-earths will be considerably greater than at present.

Because of the strategic importance of both the rare-earth metals and thorium, the production of monazite will probably be encouraged by both Government and industry. Domestic resources of monazite have to date been found in the alluvial deposits of Idaho, North Carolina, South Carolina, and Florida; but the monazite content of these deposits is usually low, making commercial exploitation marginal at best. The present method of basing the value of monazite on its rare-earth plus thoria content assigns most of the value to the rare-earths. In future, the thorium may have more value than the rare-earths in monazite, and higher prices may be paid to producers on this basis. At any rate, the future of monazite mining would appear to be more concerned with developments in requirements for thorium than for the rare-earth elements.

The supply of bastnaesite, which usually contains almost no thorium, is ample to permit expansion of domestic rare-earth production many times. These deposits are considerably higher grade than any known monazite deposit with respect to rare-earth elements, and unless a superabundance of the rare-earths becomes available as a coproduct of thorium they should become the principal source of these elements.

Other rare-earth-bearing minerals, such as euxenite, pyrochlore, samarskite, and fergusonite, are important for other valuable elements, mainly columbium, tantalum, and uranium. Current interest in these minerals is extensive, and some output of rare-earths will undoubtedly be forthcoming from their treatment for the recovery of these elements. Some of these minerals are richer in certain rare-earth elements than either bastnaesite or monazite; for this reason they could become important sources of these particular rare-earths.

PROBLEMS

The concentration and separation of rare-earth minerals from associated minerals or gangue are not simple. Investigations of present monazite-beneficiation methods indicate that an overall recovery of about 80 percent of the monazite can be made in mining and concentrating domestic placer sands. Major losses occur in the finest fractions of the sand. The recovery of fine-particle-size bastnaesite from complex gangue materials is even more difficult, resulting in relatively low recovery of the mineral by present ore-dressing and metallurgical processes. Another problem, peculiar to the monazite-mining industry of Idaho, is the lack of markets for the accessory heavy minerals being recovered with monazite. The rutile, ilmenite, and zircon could pay a large part of the cost of a mining operation if a suitable market were available.

The main problem confronting producers of bastnaesite is the possibility that increasing production of thorium from monazite will result in a surplus of rare-earth compounds as coproducts at very low cost. The solution of this problem probably lies in development of new large-scale uses for these little-known metals. A number of known uses do not employ rare-earth metals because of their relatively high cost. Research is needed to develop lower cost-reduction methods that will permit the production of individual rare-earth metals in large enough quantities at prices that will encourage their use.

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RHENIUM

By

John D. Sargent [1]

RECENT studies have indicated that the production and consumption of rhenium are likely to increase several hundred percent in the near future.

Summary

Rhenium, discovered just 30 years ago, has many unusual properties. Few minerals have been found that contain this element in quantities greater than a few tenths of 1 percent.

Molybdenite has been the only important source material for commercial production of this metal in both Germany and the United States. World production of rhenium is estimated at a few hundred pounds yearly, about half of which has been produced in the United States at the University of Tennessee.

Rhenium is being considered for utilization in electric contact points, spark-plug points, electrodes, corrosion-resistant alloys, and catalysts.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

HISTORY

When Mendeleev enunciated his periodic law in 1871 and drew up his table of the elements, he left a number of blank spaces for elements then unknown but which, he confidently predicted, would ultimately be discovered (*2*).[2]

Concerning the homologues (or congeners) of manganese Mendeleev had little to say beyond the observation that they were expected to resemble manganese in general chemical behavior. Efforts to identify these missing elements were, for a long time, unsuccessful.

In 1898 Harley and Ramage, after a spectroscopic examination of a large number of minerals, declared that the existence of the missing elements was improbable. As late as 1924, Bosanquet and Keeley could find no trace of any new element in pyrolusite (MnO_2) by a method of cathode-ray bombardment, but in the following year the discovery of dvi-manganese or rhenium was reported independently in three countries (Noddack, Tacke, and Berg in Germany, Druce and Loring in England, and Heyrovsky and Dolejsek in Czechoslovakia).

Ida and Walter Noddack, German chemists, are generally credited with the discovery of rhenium in 1925. During that year they examined 30 complex minerals, in 5 of which they believed they identified rhenium. From the treatment of 1 kg. of columbite they actually obtained about 50 milligrams of a product containing 5 percent of a new element which they named rhenium. With the help of German scientific bodies, the Noddacks visited Scandinavia and Russia to search for and acquire rhenium-containing minerals, but in no instance did they find any containing over 0.001 percent of the element. In 1928, with the assistance of Siemens and Halske, they obtained about 1 gram of rhenium from 700 kg. of molybdenum glance. Later they found that the waste liquors from a molybdenum-processing plant near Magdeburg, Germany, contained small quantities of molybdenum in which as much as 1.5 percent of rhenium was present. Several hundred grams of the metal was recovered from the molybdenum by 1930.

At the University of Tennessee in 1942, A. D. Melaven and J. A. Bacon developed a process for extracting the element from dusts that accumulated in roasting molybdenum ore. This Tennessee plant was the only rhenium producer in the United States for many years and had a total output of several hundred pounds of the metal and its salts.

PHYSICAL AND CHEMICAL PROPERTIES

Rhenium metal (*2, 4, 8*) in the powdered form varies from gray to brownish-black. When the powder is heated, it can be compressed into bars or drawn into filaments. The metal when melted in an electric arc or electrically sintered into rods, is silver white and resembles platinum. In rod form the metal is exceptionally hard and has a density of about 21. The melting point of rhenium has been reported as $3{,}180 \pm 20°$ C. Only carbon and tungsten have higher melting points. Rods of rhenium are ductile and can be forged, hammered, and rolled.

The latest accepted atomic weight for rhenium is 186.31, on the basis of a composition of 37.1 percent rhenium–185 and 62.9 percent rhenium–187. Rhenium isotopes 182, 183, 184, 186, 188, 189, 190, and 191 have been produced artificially; all are radioactive, with half lives ranging from 17 minutes to 200 days. Rhenium–187 as it occurs in nature is radioactive and has a half life of 4 trillion years.

Rhenium can be detected spectroscopically by triplet lines at 3,640° A, in concentrations as low as 1 part in 10 million.

Chemically, rhenium resembles manganese. Hydrochloric acid has no appreciable action on the metal; however, it is oxidized by nitric acid and hot sulfuric acid. At room temperature rhenium-metal powder is inert to air. It is oxidized to Re_2O_7 in air or oxygen at 400° C. This oxide dissolves freely in water and alcohol and is acid enough in water solution to dissolve zinc and iron, with evolution of hydrogen. Rhenium rods and bars are more resistant to attack.

Rhenium has a valence of -1, 1, 2, 3, 4, 5, 6, and 7 and occupies a position between copper and thallium in the electrochemical series.

OCCURRENCE

Rhenium has been estimated to constitute only about 1×10^{-9} of the earth's crust (*4*). The discoverers of the element found it in the spectra of just 100 of 1,600 mineral specimens examined; molybdenite from Norway contained the largest quantity of rhenium—as much as 3,080 parts in a million. Gadolinite, wulfenite, alvite, and thortveitite have been found to contain up to 1 part in a million. Smaller quantities have been found in columbite, tantalite, pyrolusite, wolframite, rutile, chalcocite, chalcopyrite, pyrite, euxenite, fergusonite, orangite, samarskite, zircon, monazite, and some complex

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

platinum minerals. The range of rhenium content is indicated by the following analyses:

Locality	Material	Rhenium (p. p. m.)
Stavanger, Norway	Molybdenite	3,080.
Do	do	3,110.
Lainejaur, Sweden	do	2,500.
Arendal, Norway	do	1,420.
Cananea, Mexico	do	130.
Hvalerogene, Norway	do	91.
Turkey	do	51.
Peru	do	49.
Flekkefiord, Norway	do	13.
Shirakawa, Japan	do	10.
Bulgaria	do	11.
Various localities	Wulfenite	0.3–3.3.
Norway	Gadolinite	0.03–1.1.
Norway and Madagascar	Columbite	0.05–0.2.
South Africa and Finland	Tantalite	0.03–0.08.

A Bureau of Mines diamond-drilling investigation of two small molybdenite deposits in Halifax County, N. C., in 1943 (Report of Investigations 4156) indicated that molybdenum sulfide concentrates from drill cores contained 0.005 to 0.009 percent rhenium. The percentage calculated for pure molybdenite would be about 0.016 (*7*).

Rhenium has been produced commercially from the treatment of flue dusts and residues obtained from the roasting and smelting of molybdenum ores. A number of analyses by the University of Tennessee of ores and dusts of United States operations are listed as follows:

Company	Material	Rhenium (p. p. m.)
Miami Copper Co	MoS_2 concentrates	(About) 20.
	MoS_2 roaster flue dust	3,500–15,000.
Climax Molybdenum Co	Hot precipitator dusts	70–140.
Molybdenum Corp	Dust from stack	2,600.
	Dust-collector product	180.
Utah Copper Co	MoS_2 concentrates	400.
	Cu concentrate	0.
Kennecott Copper Corp. (Hurley, N. Mex.).	Roaster dusts	120–220.
Squaw Peak mine (Ariz.)	MoS_2 concentrates	12.

SIZE AND DISTRIBUTION OF INDUSTRY

The initial production of rhenium was from molybdenum residues at a plant of Vereinigte chemische Fabriken at Leopoldshall, near Magdeburg, Germany. Production began in 1930 and continued until World War II. This production was estimated to be about 200 pounds yearly and supplied the demands of the world. The Kupferkammer lead smelter of Mansfeld Kupferbergbau und Hüttenwerk, Hettstedt, East Germany, is reported to have been producing the metal since October 1949.

In 1942 the University of Tennessee built a plant to produce potassium perrhenate and rhenium-metal powder. The production from this small plant has increased from 10 pounds in 1943 to a present annual production about 100 pounds, depending upon the demand for the metal or its salts. About 240 pounds of rhenium had been produced from the 17 tons of molybdenite roaster flue dust processed to 1954.

Raw material for production of rhenium in the United States has been obtained from the Miami Copper Co., at Miami, Ariz. This company is mining a copper sulfide ore with a small molybdenum sulfide content. About 500,000 pounds of MoS_2 is separated by selective flotation from this ore annually. The rhenium content of this flotation concentrate is estimated to be 20 p. p. m.

Kennecott Copper Corp. probably will be in relatively large scale rhenium production in the near future and coupled with production from other plants the total should be over 1,000 pounds a year by 1956.

EXTRACTIVE METALLURGY

At the Miami Copper Co. plant the MoS_2 is sent to a roaster for conversion to an oxide (MoO_3). The rhenium contained in the MoS_2, probably as a sulfide, is converted into a volatile oxide (Re_2O_7). The flue dust is sent to a cyclone precipitator for concentration. For several years these cyclone concentrated dusts ran 1.0 to 1.5 percent (10,000 to 15,000 p. p. m.) rhenium. However, the roasting temperature was raised a few years ago. This resulted in a greater loss of the volatile Re_2O_7, so that the dusts assayed only 0.38 percent (3,800 p. p. m.) rhenium. (The Re_2O_7 goes up the stack in a white, foglike vapor and is trapped by other Re_2O_7 particles.)

The concentrated flue dusts are shipped in wooden barrels to the plant of the University of Tennessee at Knoxville.

The rhenium oxides are soluble in water and easily removed from the flue dust with water and compressed-air agitation in a filter-bottom tank.

The resultant product is a crude potassium perrhenate, but this is refined by digesting the salt in water, filtering off the insoluble residues, and cooling to 7° C. to recrystallize the $KReO_4$. (A saturated solution is 4 grams per liter at 7° C.) The white $KReO_4$ crystals are removed and dissolved in hot water, and the above recrystallization process is repeated 5 times to produce $KReO_4$ with a purity of over 99.9 percent.

Powdered rhenium metal is made by placing the refined $KReO_4$ in a silver tube, which in turn is placed in a heavy-metal bomblike furnace. The air is removed by forcing nitrogen through the container, followed by hydrogen under pressure, and the container is sealed. Heat is applied to the furnace, and at 235° to 350° C. the $KReO_4$ is converted to the metal ($2\ KReO_4+7H_2 \rightarrow 2KOH+6H_2O+2Re$). The furnace is cooled slowly, and the mixture is removed and boiled to make the KOH soluble. The grayish-black rhenium-metal powder is then filtered from the solution. Rhenium-metal powder produced from ammonium perrhenate is easier to sinter.

Dr. S. R. Zimmerly and his staff at the Salt Lake City research laboratories of the Kennecott Copper Corp. have obtained experimental recovery of rhenium from corporation ores and have succeeded in developing an entirely new method of separation. Melting and fabrication problems have been worked out by Dr. D. K. Crampton at Kennecott's subsidiary, Chase Brass & Copper Co., in Waterbury, Conn.

USES

Rhenium alloys readily with tungsten, nickel, cobalt, tantalum, platinum, rhodium, iridium, gold, and iron. These alloys are reported to offer excellent corrosion-resistant properties. For pen nibs an alloy containing 75 percent rhenium, 15 percent rubidium, 7.5 percent niobium, and 2.5 percent beryllium has been proposed. Iridium and rhodium alloys are worked more easily if alloyed with 10 percent rhenium, as the resultant alloy can be drawn into fine wires and rolled into thin ribbons. Thermocouples of platinum-rhenium and platinum, platinum-rhenium and rhodium, and iridium-rhenium and iridium, have high electromotive force and can be used at temperatures up to 1,900° C.

Rhenium and rhenium alloys are being investigated for use as electric contact points, sparkplug points, and non-consumable-electrode tips for melting and welding.

The use of rhenium sulfides as hydrogenation catalysts has been studied. In comparison to molybdenum sulfide and cobalt polysulfide, rhenium heptasulfide was found to be the most active, most consistent, most easily stored for indefinite periods, most stable to hydrogenative decomposition at high temperatures, most resistant to "poisoning," and most insoluble in strong nonoxidizing acids. Colloidal rhenium and rhenium compounds have been utilized in the hydrogenation of coal, coal tar, and mineral oil. Ammonia can be synthesized with a rhenium catalyst.

Both in the free state and alloyed with copper, rhenium exhibits distinct catalytic properties, which have been used for hydrogenation of ethylene to ethane and conversion of carbon monoxide and hydrogen to methane (*2*). Rhenium oxides also can promote the formation of sulfur trioxide from the dioxide and oxygen, serving as oxygen carrier. The disulfide of rhenium has been used as a catalyst for the dehydrogenation of alcohols to aldehydes and ketones (*2*). Powdered rhenium is used as a catalyst in the manufacture of chloroform and antiknock compounds.

Rhenium wire has been patented for use as a filament in electric-lamp bulbs and in radio tubes. Rhenium is said to volatilize even less than tungsten, but no lamps or tubes containing it have appeared on the market. Rhenium filaments having a long electron emission should merit consideration in the electronics field.

Electroplating thin coatings of rhenium is readily accomplished. These coatings resist corrosive agents. Bright, hard deposits are obtainable from both acid and alkali baths. Thus, rhenium plating can be used to protect such metals as silver. The baths had good "throwing powers," and action was complete in 1 to 1½ minutes. The resistance of rhenium to hydrochloric acid is so great that rhenium-plated brass articles submerged in this acid for 3 days showed no effect.

PRICES

The total cost of producing the first gram of rhenium in 1928 by the discoverers is estimated to have been $15,000. In 1942 the price of the metal in the United States was $14 per gram; however, in Germany the price was reportedly $4.00 per gram.

The present price per gram of rhenium salts and metal as sold by University of Tennessee are as follows:

Product	First 10 grams at—	Next 90 grams at—	Each gram over 100 grams at—
Potassium perrhenate ($KReO_4$)	$2.00	$1.50	$1.00
Rhenium metal from $KReO_4$	3.25	2.50	1.75
Rhenium oxide (Re_2O_7)	3.00	2.40	1.60
Ammonium perrhenate (NH_4ReO_4)	4.00	3.25	2.50
Rhenium metal from NH_4ReO_4	6.00	4.90	3.80

OUTLOOK

In the past, industry had little interest in producing rhenium or in saving of byproduct materials containing the element because of the low content of rhenium in ores and byproducts and the limited industrial demand for the metal and its compounds. Now that important uses and markets for rhenium have developed, rhenium probably will take its place among the industrial metals.

PROBLEMS

The cost of producing rhenium in the United States has been high for two reasons.

(1) The source material—molybdenum sulfide roaster flue dust—was shipped some 2,000 miles from the Miami Copper Co. plant, Miami, Ariz. to eastern Tennessee. The dust contained only 1 to 20 pounds of rhenium per ton of material. After extraction of the rhenium, the dust was shipped back to Arizona. It had not appeared to be economic to install a rhenium plant at the site of Miami Copper Co. because the company was rapidly depleting its known molybdenum-bearing ore bodies. This company has since developed another ore deposit, also reported to be rhenium-bearing.

(2) Although the extractive metallurgy of rhenium is not difficult, the filtering of residues and the crystallization of $KReO_4$ is time consuming. The metal and its salts are produced by students in the Department of Chemistry at the University of Tennessee under supervision of staff members in their spare time from classes. Most methods of extracting rhenium from source materials are slow and costly. Better processes for producing both the salts and the metal are needed.

Of primary importance is the need to find a source of supply of rhenium. Insufficient investigation has been conducted to ascertain the content of the metal in ores and smelter byproducts. Molybdenum sulfides are known to contain small quantities of the element, but further investigation might disclose the existence of significant quantities of rhenium in other minerals or complex ores.

The absence of rhenium in flue dusts of some smelters may only indicate that high temperatures have dissipated the highly volatile rhenium oxide into the atmosphere. If such losses are discovered, processes to retain the rhenium in the flue dusts may be developed.

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RUBIDIUM

By

John D. Sargent[1]

RUBIDIUM is a little-known alkali metal usually found associated with cesium, which it closely resembles. Although costly, rubidium has several unusual properties that warrant its utilization in specialized applications in electronics, chemistry, and medicine.

Summary

The chief source of rubidium is lepidolite—a micaceous lithium mineral usually occurring in pegmatite dikes. Rubidium is recovered by fractional crystallization in processing lepidolite for its lithium content. Only about 100 pounds of rubidium metal and compounds is produced annually, mainly for electronic uses, but raw materials are available if demand should warrant greater output.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

HISTORY

In 1861 Bunsen and Kirchoff discovered rubidium while making a spectrographic analysis of the alkaline constituents of lepidolite from Saxony. The dark red lines of the spectrum, which indicated the presence of the new alkali metal suggested the name rubidium from the Latin word rubidus, meaning dark red (*4*).

OCCURRENCE

Rubidium and cesium are associated in nature, rubidium usually predominative. The abundance of rubidium in the igneous rocks of the crust of the earth has been estimated at 0.035 percent. Lepidolite, a complex lithia mica found in pegmatites, is the principal commercial source of the metal. The rubidium content of lepidolite in the United States ranges from a trace to over 3 percent Rb_2O, and averages about 1.5 percent (*3*). Carnallite, pollucite, leucite, and zinnwaldite are other source minerals that contain up to 1 percent Rb_2O. Other natural occurrences of the element are in salt wells, sea water, and animal organisms (*3*). Traces of the element have been reported in meteorites, soils, mineral water, and seaweed. The rubidium content of commercial potassium chloride recovered from subterranean brines in Michigan is approximately 0.05 percent. It is estimated that about 1 ton of rubidium might be recovered annually from such brines.

Reserves of lepidolite in South-West Africa are estimated to be about 60,000 tons, assaying about 2.20 percent Rb_2O. African pollucite sometimes contains as much as 3.73 percent Rb_2O. Zinnwaldite from the Union of South Africa is reported to contain up to 1.49 percent Rb_2O and that from Japan about 0.09 percent.

In the United States, deposits of rubidium minerals occur in California, South Dakota, New Mexico, and Maine (*3*). Pollucite is found in tonnage quantities in South Dakota and Maine. Lepidolite occurs in New Mexico, South Dakota, Colorado, Maine, and California.

PROPERTIES

Rubidium closely resembles cesium and potassium in chemical properties. It is oxidized more readily than potassium and, upon exposure to air, rapidly becomes covered with a gray-blue oxide film consisting of a mixture of Rb_2O_2 and RbO_2. It ignites spontaneously if exposed to dry oxygen at room temperature (*3*). Because of its volatility and reactivity, rubidium usually is kept in a vacuum, in hydrogen, or under a liquid, such as benzene or oil. It is usually marketed in sealed-glass tubes or under oil in metal containers.

Rubidium reacts readily with water, even at low temperatures, to form rubidium hydroxide. The reaction at room temperature is violent, and the evolved hydrogen is ignited. Rubidium unites directly with bromine or chlorine, accompanied by ignition (*3*).

Rubidium alloys with the other alkali metals except lithium. It also alloys with gold, forming a green compound ($RbAu_2$), which is not as stable as the corresponding potassium-gold compound (*3*). Cesium and rubidium are completely miscible in solid solutions, and the compounds of rubidium resemble those of potassium, with which they are isomorphous. The solubility of these compounds is so similar that it is difficult to separate them. Rubidium has a marked tendency to form well-defined crystals of double salts, which are used in microchemistry. Amalgams of rubidium and mercury, with various compositions, may be formed.

The naturally occurring rubidium radioisotope Rb–87, which emits beta rays, has a half life computed to be about 600,000,000 years. A method for determining the age of rocks is based upon observing the ratio of Rb–87 content to that of the stable strontium end product (*1*).[2]

METALLURGY

Rubidium is not produced on a large scale, but small quantities are extracted from lepidolite as demand requires. Rubidium and cesium are extracted with the more abundant alkali metals by decomposition with sulfuric and hydrofluoric acids. To a solution containing sodium, potassium, rubidium, and cesium a quantity of ammonium alum is added, and the solution is evaporated to crystallization. Mixed crystals of rubidium and cesium alums form, which can be removed from the mother liquor. The rubidium and cesium are subsequently separated from each other by fractional crystallization. Metallic rubidium may be pre-

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

pared in the following ways, depending upon the raw material: (1) Electrolysis of the fused cyanide or chloride; (2) reduction of rubidium hydroxide with aluminum or magnesium; (3) reduction of rubidium carbonate with carbon or magnesium; (4) reduction of rubiduim chloride with calcium; or (5) ignition of rubidium tartrate to white heat (*2*). The metal usually is purified by distillation after reduction.

The electrolysis of fused rubidium chloride with a graphite anode and an iron cathode was carried out by Bunsen in his initial preparation of rubidium in 1861.

In Germany lepidolite residues resulting from extraction of lithium have been reprocessed for rubidium and cesium since 1935. Before 1935, pollucite imported from the United States was the source of supply. Lepidolite residues were treated with sulfuric acid and water, and rubidium alum was recovered by fractional precipitation. About 4 metric tons of residues, containing 2.5 to 4.0 percent rubidium and 0.7 to 1.5 percent cesium, were processed annually. Metallurgical recovery was estimated to be about 90 percent.

Hydrochloric acid was used to process pollucite. After proper digestion and pH adjustment, the rubidium and cesium contents were precipitated as oxalates, which were then converted to hydroxides and reacted with powdered magnesium in a hydrogen atmosphere, yielding rubidium metal. Both rubidium and cesium metal were handled, stored, and shipped under liquid paraffin.

PRODUCTION

The United States and Germany are the principal suppliers of rubidium and its compounds (*1*). During World War II the Germans produced about 100 pounds of cesium and rubidium annually. Rubidium metal and salts were produced from lepidolite and pollucite by E. Merck, Chemische Fabrik Co., Darmstadt. Germany reportedly has resumed production of limited quantities of rubidium.

Producers of rubidium and its compounds in the United States are:

DeRewal International Rare Metals Co., Philadelphia, Pa. (Rb metal and compounds.)

Fairmount Chemical Co., Newark, N. J. (Rb metal and compounds.)

A. D. Mackay, Inc., New York, N. Y. (Rb metal and compounds.)

Maywood Chemical Works, Maywood, N. J. (Rb compounds.)

Harshaw Chemical Co., Cleveland, Ohio. (Rb compounds.)

USES

Rubidium is used for much the same application as cesium, often in combination with that metal, and marketed as cesium. It is employed as a getter in radio vacuum tubes and can be plated in very thin layers having a high degree of photoemission.

Rubidium compounds are used to a greater extent than the metal. The iodide is particularly useful in medicine, where it is employed for syphilis therapy and also is substituted for potassium iodide in goiter treatment (*1*).

Rubidium-mercury amalgams are used as catalytic agents in hydrogenation processes.

The formation of characteristic crystals of double salts of rubidium is employed in microchemical procedures (*2*).

PRICES

Rubidium metal, chemically pure, double distilled, in sealed glass tubes, was quoted during 1954 at \$2.75 per gram, and rubidium salts at 30 to 60 cents per gram (*1*).

OUTLOOK

Commercial activity in rubidium is very small; but, because there is an ample supply of lepidolite ore in the United States and abroad, particularly in South-West Africa and the Union of South Africa, production can be increased considerably if new uses or increased consumption develops in the future.

PROBLEMS

The relatively high cost of rubidium and its compounds has retarded wider utilization of the element. A less costly and improved method for processing rubidium-bearing materials is needed.

Lepidolite—the chief source of rubidium—is used in the manufacture of glass. Much of the rubidium, therefore, is lost in the glass products or in the residues of the glass industry.

The very fact that the principal source of rubidium is a pegmatite mineral subjects the metal to the vagaries common to other rare metals found in such deposits.

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SALT

(Including Sodium)

By

Joseph C. Arundale[1]

SALT has been invested with symbolic meaning by mankind since earliest times. To various peoples it has been the symbol of friendship, fidelity, purity, hospitality, permanence, and destiny. It has been used as a currency; has had a religious significance; been a source of political power; and caused wars. The many popular expressions and beliefs involving salt are evidence of the importance attributed to it. Today it is one of the major pillars of the chemical industry.

Summary

Salt (NaCl) is abundant, widespread, and inexpensive. It occurs and is recovered and used in both solid and brine form. Reserves are enormous. The sea alone is an inexhaustible source. However, economics and the requirements for chemical purity limit the areas in which salt is produced. Fortunately, these areas are widely distributed in the United States, and commercial salt is available to most major industrial districts at reasonable cost.

There are thousands of uses for salt. The well-known and oldest uses of salt are as a food, food seasoning, and preservative; today, however, the major domestic use is as a raw material in the manufacture of other chemicals.

Salt is produced in nearly every country in the world. Recovery methods vary from solar evaporation to complex procedures that result in highly refined products with strict chemical and physical specifications.

Salt production is expected to follow the trend of overall industrial activity; consumption for most uses is increasing.

Problems of the industry include development of new uses, a comprehensive appraisal of salt resources as a basis for future industrial expansion, more comprehensive economic and statistical information, and recovery of byproducts.

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[1] Assistant chief, Branch of Construction and Chemical Materials, Bureau of Mines.

BACKGROUND

HISTORY

Salt—sodium chloride—is one of the most common and widely distributed mineral materials. It occurs in solid form as the mineral halite or "rock salt" in bedded deposits and as domes or plugs. Typical of the first type of occurrence are the great salt beds of Michigan, Ohio, New York, Pennsylvania, Kansas, Oklahoma, and Texas, which underlie thousands of square miles and often are hundreds of feet thick. The large salt domes of Texas and Louisiana are typical of the latter occurrence. Rock salt occurs widespread and abundantly in sedimentary rocks of all geologic ages since early Cambrian. Salt also occurs in solution. It is found in lakes, lagoons, springs, and underground brines. The brines of Michigan, Ohio, New York, and Pennsylvania and the salt lakes of Utah and California are examples of this type of occurrence. Occurrences are widely distributed throughout the world. Finally, there is the salt in the sea, which continually is being replenished by salt leached from the earth by water and transported in solution by streams and rivers.

The first use of salt is lost in antiquity. Since the human body has a relatively constant requirement for salt, certainly its first use was as a food and since earliest time as a food seasoning and preservative. Many ancient civilizations had well-developed salt industries. Salt has been used as currency; it has caused wars and revolutions; it has been a traditional source of tax revenue. Few, if any, mineral commodities have such a long and varied history or have greater importance to mankind.

One of the first activities of the Pilgrims after landing in the New World was constructing a solar evaporation installation to supply their need for salt. From such modest projects developed an industry producing 20 million tons of salt annually from many sources and literally with thousands of uses.

PRODUCTION

In the United States about 70 percent of the salt output is produced and consumed by major chemical companies as a raw material in manufacturing other chemicals. It is the ore of sodium metal and chlorine.

Salt is produced at about 85 facilities in the United States. Of these, 9 have a production of over 1 million tons each, with a combined output of over half the United States total. Three plants produce between 500,000 and 1,000,000 tons, 30 from 100,000 to 500,000, and the remainder less than 100,000 each. Twenty-six plants produce less than 10,000 tons each.

The total United States output is over 35 percent of the estimated world production.

Three firms in the United States produce sodium metal, using salt as a raw material.

The geographic distribution of the principal salt-producing areas in the United States is as follows:

Michigan:
  (*a*) Vicinity of Detroit.
  (*b*) Manistee and Ludington.
West Central New York State.
Northern and Central Ohio.
Louisiana, in the vicinity of Weeks, Avery, and Jefferson Islands on the coastal waterway and at Winnfield in Winn Parish.
Central and Southern Kansas and the contiguous portion of Oklahoma.
Eastern Texas.
Utah, at Salt Lake and Redmond.
California, in the San Francisco Bay region, San Diego, and Saltus.

TECHNOLOGY

Salt is mined and processed by many methods throughout the world. In the most primitive method it is simply gathered in more or less pure form from surface deposits around salt springs or "dry lakes." In many regions salt is recovered by solar evaporation of sea water or brines. The method is relatively simple in principle, but many technical improvements have been developed. Sea water is introduced into shallow ponds, either by flooding at high tide or by pumping. Evaporation causes concentration of the brine, which may be moved from pond to pond as concentration is increased. Carried to its end point, a "solar salt" is produced.

Rock salt is mined either by: (1) Underground room-and-pillar or (2) dissolving by water. The first is very similar to the method used in mining soft coal. The salt bed is reached by shaft and extracted by driving gangways and panels or rooms, leaving pillars for roof support. The "ore" is broken by undercutting and blasting, then loaded and hauled mechanically. In the second method water is introduced into the salt bed through concentric pipes in a drill hole. Salt is dissolved in the water and pumped to the surface.

Where natural salt brines are being utilized the reservoir is tapped by drillholes and the brine pumped to the surface.

Much of the crude solar salt and rock salt is marketable without further processing, except crushing and screening to size. However, brines usually require further processing and refining. This is accomplished in a variety of ways. The various impurities that commonly occur in salt brine are removed chemically by treatment with lime, soda ash, and other chemicals. The salt then is crystallized, either in vacuum pans wherein the brine is evaporated in a vacuum by steam to produce "granulated" or vacuum-pan salt or in steel or concrete pans heated by steam. The product of the latter method is referred to as "grainer" salt.

Other chemical additives and further processing are determined by the ultimate use. For example, a nonhygroscopic material may be used to keep table salt "free running"; iodides may be added to prevent goiter; and sulfur and other mineral materials may be added to be fed to livestock. The final product is bagged, packaged, pressed into blocks, or sold in bulk.

Most sodium metal is produced electrolytically in Downs cells. In this cell the anode is graphite; the cathode is a cylindrical cast steel ring, and the electrolyte is a mixture of salt and calcium chloride.

USES

Probably no other mineral material has such an imposing number of uses as "common salt." It has literally thousands of direct and indirect applications in a wide range of commodities and services.

TABLE 1.—*Salt sold or used by producers in the United States, 1950–52, by uses*

[Thousand short tons]

Use	1950	1951	1952
Chlorine, bleaches, chlorates, etc	3,889	4,709	5,251
Soda ash	6,379	8,303	7,195
Dyes and organic chemicals	152	129	110
Soap (precipitant)	64	61	41
Other chemicals	711	835	721
Textile processing	129	124	127
Hides and leather	257	232	228
Meat packing	727	747	764
Fish curing	46	46	31
Butter, cheese, and other dairy products	76	69	72
Canning and preserving	163	202	212
Other food processing	239	231	258
Refrigeration	187	192	219
Livestock, agriculture, and general farm use [1]	956	1,056	1,082
Highways, railroads and other dust and ice control	591	785	817
Table and other household use	622	600	546
Water treatment	552	610	620
Metallurgy	99	117	106
Undistributed [2]	791	1,159	1,145
Total	16,630	20,207	19,545

[1] Livestock salt is about 90 percent of the total.

[2] Comprises miscellaneous uses and uses for which data may not be shown separately; also includes some exports and consumption in Territories and possessions.

The chemical industry is the predominant direct consumer of salt; and the largest single use of salt is in manufacturing soda ash, which in turn has a multitude of uses. Other uses for salt by the chemical industry are in manufacturing sodium metal, chlorine, bleaches, chlorates, hydrochloric acid, and other chlorine and sodium chemicals. Salt is used in manufacturing soap and dyes and in textile and leather processing. In the food industry salt is used in meat packing, fish curing, dairy products, refrigeration, and livestock feed. On roads it is used for ice and snow removal and for stabilizing the soil in shoulders, base courses, and secondary gravel roads. Salt is used in municipal water purification and in heat-treating, smelting, and refining metals.

"Table salt," the first use that comes to mind when salt is mentioned, consumes less than 3 percent of the total.

Table 1 shows salt sold or used by producers in the United States, 1950–52, classified according to markets.

The abundance of salt and its relatively low cost discourage the use of substitutes; however, cost or specific performance has placed other materials in competition with salt in a few uses.

The output of salt is closely geared to current demand. Most large consumers pump brine only as it is needed in manufacturing various chemicals. Some underground mines and solar-evaporation plants accumulate substantial stocks to serve their customers adequately, but, in general, salt is so readily available it is unnecessary to maintain large industry stocks. Imports and exports are comparatively small, and it is assumed that domestic production closely approximates consumption.

Coproducts or byproducts have some effect on the rate of salt production. Natural brines, sea water, and occasionally rock salt contain various other salts of commercial value. In New Mexico potash minerals are associated with bedded salt deposits. Bromine, calcium chloride, and magnesium compounds are recovered from the salt brines at various locations.

PRICES

The value ascribed to salt at different operations is variable. In brine form salt usually is consumed by the producer, and the bookkeeping value is very low—usually less than $1 per ton. Rock salt and evaporated salt are priced higher. The average value, f. o. b. plant, reported for bulk rock-salt output in 1951 was $5.06 and for evaporated, $9.53. Prices of packaged salt are much higher.

TARIFF

A tariff first was placed on salt in 1789. It has been removed and reimposed several times. The present duty has been in effect since the Geneva conference of 1948, when the tariff was established at 2 cents a 100 pounds for bulk salt and 3½ cents a 100 pounds for bagged or packaged salt.

RESEARCH

Research on products and production methods is done by individual companies. Usually these investigations are directed toward solving individual problems facing a firm. The Salt Producers Association sponsors projects on salt utilization.

As a vital raw material the importance of salt to a defense program is obvious. Fortunately its abundance and widespread occurrence tend to assure adequate supplies for all strategic requirements.

OUTLOOK

Since salt is such an important basic raw material, it may be expected to follow closely such trends as general industrial activity and population increase.

In forecasting requirements for salt it might be assumed that they will follow the trend of the gross national product. If this assumption is accepted, demand for salt may be expected to increase substantially in the next 10 years.

Demands on industry for products and services required for the defense effort and for an expanding civilian economy have necessitated record production goals for this raw material in the past few years. In 1951 consumption of salt in manufacturing soda ash and chlorine chemicals exceeded total output of salt for any year before 1942. Further increase in production and consumption is expected.

Increases in most present uses of salt are anticipated, and it is safe to assume that salt in some way will be used in the many new products continually being developed.

Large known salt deposits are undeveloped, and others are being found in drilling. When demand warrants, these deposits may be developed.

The United States Department of the Interior is conducting a program of research on the desalting of sea or brackish water. Industry also is working on this problem. If economically feasible and if practiced on a large scale, prodigious quantities of sodium chloride and other materials might be produced as a result; this might have a major impact on the salt industry.

PROBLEMS

Overall domestic reserves are ample, although certain areas do not have adequate local deposits. Exploration for commercial deposits in some of these areas would be desirable. The need for developing new uses for salt, products of salt, and byproducts is an important problem. The salt industry is considering the problem of how it will participate and be affected if desalting sea water becomes a major commercial reality. There is a need for a vigorous technical research program and development and maintenance of a body of adequate statistical and economic data to guide both Government and management of this industry.

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SAND AND GRAVEL

By

J. R. Thoenen[1] and Oliver Bowles[2]

THE SAND AND GRAVEL INDUSTRY at the beginning of the 20th century was one of the small mining industries, with a recorded annual production of about 1½ million tons valued at a little over 1 million dollars. In the past half century it has grown to a point where it exceeds all other nonfuel minerals in tonnage produced and ranks among the highest in dollar value. Production in 1952 exceeded 435 million tons valued at over 353 million dollars.

Although a low-priced material, the stringency and comprehensiveness of the specifications its products must meet and the problems in meeting them challenge the best attention and skill of mining engineers and ore dressers.

Summary

The sand and gravel industry is widespread throughout the United States. Sand and gravel have become as essential to the national economy in peace or war as the construction industry itself.

The industry is versatile in that it embraces both small (1- or 2-man operations) and huge corporations with 20 or more large producing plants. It is largely mechanized and produces nearly 5 short tons of marketable material for each man-hour worked or 40 tons per day per employee.

About 91 percent of the production enters the construction industry as fine or coarse aggregate, involving cement, bitumen, or other type binders, or as road metal without binder. The remaining 9 percent is marketed for molding, glass, engine, and abrasive sand, ballast sand and gravel, or other minor uses.

The bulk of the production is marketed at prices under $1 per ton; hence, it is essentially a local industry depending on nearby rather than national markets. Foreign trade is negligible and represents principally small tonnages produced near and sold across national borders. Small tonnages of sand for special purposes commanding relatively high prices are imported from trans-ocean sources. Exports are not recorded.

The stability and progress of the industry are proved by the facts that the number of plants, the average production per plant, and the consumption of sand and gravel per capita in continental United States have all shown an upward trend for many years. Numerous economic and technical problems face the sand and gravel industry.

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[1] Chief, Bureau of Mines, Field Office, Knoxville, Tenn., Division of Mineral Industries, Region V.
[2] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

HISTORY, DISTRIBUTION, AND ORGANIZATION

Sand and gravel have been used by man for various purposes throughout historic time and probably much earlier. With the advent of lime, sand became important as a mortar mix. With the introduction of roman cement, both sand and gravel increased in utility. However, the invention of portland cement and steel-reinforced concrete brought the use of sand and gravel into its present prominence (*4*).[3]

Table 1 illustrates the wide distribution of the industry among the several States and Territories for 1952. This table includes production for commercial use and also a very considerable tonnage produced by municipalities, counties, States, the Federal Government, and railroads for their own use. A comparison of the production in these two categories is shown in figure 1.

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[3] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

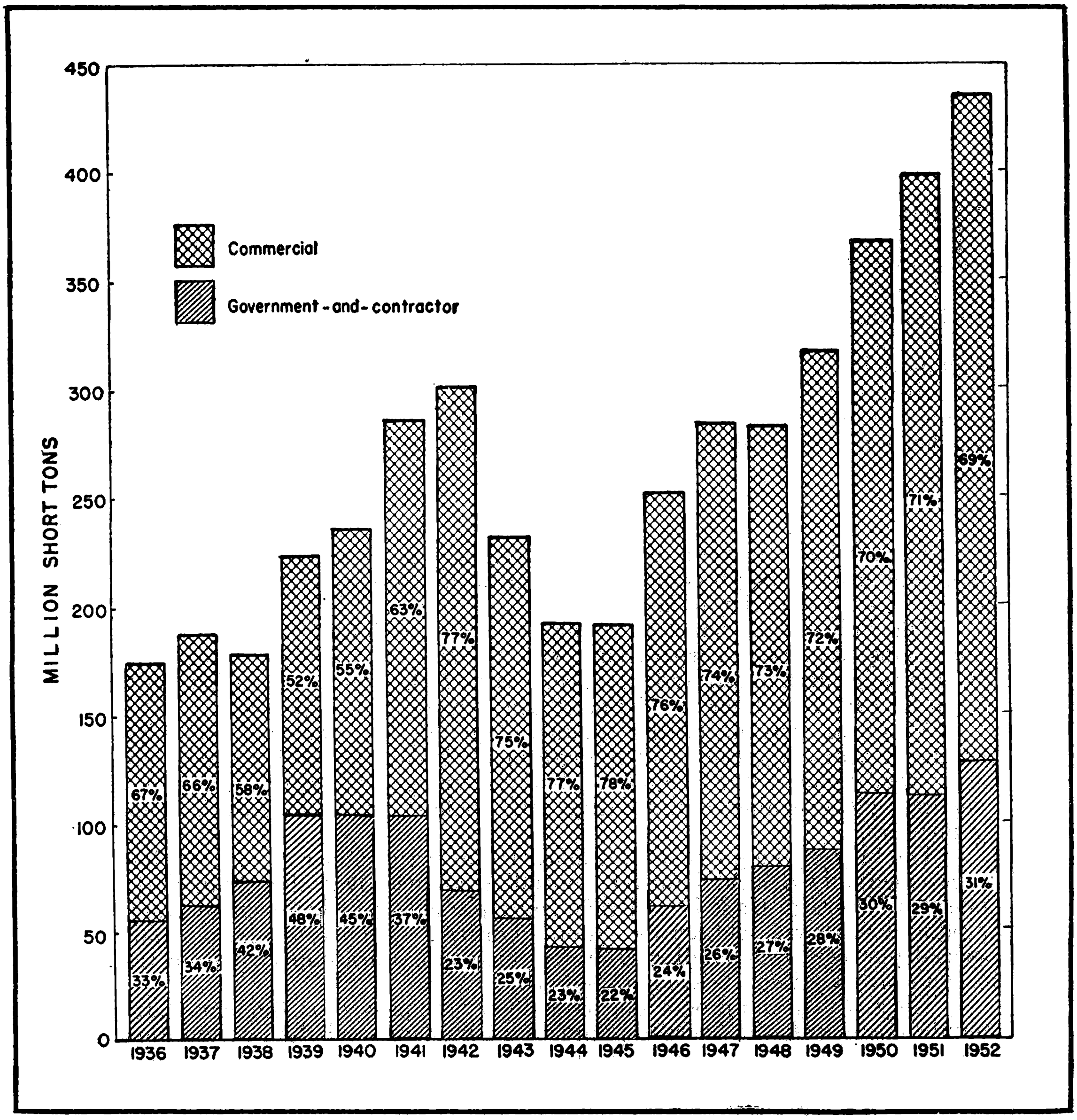

FIGURE 1.—Sand and Gravel Sold or Used in the United States by Commercial and Government-and-Contractor Producers, 1936–52.

TABLE 1.—*Sand and gravel sold or used by producers in the United States in 1952, by States*

State	Short tons	Value	State	Short tons	Value
Alabama	3, 722, 555	$2, 955, 630	Nevada	2, 098, 211	$2, 380, 419
Alaska	10, 781, 926	8, 650, 582	New Hampshire	3, 200, 232	1, 001, 591
Arizona	1, 824, 330	1, 635, 903	New Jersey	7, 060, 074	9, 473, 428
Arkansas	5, 011, 095	4, 977, 219	New Mexico	496, 921	499, 589
California	53, 051, 260	43, 633, 125	New York	20, 270, 058	18, 287, 623
Colorado	8, 461, 039	6, 268, 367	North Carolina	8, 724, 748	5, 665, 169
Connecticut	2, 581, 247	1, 933, 214	North Dakota	6, 557, 069	1, 841, 216
Delaware	515, 399	382, 484	Ohio	20, 751, 493	23, 069, 458
Florida	4, 154, 613	3, 848, 077	Oklahoma	3, 769, 663	2, 911, 845
Georgia	2, 133, 970	2, 029, 367	Oregon	12, 219, 486	8, 556, 218
Hawaii	1, 069	936	Pennsylvania	14, 696, 106	19, 920, 003
Idaho	3, 925, 863	2, 745, 201	Puerto Rico	122, 730	164, 166
Illinois	19, 584, 308	19, 214, 195	Rhode Island	589, 451	557, 396
Indiana	11, 546, 014	9, 279, 908	South Carolina	1, 048, 099	892, 312
Iowa	10, 796, 979	6, 032, 898	South Dakota	5, 846, 140	2, 478, 314
Kansas	8, 380, 065	5, 023, 593	Tennessee	5, 173, 401	5, 303, 321
Kentucky	3, 334, 261	2, 656, 053	Texas	18, 661, 403	17, 275, 255
Louisiana	6, 005, 119	6, 736, 524	Utah	3, 260, 044	2, 350, 412
Maine	7, 078, 078	2, 187, 531	Vermont	1, 264, 490	749, 835
Maryland	6, 956, 640	8, 136, 697	Virginia	7, 136, 112	5, 556, 953
Massachusetts	7, 645, 728	6, 128, 744	Washington	13, 322, 279	9, 422, 117
Michigan	29, 193, 763	22, 400, 879	West Virginia	4, 120, 105	7, 275, 370
Minnesota	19, 825, 157	6, 808, 763	Wisconsin	24, 895, 947	16, 938, 228
Mississippi	2, 296, 577	1, 833, 306	Wyoming	2, 426, 999	1, 738, 548
Missouri	6, 790, 422	6, 122, 195			
Montana	6, 765, 955	3, 579, 932	Total	435, 511, 233	353, 384, 215
Nebraska	5, 436, 540	3, 874, 106			

In table 2 the production has been broken down for both commercial operators and the Government-and-contractor groups by uses. This shows that for 1952 about 85 percent of the sand and 90 percent of the gravel produced by commercial companies were used as aggregate for buildings or paving. All sand and gravel produced at Government-and-contractor operations were used in the building and paving industries. Sand sold for other uses is indicated in the table.

Table 3 shows the distribution of production by size of commercial plants, exclusive of that produced for their own use by Government agencies and railroads.

The National Sand and Gravel Association, the largest trade association of the industry and the only one of national scope, has about 260 active members (representing 65 to 75 percent of the total national commercial production of sand and gravel) and about 100 associate members (representing companies manufacturing equipment and machinery used in producing and processing sand and gravel).

TECHNOLOGY

DEFINITIONS

An accurate definition of either sand or gravel presents some technical difficulties. A search of the literature uncovers a surprisingly wide variation among authorities, but the consensus would seem to be that sand is the unconsolidated granular material, coarser than 200-mesh and finer than ¼ inch, resulting from natural disintegration of rocks; and that gravel is the similar unconsolidated granular material, coarser than ¼ inch and finer than 3½ inch, resulting from the same causes. However, some authorities consider the upper size limit for gravel as 6 inches or more.

Both may be composed of materials in which a single mineral predominates or of a mixture of many minerals, depending upon the composition of the original rocks. Residual deposits and glacial drift often contain appreciable quantities of deleterious materials, relatively soft and nondurable, such as shales, ochers, argillaceous sandstones and limestones, feldspars, etc. By far the largest proportion of gravels, however, is composed of the harder or more resistant minerals, such as quartz and various silicates, while sands as a rule are predominantly silica, represented by quartz particles.

Residual sand and gravel particles are sharp-edged or angular, whereas transported gravels and sands consist of subangular to well-rounded or water-worn particles. Glacial gravels may be either angular or rounded, and individual particles may show abrasion scratches. Color varies due to the mineral ingredients. Loose, dry sand and gravel range in weight from 90 to 100 pounds per cubic foot.

TABLE 2.—*Sand and gravel sold or used by producers in the United States,*[1] *1951–52, by class of operation and use*

	1951			1952			Change (percent)	
	Short tons	Value: Total	Value: Average	Short tons	Value: Total	Value: Average	Tonnage	Average value
COMMERCIAL OPERATIONS								
Sand:								
Glass	5,515,588	$14,412,339	$2.61	5,227,927	$13,918,171	$2.66	−5	+2
Molding	9,107,003	16,823,440	1.85	8,253,167	16,252,433	1.97	−9	+6
Building	71,503,981	62,168,319	.87	73,660,508	63,670,537	.86	+3	−1
Paving	40,789,625	34,531,591	.85	43,666,274	36,746,584	.84	+7	−1
Grinding and polishing[2]	1,476,912	3,111,649	2.11	1,229,794	2,920,088	2.37	−17	+12
Fire or furnace	471,540	791,682	1.68	413,789	819,908	1.98	−12	+18
Engine	2,208,903	2,339,753	1.06	1,900,621	1,939,025	1.02	−14	−4
Filter	202,739	484,724	2.39	288,207	606,501	2.10	+42	−12
Railroad ballast [3]	1,087,669	604,841	.56	828,750	404,669	.49	−24	−12
Other [4]	2,793,226	3,102,669	1.11	4,037,053	4,063,914	1.01	+45	−9
Total commercial sand	135,157,186	138,371,007	1.02	139,506,090	141,341,830	1.01	+3	−1
Gravel:								
Building	62,550,990	65,648,107	1.05	64,263,744	68,212,707	1.06	+3	+1
Paving	72,335,348	65,944,983	.91	81,652,021	74,166,945	.91	+13	------
Railroad ballast [5]	11,362,531	7,032,599	.62	10,669,141	6,487,822	.61	−6	−2
Other [6]	4,413,449	3,180,659	.72	5,637,498	4,930,002	.87	+28	+21
Total commercial gravel	150,662,318	141,806,348	.94	162,222,404	153,797,476	.95	+8	+1
Total commercial sand and gravel	285,819,504	280,177,355	.98	301,728,494	295,139,306	.98	+6	------
GOVERNMENT-AND-CONTRACTOR OPERATIONS [7]								
Sand:								
Building	[8] 1,869,483	[8] 2,001,392	[8] 1.07	1,183,968	1,140,413	.96	−37	−10
Paving	[8] 12,563,827	[8] 4,775,708	.38	15,402,448	6,229,943	.40	+23	+5
Total Government-and-contractor sand	[8] 14,433,310	[8] 6,777,100	[8] .47	16,586,416	7,370,356	.44	+15	−6
Gravel:								
Building	7,664,694	6,905,832	.90	3,561,751	2,857,283	.80	−54	−11
Paving	[8] 92,716,945	[8] 39,854,062	.43	113,634,572	48,017,270	.42	+23	−2
Total Government-and-contractor gravel	[8] 100,381,639	[8] 46,759,894	.47	117,196,323	50,874,553	.43	+17	−9
Total Government-and-contractor sand and gravel	[8] 114,814,949	[8] 53,536,994	[8] .47	133,782,739	58,244,909	.44	+17	−6
COMMERCIAL AND GOVERNMENT-AND-CONTRACTOR OPERATIONS								
Sand	[8] 149,590,496	[8] 145,148,107	.97	156,092,506	148,712,186	.95	+4	−2
Gravel	[8] 251,043,957	[8] 188,566,242	.75	279,418,727	204,672,029	.73	+11	−3
Grand total	[8] 400,634,453	[8] 333,714,349	.83	435,511,233	353,384,215	.81	+9	−2

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[1] Includes Alaska, Hawaii, and Puerto Rico.

[2] Includes blast sand as follows—1951, 549,955 tons valued at $1,875,775; 1952, 557,305 tons, $2,016,747.

[3] Includes ballast sand produced by railroads for their own use as follows—1951, 140,111 tons valued at $17,745; 1952, 204,358 tons, $41,848.

[4] Includes some sand used by railroads for fills and similar purposes as follows—1951, 263,997 tons valued at $78,686; 1952, 208,591 tons, $64,199.

[5] Includes ballast gravel produced by railroads for their own use as follows—1951, 4,100,872 tons valued at $1,709,860; 1952, 4,867,003 tons, $2,214,808.

[6] Includes some gravel used by railroads for fills and similar purposes as follows—1951, 904,402 tons valued at $244,119; 1952, 1,623,166 tons, $794,405.

[7] Approximate figures for States, counties, municipalities, and other Government agencies directly or under lease.

[8] Revised figure.

TABLE 3.—*Comparison of number and production of commercial sand and gravel plants in the United States, 1951–52, by size groups* [1]

Size groups (short tons)	1951				1952			
	Plants [2]		Production		Plants [2]		Production	
	Number	Percent of total	Thousand short tons	Percent of total	Number	Percent of total	Thousand short tons	Percent of total
Less than 25,000	759	32. 5	7, 604	2. 7	824	32. 9	8, 356	2. 8
25,000 to less than 50,000	381	16. 3	13, 734	4. 9	429	17. 1	15, 644	5. 3
50,000 to less than 100,000	435	18. 6	31, 006	11. 1	438	17. 5	31, 188	10. 6
100,000 to less than 200,000	394	16. 8	56, 318	20. 1	405	16. 2	56, 760	19. 3
200,000 to less than 300,000	146	6. 2	35, 538	12. 7	179	7. 1	43, 562	14. 8
300,000 to less than 400,000	81	3. 5	27, 694	9. 9	82	3. 3	27, 656	9. 4
400,000 to less than 500,000	49	2. 1	21, 528	7. 7	40	1. 6	17, 799	6. 0
500,000 to less than 600,000	30	1. 3	16, 423	5. 8	29	1. 2	15, 594	5. 3
600,000 to less than 700,000	17	. 7	11, 016	3. 9	24	1. 0	15, 668	5. 3
700,000 to less than 800,000	10	. 4	7, 374	2. 6	17	. 7	12, 652	4. 3
800,000 to less than 900,000	5	. 2	4, 291	1. 5	5	. 2	4, 233	1. 4
900,000 to less than 1,000,000	4	. 2	3, 836	1. 4	2	. 1	1, 921	. 7
1,000,000 and over	27	1. 2	43, 953	15. 7	27	1. 1	43, 688	14. 8
Total	2, 338	100. 0	280, 315	100. 0	2, 501	100. 0	294, 721	100. 0

[1] Excludes operations by or for States, counties, municipalities, and Federal Government agencies as follows—1951: 750 operations with an output of 115,801,000 tons of sand and gravel; 1952: 806 operations, 134,788,739 tons. Excludes operations by or for railroads as follows—1951: 131 operations with an output of 5,504,000 tons of sand and gravel; 1952: 176 operations, 7,007,033 tons. Includes Alaska.

[2] Includes a few companies operating more than 1 plant but not submitting separate returns for individual plants.

GEOLOGY

Roughly, sand and gravel deposits may be classified in four groups with reference to their method of formation as follows:

(1) Residual.
(2) Fluvial.
(3) Marine and lake.
(4) Glacial.

Residual deposits are commonly unstratified, heterogeneous mixtures of boulders, pebbles, sand, and clay. They may contain much soft or inferior material, which through prolonged weathering has become porous and weak in structure. The particles may be angular or rounded but usually have rough or granular surfaces. Residual deposits commonly form a rock mantle over the parent formation and are formed by the weathering, in place, of the parent or superimposed formations. Ordinarily they are so intermixed with clay and inferior particles as to have minor commercial importance.

Sand and gravel picked up, transported, and deposited by stream action form fluvial deposits. Such deposits may be imperfectly stratified and frequently show size gradation. Coarse sand and gravel may be interspersed with lenses of fine sand or clay. Deposits have their long axis parallel to the direction of the stream flow. The beds usually vary greatly in thickness and may be complex in composition. As a rule, particles are poorly sorted and may be angular or rounded, depending on hardness and the distance transported. When rounded, the surfaces are typically smooth.

Fluvial deposits occur in a variety of structural forms and usually overlie an eroded rock floor but may occur above finer materials. Present rivers are constantly constructing, tearing down, and reworking bars within their banks. During periods of high water, rivers leave their banks and extend over wide flood plains, leaving considerable material behind as they recede. At their mouths they deposit their loads in delta formations. Swift mountain streams emerging from narrow canyons or gorges onto broad, flat plains deposit their material in alluvial fans or cones. Old streams in wide valleys leave terraces behind as they cut their way down to lower levels. These are all typical fluvial deposits.

Marine and lake deposits frequently consist of well-sorted materials with coarse and fine particles segregated. The various segregations usually have their long axis parallel to the shore line. Individual particles of marine sands range from angular to well-rounded, while pebbles are usually well-rounded and smooth. Marine deposits ordinarily consist of hard, tough materials received from streams and reworked by wave, tidal, and marine-current action. Occasionally pebbles of limestone of organic origin, such as corals, may be present. Sand and gravel collected in lake beds present structures similar to marine deposits, except that the materials are not usually so well sorted and cleaned, and they are also less apt to be well rounded.

Glacial deposits are confined to areas that have been covered by the great ice sheets of the past. The ice originating about Hudson Bay, where igneous rocks predominate, brought down huge quantities of boulders, forming sand and gravel of igneous origin now scattered over much of the northern United States. The ice also scoured out valleys in sedimentary rocks enroute and carried along vast quantities of this material. Therefore, glacial materials are composed of a mixture of hard and soft rocks.

MINING

The mining of sand and gravel is essentially an earth-moving industry, using all the equipment associated with projects involving the removal of obstructing hills or the filling of valleys to provide level surfaces for airports or city construction. The analogy may be carried farther by reference to the removal of underwater material in the excavation of ship channels in rivers, and lake and marine bars. In fact, there is even a more direct comparison to alluvial gold mining from river bars by ladder dredges.

The great variety of deposits necessitates a diversity of mining machinery. Dry-land operations employ power shovels, draglines, caterpillar (crawler)-mounted cranes, slackline cableway excavators, cable-drawn power scrapers, motorized self-loading tractor scrapers, and bulldozers.

Excavation may be from the bottom of the pit, as with power shovels and motorized scrapers, or from the bank, as with draglines, cableways, and cranes. The choice between gasoline, diesel, or electric power depends upon factors that may vary from plant to plant.

Where the water table is close to the surface, mining may be by shovel to the water and then by some other unit below that elevation; by dragline, power scraper, or slackline cableway from the surface of the deposit to the desired underwater depth; or by hydraulic, ladder, or dipper dredges. In addition, many varied combinations of equipment may be and are used.

The preceding discussion has dealt with methods and equipment used by the large producers. As noted earlier in this report, a large percentage of sand and gravel production comes from small producers. Designed for such use are many ½- to 2-cubic-yard-capacity gasoline- or diesel-driven excavators that have recently come on the market. There is a wide field for study in the new small excavators and truck units now available for the small operator.

Haulage from pit to treatment plant is as varied as is the excavating equipment. Dry pits utilize motor trucks, locomotives and cars, carryall excavators, bulldozers, and belt conveyors. The last mentioned may be permanent installations, portable, or combinations of portable and permanent.

In wet pits, if a dragline is used it may transport the excavated material to dry land, depositing it either into the washing plant, a transfer bin, or in trucks or other transportation equipment. The power scraper and slackline cableway also convey from wet pit to shore with delivery similar to the dragline; or one cableway may deliver to another, which takes material to the washer.

Depending on their type, dredges deliver to barges, pipelines or to washing plants built integral with the dredge hull.

Some dredges are designed and built for excavating from lake or ocean bars, washing and screening the product and then carrying the cargo to a shore point and discharging it with self-unloading equipment to shore yards or further processing plant.

From the above it can be seen that the mining of sand and gravel is a wide-flung, versatile business involving all types of dry-land and marine excavation methods (*8*).

MILLING

In the early days of sand and gravel mining most of the material was marketed as "bank run," without any further processing. However, with the advent of scientifically planned concrete, requirements became progressively more severe for clean-washed products, sized to strict specifications either in single sizes with top and bottom limits or in various combinations of sizes, each with its top and bottom limits, and the whole within rather strict fineness modulus limits.

To meet these requirements, the sand and gravel producer has had to become a mineral dresser of no mean magnitude, employing virtually all the tools and equipment of ore-dressing engineers on a large scale. These include bucket elevators, screw and belt conveyors, trommel and log washers, revolving (trommel) and vibrating screens, crushers of all types, disintegrators, cone, screw and rake classifiers, scrubbers, dewaterers, settling tanks and basins, stockpiling and recovery systems, pumps and pipelines, aerial trams, and a host of minor tools. Ball and rod mills, jigs, tables and heavy-medium separation are used where special problems are encountered.

Preparation of the material for market may be a complex process. The operator commonly starts with a contaminated, heterogeneous mass of sand, gravel, and boulders that not only must be converted into a clean material but one prepared in various sizes within close limitations.

To prepare products that will meet current specifications most sand and gravel plants must have washing and screening equipment, and generally they must have crushers. Crushers range from huge gyratories, as used in western pits, to no crushers at all in some deposits with material well-sized by nature and with no coarse sizes. Washing equipment ranges from mere rinsing to remove fine sand to heavy log and trommel washers necessary to break up and remove heavy, sticky clay.

Sizing by wet screening is common practice. Dry screening often involves silicosis or other dust problems. Screen cloth is made in a multiplicity of design and aperture sizes, shapes, and wire diameter. Screens also are made of perforated plate.

BYPRODUCTS

The gravels themselves do not ordinarily afford much in the way of byproducts. However, a number of plants have installed equipment for recovering gold, as well as the heavy "black" sands containing ilmenite, rutile, monazite, columbite, tantalite, zircon, and other rare constituents that formerly stayed in the bulk product or went to waste piles. More and more attention is being paid to the recovery of these byproduct minerals.

SUBSTITUTES

Crushed stone and blast-furnace slag are highly competitive substitutes for gravel. Expanded clay or slag is frequently used for coarse aggregate where light weight is a requisite, and pumice is occasionally employed for both light-weight and high-temperature requirements.

Crushed stone or gravel properly ground and sized is used as a supplement for sand. Some specifications permit stone crushed to sand sizes in place of sand. Limestone, granite, chert, and sandstone have been used for this purpose.

SYNTHETICS

In areas where suitable clays or shales are plentiful they may be burned and used as coarse aggregate. Where light weight is a requisite, a rather wide choice of expanded materials is available for both coarse and fine aggregate.

BYPRODUCT SOURCES

Tailings from ore milling and slag from smelters are in some instances used as sources of both coarse and fine aggregate.

RESERVES

United States reserves of sand and gravel are large but unevenly distributed; moreover, sand and gravel of the quality required by present specifications are definitely limited in many areas. The markets are restricted to areas that they can reach in competition with other materials. The expense of transportation from plant to point of use is therefore a limiting factor in determining available commercial reserves. A future shortage of available reserves in some metropolitan areas is forecast by several producers.

Sand and gravel deposits of suitable quality within a short transportation distance of large construction areas are becoming depleted. River and lake bars contiguous to construction areas are also becoming increasingly contaminated. This results from working and reworking the same bars. Not all of the suitable material is extracted during the first operation. The waste returned to the water accumulates and becomes a larger percentage of the material to be worked on the second pass.

USES

The principal uses for sand and gravel are as fine and coarse aggregate for concrete for construction of buildings, bridges, dams, highways, runways, and similar structures and for concrete blocks, pipe, and other concrete products. Large quantities are used without admixtures for subgrade or bases for highways and airports. Significant quantities are used also for township and county secondary or tertiary roads, with or without admixtures, and for railroad ballast.

Metal industries consume substantial quantities annually as molding sands. Similar but smaller tonnages of sand are consumed in making glass of all types. Railroads consume large quantities to increase brake friction on slippery or wet rails.

Municipalities require considerable quantities of both sand and gravel for water filters in purification of city water supplies.

Smaller quantities of special types of sands are consumed in sand blasting, grinding, and polishing.

PRODUCTION AND CONSUMPTION

Tables 1 and 2 present general data on production. More detailed statistical data are given each year in the chapter on Sand and Gravel in the Minerals Yearbook, published by the Bureau of Mines. Consumption is regarded as synonymous with production.

PRICES AND COSTS

The values for the various grades of sand and gravel as listed in table 2 are computed as the value at the producer's plant. The average value of commercial output in 1952 was 98 cents per ton, whereas the noncommercial average was 44 cents. The price the consumer must pay will include the cost of transportation from plant to consumer. Prices at the plant, including both commercial and noncommercial, have increased from about 40 cents a ton in 1905 to 81 cents in 1952. This means an increase in 45 years of about 100 percent, which is considerably less than the increases for building materials in general. The reason for this undoubtedly lies in the fact that, during that 45-year period, the sand and gravel industry has emerged from a manual operation to a highly mechanized one.

Sand and gravel are low-valued materials. As such they cannot tolerate relatively high transportation costs over long distances. As a result, markets must be found within economic shipping distances. Costs at individual plants depend upon many variables, such as efficiency of management, cleanliness, quality and gradation of the natural product, suitability of the natural bank-run ratio of sand to gravel for the market supplied, depth of overburden, type of mechanization used and its efficiency with respect to the deposit, the degree and expense of processing required to meet market specifications, the sales ability of the producer, and the percentage of capacity utilized. The margin of profit will vary with all of these factors; but the overall average margin for all producers is small—perhaps in the range of 8 to 10 percent of the plant values.

TRANSPORTATION

A marked trend toward increasing truck transportation has been in evidence for some years. In 1936 railroads handled 51 percent of the sand and gravel shipments, trucks 37 percent, and waterways 12 percent. By 1952 truck transport had increased to 71 percent, while railroads carried 20 percent and waterways 6, while 3 percent was unspecified.

CONSERVATION

So far conservation of reserves has not been a pressing problem except in some metropolitan areas where, as pointed out in the discussion of reserves, depletion is threatened. To conserve available supplies in such areas, increasing depths of overburden are being removed, and lower grade deposits are being worked.

In a sense, conservation has been practiced by operators so located, in that they have procured portable plants and set them up on deposits in the field in closer proximity to highway or other contracts that they would otherwise have supplied from their city plant. This might be construed as conservation of the city deposit for city markets.

RESEARCH

The National Sand and Gravel Association has maintained laboratory research since 1928. Since 1937 the work has been carried on at the University of Maryland. The research conducted at this laboratory is directed primarily at problems concerning the use of sand and gravel in cement and bituminous concretes, as well as for other purposes (*9*). The association laboratory provides answers insofar as possible to its members in reaching solutions of technical problems.

OUTLOOK

SIGNIFICANT TRENDS

There does not appear to be any reason to doubt that the sand and gravel industry will continue to expand in line with the expansion of total national construction. As deposits are depleted and competitive materials enter the market, sand and gravel output may decline locally, but the development of new sources and new markets probably will more than counterbalance such losses on a national basis. Some trends tend to accelerate and others to retard the use of sand and gravel, but it is not anticipated that the net effect of these will alter the overall situation significantly. For example, in constructing the so-called "flexible" types of pavement for airport runways, crushed stone is to a considerable extent being displaced by sand and crushed gravel. On the other hand, the recent rapid increase in the use of lightweight aggregates, whether of natural or expanded materials, has caused a relatively smaller use of sand in the concrete block industry.

With respect to transportation of sand and gravel, current trends indicate that railroads will continue to lose tonnage to trucks; however, the trend toward removal of sand and gravel operations to locations farther from metropolitan centers may give railroads an opportunity to win back some of their lost haulage. It is doubtful if the percentage handled by water transportation will change materially.

In 1927 the Minerals Yearbook (*2*) presented the first table showing the number of plants and production in various size groups. A study of these annual tables from 1931 to 1952 shows some interesting trends. Table 4 shows the tabulations for 1931 and 1952 for direct comparison.

Plants shown in these tables are those designated as "commercial," or producing for sale. Plants operated by contractors, railroads, and municipalities are not included. An important trend shown in table 4 is toward larger unit plants; and this trend, which appears to be continuing, has been accentuated in those plants producing less than 400,000 tons annually. Other trends will appear with a study of this table.

NEW SOURCES IN PROSPECT

As new building, public works, or highway-construction projects are undertaken new sand and gravel deposits will be developed as close as possible to the projects if suitable materials are available. One reason for their opening will be to reduce delivery expense, because the haul would be shorter than in obtaining material from more remote existing plants. Conveniently located new deposits are available in certain areas, in other regions satisfactory undeveloped deposits are scarce, and a long haul may be unavoidable. In some instances material from a new deposit may be accepted, even though the quality is below that of the products derived from established plants.

With depletion of deposits, encroachment of residential areas, and increasing stringency of specifications, operators of metropolitan plants have had to consider more adequate and accurate exploration for new deposits. In the course of this study, they have shown increasing interest in the use of standard tools for such exploration, as well as the design of new and better tools.

NEW USES IN PROSPECT

The design of pavements for runways at airports, formerly directed toward heavy, rigid, concrete construction, has gradually shifted to the so-called flexible-type pavement utilizing bitumens as binders. Specifications for the flexible-type pavement originally called for crushed-stone aggregate, but recently sand and gravel have been included, offering a potentially large new market for those materials; however, a preference for heavy concrete for jet-plane runways has been reported.

In northern climates where prevention of heaving of airport runways caused by freezing

TABLE 4.—*Number of commercial sand and gravel plants and production, by size groups, 1931 and 1952*

Size group (short tons) (annual)	1931						1952					
	Plants			Production			Plants			Production		
	Number	Percent of total	Cumulative percent	Thousand short tons	Percent of total	Cumulative percent	Number	Percent of total	Cumulative percent	Thousand short tons	Percent of total	Cumulative percent
Less than 25,000	1, 277	59. 1	59. 1	9, 518	7. 4	7. 4	824	32. 9	32. 9	8, 356	2. 8	2. 8
25,000–49,999	295	13. 6	72. 7	10, 875	8. 5	15. 9	429	17. 1	50. 0	15, 644	5. 3	8. 1
50,000–99,999	297	13. 7	86. 4	20, 969	16. 4	32. 3	438	17. 5	67. 5	31, 188	10. 6	18. 7
100,000–199,999	166	7. 7	94. 1	23, 939	18. 7	51. 0	405	16. 2	83. 7	56, 760	19. 3	38. 0
200,000–299,999	49	2. 3	96. 4	12, 009	9. 4	60. 4	179	7. 1	90. 8	43, 562	14. 8	52. 8
300,000–399,999	24	1. 1	97. 5	8, 130	6. 3	66. 7	82	3. 3	94. 1	27, 656	9. 4	62. 2
400,000–499,999	16	. 7	98. 2	7, 302	5. 7	72. 4	40	1. 6	95. 7	17, 799	6. 0	68. 2
500,000–599,999	11	. 5	98. 7	5, 865	4. 6	77. 0	29	1. 2	96. 9	15, 594	5. 3	73. 5
600,000–699,999	4	. 2	98. 9	2, 649	2. 1	79. 1	24	1. 0	97. 9	15, 668	5. 3	78. 8
700,000–799,999	5	. 3	99. 2	3, 795	2. 9	82. 0	17	. 7	98. 6	12, 652	4. 3	83. 1
800,000 and over	17	. 8	100. 0	23, 035	18. 0	100. 0	34	1. 4	100. 0	49, 842	16. 9	100. 0
	2, 161	------	------	128, 086	------	------	2, 501	------	------	294, 721	------	------

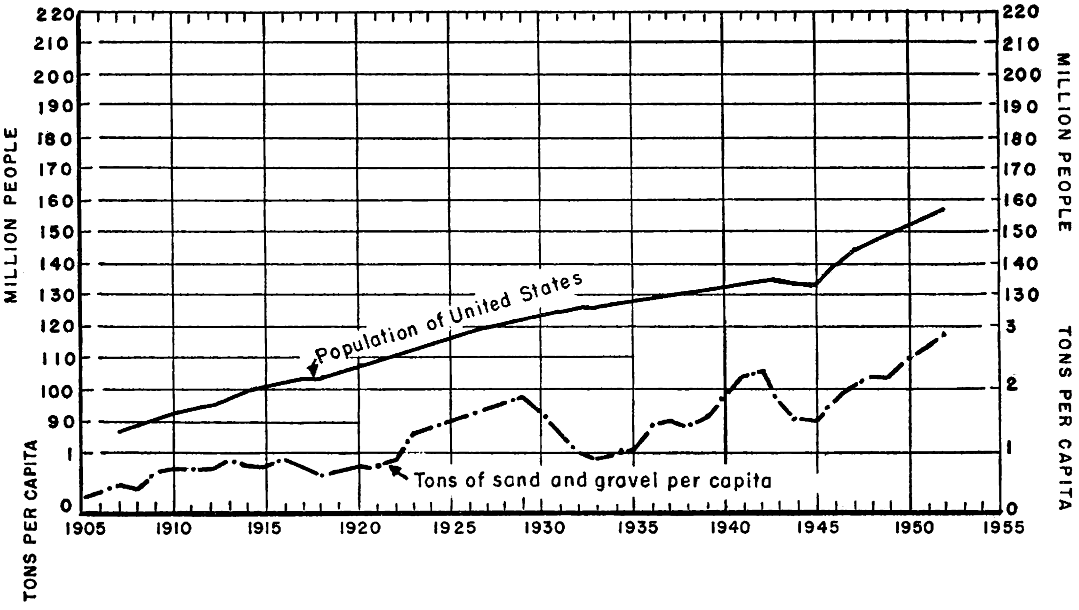

Figure 2.—Comparison of the Growth of Population in the United States With the Growth of Use of Sand and Gravel per Capita.

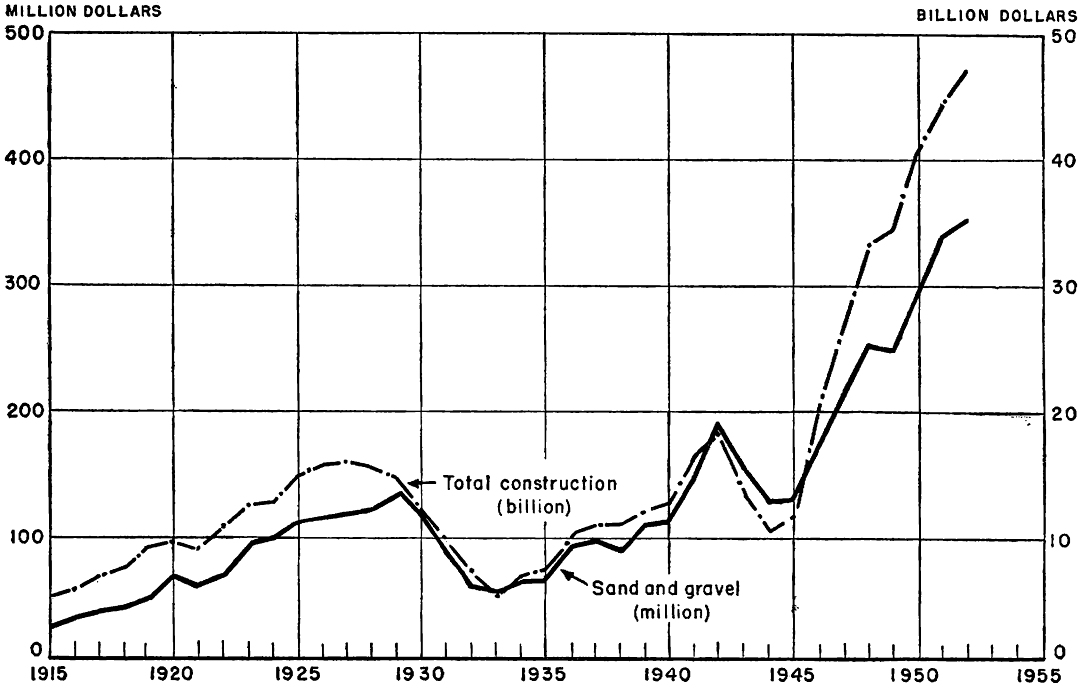

Figure 3.—Sand and Gravel Production Compared with Total Building Construction, 1915–52. (Total construction value from Dept. of Commerce.)

and thawing is serious, recourse has been had to deep subbases of sand and gravel or crushed stone. These bases may extend to depths of 6 feet or more and will require large tonnages of materials as more and more northern airports are opened.

On the other hand, the use of cinders, other lightweight, porous aggregates, and the many different types of expanded mineral products has made large inroads into the use of sand and gravel for making concrete blocks and other structural shapes.

FUTURE TRENDS

Production of sand and gravel follows the trend of population growth. This is indicated in figure 2, which compares the population of the United States with per capita consumption of sand and gravel; however, a closer relationship exists between sand and gravel output and the volume of building construction. Such a relationship is to be expected because most of the sand and gravel produced is used in the construction industries. The remarkable parallelism that has existed since 1915 between the output of sand and gravel and construction activity is shown in figure 3. It appears from this figure that a future upward trend in consumption of sand and gravel may be anticipated. The fluctuations that may take place from year to year and the overall magnitude of output will probably be governed chiefly by the volume of total building construction.

PROBLEMS

The problems of the sand and gravel industry are many and varied. They fall naturally into two general classes, economic and technical. Both are important, and both effect the cost of production.

ECONOMIC PROBLEMS

Of growing concern to producers is the increasing public interest in zoning restrictions around metropolitan areas. The accelerated movement of city workers to suburban areas causes expansion of such residential sites and encroachment on established sand and gravel deposits. These were originally located and developed because of the advantage of short hauls to large metropolitan markets. Zoning of residential sections restricts producers in future expansion as deposits become depleted. More than that, as the residential area grows and finally surrounds the pit, residents become critical of the noise of plant operations and truck haul into and out of the plant. Eventually a producer on such a site may be forced to go out of business or move elsewhere.

Analogous to this is the growing demand by the public, as well as metropolitan, county, and State governments, that gravel producers rehabilitate the area from which material has been extracted. This rehabilitation is expensive and adds to operating costs. If the expense of restoration is not at least partly borne by the civic authorities in cooperation with producers, the latter have no other recourse than to raise their selling prices to compensate for the added expense.

Operators mining their material from stream beds, bars, or banks customarily use water from the stream to wash their material in preparing it for the market. This wash water, carrying the fine silt and clay removed from the sand and gravel, is wasted to the stream. There is increasing complaint against turbidity thus produced. Some land-based gravel producers have built large settling basins in which to discharge their wash water, thereby settling out the fine sand and silt and overflowing only clear water to the stream. This procedure, of course, adds to production costs.

TECHNICAL PROBLEMS

One of the most difficult technical problems confronting sand and gravel producers is the complexity and inflexibility of specifications to be met in marketing their products. Concrete aggregate is used by Federal, State, county, township, and municipal officials, as well as private contractors and individuals. Specifications promulgated by various groups may differ widely, even for identical uses. Aggregate producers have attempted to harmonize these divergent specifications and have succeeded in reducing the number greatly; however, these simplified specifications have not been accepted by all design engineers. The staffs of the trade associations of aggregates producers, major consumers, the American Society for Testing Materials, and other organizations have taken an active part in this simplification study, but it remains a big problem.

The average operator of a stationary sand and gravel plant, to meet the specifications of his various customers, must screen his products to very close size gradations. By stockpiling these sizes separately and reclaiming them in proper proportion, he can make up mixtures to meet various specifications. The need for maintaining elaborate stockpiling and reclaiming facilities is a serious problem facing some operators.

In attempting to meet specifications no two sand and gravel deposits will give the same representative overall sieve analysis, even when identical equipment is used. This means that operators must adjust their equipment to suit the feed. Much of this adjustment is being done by trial-and-error methods. There is need for research into the fundamental laws governing screening for discrete sizes from complex feeds.

To meet the exacting requirements of modern specifications operators face a multiplicity of problems relating to the proper selection and operation of mining and milling equipment. Special equipment may be required to accomplish effective separation of soft or unsound particles or undesirable reactive fragments.

Portable crushing, screening, and washing plants have become an important factor in the industry. Usually they are designed to produce materials to satisfy specific requirements at one location and are not suited for making more than a limited variety of sized materials. They have no elaborate stockpiling and reclaiming facilities, as are required at the stationary plant. This type of operation is becoming a serious competitor of long-established permanent plants. Many permanent-plant owners, to compete with transient producers, operate supplemental portable plants.

Research on the qualities of sand and gravel aggregates in relation to the strength and durability of concrete is an ever-present problem.

Transportation from mines or pits to crushing or washing plants involves many types of equipment, including standard- or narrow-gage railway cars and steam, gasoline, diesel, or electric locomotives, trucks, pumps and pipelines, conveyor belts, towboats and barges, and even (at least in one instance) an aerial tramway. The power scraper and slackline cableway may be used for such transportation also. When to use what type is an economic and technical problem that could involve considerable research.

Other problems that deserve attention concern (1) crushers, to prevent the production of flat and elongated particles; (2) conveyor belts, to determine the proper operating characteristics; (3) dewaterers; (4) wet and dry classifiers; (5) heavy-medium separation equipment for sand as well as gravel; (6) equipment for byproduct recovery; (7) equipment for recovering minus-200-mesh sand from plant waste; (8) a study of the newer types of equipment for small plants; and (9) improved exploration methods.

As may be seen from the foregoing, the problems of sand and gravel operators are many and varied; consequently, dependable technical, statistical, and economic information is needed by Government, by producers and consumers, and by many other organizations and individuals interested in sand and gravel.

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SCANDIUM

By

John D. Sargent[1]

SCANDIUM is a relatively rare and costly element. In recent years demand has developed for increasing quantities of the metal; its end use, however, has not been ascertained. Demand for scandium exceeds known supply, and new sources are being sought.

Summary

Scandium is a silvery white metal closely associated with the rare-earth metals and yttrium. The abundance of scandium in the earth's crust has been estimated to be about the same as that of beryllium, arsenic, molybdenum, tin, and uranium. It is widely distributed as an impurity in minerals, but only one mineral—thortveitite—contains scandium as an essential constituent.

Lack of high-grade scandium ores and the difficulties of extracting the metal from its ores account for the limited knowledge of its physical and chemical properties and for the retarded development of industrial applications.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

HISTORY

Scandium was discovered in 1879 by L. F. Nilson, who named the new metal in honor of the Scandinavian peninsula. Mendeleev had predicted the existence of scandium 8 years earlier and had called it eka-boron.

PHYSICAL AND CHEMICAL PROPERTIES

W. R. Mott determined 1,200° C. for the melting point and 2,400° C, for the boiling point of scandium. Scandium exists only in the +3 oxidation state in crystals and in aqueous solutions. This element forms a white oxide—Sc_2O_3—which reacts with water to give the hydroxide, which is a stronger base than aluminum hydroxide but weaker than the hydroxides of all the tripositive rare-earth elements (the basicity of scandium is greater than that of thorium and tetravalent cerium, however). Scandium salts are colorless and have a sweet, astringent taste. They are readily hydrolyzed and show a decided tendency toward complex-ion formation. Scandium, like the cerium group of rare-earth elements, forms a sparingly soluble carbonate, a phosphate, an oxalate, and a double alkali sulfate. The nitrate, sulfate, and halides of scandium are readily soluble in water.

GEOLOGY

Because scandium occurs in such minerals as cassiterite, wolframite, euxenite, monazite, eschynite, keilhauite, zinnwaldite, beryl, lepidolite, and biotite, early researchers considered granites and granitic pegmatites to be especially enriched in scandium. Later workers (Goldschmidt and Peters) found that the bulk of the scandium present in the upper lithosphere is concealed in ferromagnesium minerals of ultrabasic and basic rocks, such as pyroxenites and gabbros. Scandium is widely distributed throughout the earth's crust because of its occurrence in such common deposits as basic igneous rocks and sedimentary iron deposits.

The only scandium mineral discovered thus far is thortveitite,$(Sc, Y)_2$ (Si_2O_7), containing 42 percent Sc_2O_3. Thortveitite, the richest source of scandium, has been mined at Madagascar and Norway, and other occurrences have been reported. The mineral bazzite from Baveno, Italy, formerly reported as a scandium silicate, has recently been determined to be a scandian beryl. In addition, scandium has been detected in concentrations as high as 1 percent in beryl, biotite, wolframite, cassiterite, zinnwaldite, wiikite, monazite, and orthite. There are 5 or 6 parts of scandium in every million parts of the earth's crust.

EXTRACTION

Scandium oxide is prepared by roasting thortveitite with excess carbon at 1,800°C. The resulting carbides are dissolved with HCl to form a solution of scandium oxide that contains 10 percent rare-earth oxides. Scandium can then be separated efficiently from lanthanum and yttrium in an ion-exchange resin column.

In another method, finely powdered thortveitite is fused with an excess of sodium hydroxide. The resultant scandium and rare-earth hydroxides are leached with water and converted to nitrates by the addition of nitric acid. The several nitrates are then separated and purified.

Scandium can be separated from yttrium and the rare-earth metals by precipitating yttrium and the rare-earth fluorides from a strong ammonium bifluoride solution; the scandium remains in solution as the soluble fluoride complex ScF_6. Scandium has also been separated from associated elements by electrochromatographic methods.

Scandium has been recovered from wolframite and cassiterite concentrates by the following process: Residues from cassiterite processing, after removal of the tin, are fused with soda, and the cooled melt is extracted with water. After the residue is washed and dissolved in hydrochloric acid, the rare earths are precipitated by either oxalic or hydrofluoric acid, a large excess of acid being used to keep the iron and manganese present in solution. The scandium content of the rare-earth precipitate is about 90 to 95 percent. Scandium, in the form of the oxide, is then dissolved in hydrochloric acid and the resulting solution boiled with sodium silicofluoride; the scandium is precipitated as chemically pure scandium fluoride. Traces of the rare earths may be removed by reprecipitation with sodium thiosulfate. In this manner scandium, with thorium as its only impurity, is obtained. The thorium is removed by fractional sublimation of the anhydrous chlorides, by precipitation of the thorium as iodate, or by means of the double alkali carbonates.

Scandium metal can be prepared by electrolysis of a fused bath of scandium fluoride (ScF_3) plus sodium fluoride (NaF), using a graphite anode and additions of scandium oxide (Sc_2O_3) as a depolarizer.

RESERVES

Although scandium is a rare metal, unlimited reserves are available as a trace-element constituent of pyroxene and amphibole. Tin- and tungsten-ore reserves can also be considered to be scandium reserves.

PRODUCTION

Companies that have been purchasing scandium-bearing materials or producing scandium chemicals include Fairmont Chemical Co., Newark, N. J.; A. D. Mackay, Inc., New York, N. Y.; F. A. Lewis, Hempstead, N. Y.; Research Chemicals, Inc., Burbank, Calif.; King Products, Arlington, N. J.; De Rewal International Rare Metals Co., Philadelphia, Pa.; Pegasus International Corp., New York, N. Y.; and Rare Earths, Inc., Pompton Plains, N. J.

USES

Scandium is used for research, as an analytical standard, to intensify the blue luminescence of MgO, to cause phosphorescence in ethyl alcohol, and as a radioactive tracer.

PRICES

Thortveitite ore ranges in price from $5 to $1,200 a pound, depending on grade and current demand. Some Norwegian thortveitite was purchased in 1953 from Christiania Minekompani at a price of $2.70 per gram. Refined scandium oxide sells for $20 to $50 per gram, depending on quality and quantity. Scandium sulfate has been offered at $17.50 per gram in recent years.

OUTLOOK

An increased demand for scandium at lower prices is probable. The supply of thortveitite is unlikely to increase enough to satisfy the demand for scandium. Byproducts of the treatment of beryllium, uranium, thorium, tin, tungsten, and the rare-earth ores are likely to be the only other productive sources of scandium in the next few years. If high prices and large demand for scandium continue, large-scale treatment of pyroxene and amphibole for recovery of the metal may be attempted.

PROBLEMS

The chief problem associated with scandium is development of a better supply of raw materials and improved methods for extracting the metal from ore, so that larger quantities will be available at lower prices to meet demand.

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SELENIUM

By

John D. Sargent[1]

THE ELEMENT selenium might be described as paradoxical, being either a metal or a nonmetal, a conductor or nonconductor, amorphous or crystalline, colorant or decolorant, and a hydrogenator or dehydrogenator. Selenium is one of the few strategic materials that remain in short supply.

Summary

Selenium was discovered in 1817 by the Swedish chemist Berzelius while studying flue dust from the lead chambers of the sulfuric acid works at Gripsholm, Sweden.

The ordinary stable form of selenium is a crystalline, grayish solid with semimetallic luster. It melts at 220.2° C., boils at 684.8° C., and has a specific gravity of 4.47. Selenium is chemically related to sulfur, with which it is commonly associated in nature. It is found native, like sulfur, and also in the selenides of copper, silver, lead, mercury, bismuth, and thallium. About 25 minerals contain selenium in varying percentages, but none of the species is considered a commercial source.

The most important commercial source of selenium at present is the anode mud or slime produced in the electrolytic refining of blister copper. The selenium content of blister-copper anodes handled by domestic refineries ranges from 0.01 to 0.07 percent; the slimes usually contain 4 to 25 percent selenium.

Commercial uses for selenium began to develop on a substantial scale between World Wars I and II in the glass, ceramic, chemical, pigment, and rubber industries and later in the stainless steel industry. Selenium rectifiers for converting alternating to direct current were used industrially for many years but only on a small scale compared with the period after 1947, when the miniature-type selenium rectifier for electronic circuits was developed. These diminutive rectifiers are often employed in voltage multiplier circuits, for radio and television sets, designed to eliminate heavy and bulky transformers and rectifier tubes. Military uses of selenium rectifiers cover a wide range.

Selenium is used in the glass industry as a decolorizer and in the stainless steel industry as an additive to improve machinability and control the quality of castings. A diversified group of chemical uses, including pigments and rubber accelerators, now consumes more selenium than both the glass and steel industries combined.

Demand existed for larger quantities of selenium than were available and this shortage was expected to continue until at least 1958. Little further increase in the production of selenium from copper was expected, and development of low-grade selenium deposits appeared possible.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

GEOGRAPHIC DISTRIBUTION

CENTRAL AND SOUTH AMERICA

The only selenium recovered from Central and South America thus far has been as a byproduct of the electrolytic refining of South American copper in the United States and Europe (*3*).[2]

AFRICA

Northern Rhodesia produced almost 30,000 pounds of selenium in 1952. Selenium contained in Belgian Congo copper is recovered at Oolen, Belgium, and reported as Belgian production (*3*).

AUSTRALIA

Australian output of selenium is about 4,000 pounds a year from slimes produced at electrolytic copper refineries that contain only 1 or 2 percent selenium (*3*).

ASIA

Selenium is recovered at several Japanese copper refineries and sulfuric acid plants. Japan produces about 50,000 pounds of selenium annually (*3*).

EUROPE

Sweden is usually the largest European producer of selenium; during 1951 the Swedish output was 90,000 pounds (*3*).

The Outokumpy Oy. in Finland produces a small quantity of selenium.

Belgium is a significant selenium producer but obtains most of its selenium-bearing materials from Africa.

West Germany produced over 16,000 pounds of selenium in 1951.

In southeastern Yugoslavia the Bor Copper Mining Corp. recovers about 1,000 pounds of selenium per year at its electrolytic refinery.

NORTH AMERICA

Mexico is a major selenium producer, but most of its selenium is recovered and refined in the United States and is included in the United States production statistics (*3*).

Blister copper from the Cananea Consolidated Copper Co. at Cananea, Mexico, formerly refined in the United States, is now refined at an electrolytic copper refinery near Mexico City, and the anode muds are sent to a silver recovery plant at Monterey, Mexico, for extraction of the contained metals.

Selenium is produced in Canada by the International Nickel Co. at Copper Cliff, Ontario, and by Canadian Copper Refiners, Ltd., Montreal East, Quebec. Canadian selenium production and value in recent years are as follows: 382,603 pounds valued at C$1,239,633 in 1951; 242,030 pounds valued at C$786,599 in 1952; 262,346 pounds valued at C$1,101,854 in 1953; and 368,800 pounds valued at C$1,844,000 in 1954.

The United States is the world's leading producer of selenium.

GEOLOGY

The selenium content of igneous rocks is about 0.09 p. p. m.[3] Cores of marine sediments from the Continental Shelf average 1 to 2 p. p. m. selenium, whereas deep-sea cores assay much lower. Selenium assays of the Continental Shelf deposits correspond closely with the selenium content of the sedimentary rock strata of western United States and the soils derived from them (*3*).

A large area of North America—from Canada to Mexico and from the Mississippi to the Rocky Mountains—is underlain by seleniferous rocks and soils.

Soils derived from seleniferous rocks usually contain 2 or 3 p. p. m. selenium. Annual rainfall in the seleniferous area is not enough to leach even the more soluble compounds of selenium from the soil. Soils containing only 1 or 2 p. p. m. selenium commonly sustain plants with a selenium content of 50 to 10,000 p. p. m. and in one instance a record 14,920 p. p. m. Some plants are indicative of seleniferous soils (*6*).

Aside from the selenium minerals themselves, magmatic sulfide deposits represent the greatest known concentration of selenium in the earth's crust. Average selenium content of magmatic sulfides is about 0.02 percent.

METALLURGY

Blister copper averages about 0.05 percent selenium (*3*). About 25 percent of the selenium content of copper ores has been lost before the blister copper stage is reached.

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

[3] P. p. m.=parts per million.

Blister copper is converted to refined copper by electrolysis. Anode muds or slimes are a byproduct of this electrolytic refining. Anode slimes contain 2 to 25 percent selenium, averaging about 8 or 9 percent. The recovery of selenium from anode slime is different at each individual plant.

Several selenium-recovery processes that have been used are: Oxidizing roast, sulfating roast, boiling with dilute H_2SO_4 and niter, aeration in dilute H_2SO_4, pyrometallurgical methods, and miscellaneous methods.

The sulfate-roast process was used by the Canadian Copper Refiners, Ltd., Montreal East, Quebec.

The American Smelting & Refining Co. utilized the soda-smelt process for many years at Baltimore.

Kennecott Copper Co. fused selenium slimes with sodium bisulfate after first leaching out copper with acid.

The Perth Amboy, N. J., plant of the International Smelting & Refining Co. converted aerated slimes to alkaline slags. The selenium content of these slags was dissolved with sulfuric acid and combined with the water from leaching the flue dusts. Elemental selenium was then precipitated with sulfur dioxide.

At the Copper Cliff, Ontario, refinery of the International Nickel Co. the slimes were first fluxed with sand and fluorspar. The resulting matte was refined to doŕe, and the soda niter slags were leached with water. Selenium was recovered from this leach water byprecipitation as "coke" selenium.

In some plants slimes and fume products from other metal-refining operations are added to the byproducts-recovery processes along with selenium scrap and spent selenium catalysts. In addition, many fractions are recycled and reprocessed, the net result being a rather complicated operation. Since these raw materials are seldom assayed, the source of any given batch of selenium is unknown.

Slime-treatment processes are designed primarily to recover the precious metals. Selenium and tellurium recovery has secondary importance. Frequent changes are made in slime-treatment processes to increase recovery rates, lower costs, and accommodate changes of raw materials.

The selenium end product of all of these processes is commercial-grade selenium (97.2 to 99.94 percent pure), usually in coke form. This coke selenium is pulverized and sold as commercial-grade selenium or further refined to high-purity selenium (99.99+percent). Ferroselenium is produced as a byproduct of high-purity selenium production. Ferroselenium can be used directly in producing stainless steel.

In 1953 selenium-production facilities were sufficient for much lager operations had an increased supply of slimes become available.

RESERVES[*]

Known reserves of minable copper at present economic and technologic levels are reported to be 25 million tons.

If all of the 25 million tons of copper reserves was mined and electrolytically refined, 18,750,000 pounds of selenium would be obtained if the present recovery rate of three-fourths pound of selenium per ton of electrolytically refined copper were maintained.

A continuation of present trends toward mining lower grade deposits on a larger scale will increase the total minable reserves.

Several additional selenium occurrences are possible economic sources of selenium. The seleniferous volcanic tuffs at Lysite, Wyo., are estimated to contain 15 million pounds of selenium. Tremendous tonnages of selenium are contained in the rock strata and soils of western United States, and research is in progress directed toward economic recovery from the richer portions of the selenium-bearing deposits. Magmatic sulfide deposits, other than copper, also contain large quantities of selenium. Pyrite-mining operations may produce byproduct selenium. Lead ores frequently have a high selenium content.

USES

Before World War I (*3*), selenium was regarded as the abomination of the smelterman and was mostly discarded. Even with present strong demand, selenium is regarded of minor importance compared with precious-metal byproducts. During World War I, when manganese was in short supply, selenium first came into use as a substitute to decolorize glass by neutralizing the greenish tinge imparted by iron impurities. Because selenium has certain inherent advantages over manganese in decolorizing glass, its use continued after manganese again became available.

The glass industry continued to be the major consumer of selenium until after World War II, when the selenium-rectifier industry became the principal consumer. Selenium rectifiers have been known since 1884 but were not used extensively until about 1930, when German and American investigators improved the device. Selenium dry-plate rectifiers change alternating to direct current and are extensively used in electroplating, welding, battery chargers, magnet coils, arc lamps, and in the operation of direct-current motors. Additional military uses of selenium rectifiers include aircraft controls, degaussing equipment, dynamotor power, guided missiles, ground starters for aircraft, magnetic amplifiers, radar equipment, relay and solenoid power, sonar equipment, telephones, teletypes, and voltage regulators. The largest increase in rectifier use occurred in 1947, when miniature rectifiers suitable for radio and television circuits were developed. These diminutive rectifiers, used in voltage multiplier circuits of radio and television sets, can deliver 200 to 500 volts d. c. from a 117-volt a. c. source, eliminating a heavy and bulky transformer and rectifier tube and saving more than ½ pound of copper and 7½ pounds of steel per television set.

Selenium has photoelectric conductivity, as well as rectification properties. Selenium photoelectric cells are used in photographic exposure meters, electric eyes, colorimeters, and pyrometers.

As previously noted, selenium is added to glass to alter the greenish tinge of iron impurities to a neutral gray. From 0.03 to 0.3 pound of metallic selenium, sodium selenite, barium selenite, or sodium selenate per ton of glass is required, the exact amount being determined by the iron content to be neutralized. Excess selenium causes a pink tinge, which is desirable for milk bottles and a few other food containers. From 10 to 50 pounds of selenium per ton of glass yields a desirable ruby-red glass, which is used in tableware, taillight-signal lenses, light filters, infrared equipment, lantern globes, and traffic lights.

Selenium is important in the rubber industry. Added mostly as selenium diethyl dithiocarbamate, selenium acts as an accelerator and vulcanizing agent and, in addition, promotes resistance to heat, oxidation, and abrasion and increases the resilience of rubber.

Selenium as cadmium sulfoselenide is used as a pigment to produce orange, red, and maroon colors in paint, soap, rubber, ceramics, paper, printing inks, plastics, dyes, leather, and other products.

Selenium and its compounds are utilized in industrial and research chemistry. Selenium dioxide oxidizes organic compounds, producing reactions not obtainable by any other known method. Catalysts containing selenium can be either hydrogenating or dehydrogenating agents. Selenium is a hydrogenation and cracking catalyst in crude-oil and coal distillation. Soaps, waxes, edible fats, and plastics are produced by fat hardening with a selenium catalyst. The Kjeldahl method of nitrogen determination utilizes a selenium catalyst. It imparts exceptional antioxidant properties to printing ink, paints, mineral oils, transformer oils, and vegetable oils. The nondrying property of linseed, oiticica, and tung oils is due to contained selenium. Selenium also retards gelation in tung oil. The antioxidant and antigalling properties of selenium compounds find application in lubricating oils and in extreme pressure lubricants. Selenium is the catalyst that synthesizes cortisone, used to treat arthritis, and isonicotinic acid, used to treat tuberculosis. Selenium compounds are contained in insecticides, fungicides, parasiticides, bactericides, and herbicides and in dandruff, dermatitis, and deodorant pharamaceuticals. Photographic photosensitizers and toning baths contain selenium compounds such as sodium selenosulfate. Blasting caps, mercury-vapor detectors, flotation reagents, fireproofing agents, insect repellants, phosphorescents, and luminescents can be made from selenium compounds.

Care must be exercised in such widespread usage, inasmuch as selenium and most of its compounds are toxic or corrosive. Selenium oxychloride is so corrosive that it has been referred to as the universal solvent. The oxychloride is used in phenolic resin solvent, rubber-resin solvent, glue solvent, and paint and varnish removers and has been used to activate charcoal for gas absorption (*3*).

Selenium improves the machinability of stainless steel. Forging steels contain 0.18 to 0.22 percent selenium and the full machining grade 0.30 to 0.35 percent. In cast steel 0.01 to 0.05 percent selenium assures a fine-grained, bubble-free casting. Small quantities of selenium improve the machinability and tensile strength of copper and copper-rich alloys without appreciably lowering ductility or conductivity.

Xerography, a dry photographic process, employs selenium-coated metal disks from which the photographic image is transferred by static electricity.

Radioactive selenium has been utilized in radiography.

Rectifier manufacturers consumed 497,041 pounds of selenium in 1953, which was 45 percent of the total consumption. Two-thirds of the rectifiers manufactured in 1953 were power rectifiers, and one-third were rectifiers for radio and television receiving sets. One-third of all the selenium produced in 1953 was consumed by the chemical industry, 15 percent by the glass industry, and 6 percent by the steel industry.

GOVERNMENT WARTIME CONTROLS

Selenium was available in adequate quantities to meet all requirements during World War II.

Shortages of selenium developed just before the Korean War.

The National Production Authority placed both imported and domestically produced selenium under complete allocation in February 1952, explaining that the action was necessary if essential defense requirements were to be met. Allocation was terminated in early 1953, but the shortage continues.

In another national emergency, Government controls would probably have to be reimposed on selenium.

SUBSTITUTES

The National Bureau of Standards and several private companies have been investigating substitutes for selenium in rectifiers. For large fixed power applications, germanium rectifiers can substitute for selenium rectifiers where water cooling is feasible. In the high-temperature ranges oxidized plates of titanium appear to be promising substitutes for selenium rectifier plates. Many other semiconductors are being studied as possible substitutes for selenium in rectifiers. Of these, silicon seems the most likely to be commercially available in the near future.

Glass manufacturers have already reduced the quantities of selenium required per unit of glass, and further reduction of selenium consumption is possible by using manganese, arsenic, and rare earths in addition to selenium for decolorization. Ruby-red glass can be obtained by additions of gold or copper, but the resultant glass is somewhat inferior to selenium ruby-red glass.

Tellurium imparts desirable qualities to steel and copper equal to or better than selenium, but the garlicky halitosis developed by those handling the tellurium ingredients has thus far prevented much substitution. Sulfur can also substitute for selenium in steel, but the resultant product is less satisfactory than those containing selenium.

Sulfur and tellurium can also be substituted for selenium to a limited extent in the rubber and chemical industries.

DOMESTIC PRODUCTION, CONSUMPTION, AND FOREIGN TRADE

The American Smelting & Refining Co. (Baltimore, Md.), United States Metals Refining Co. (Cartaret, N. J.), International Smelting & Refining Co. (Perth Amboy, N. J.), and Kennecott Copper Corp. (Garfield, Utah) are the principal selenium producers in the United States. In addition, several companies, including Kawecki Chemical Co., Boyertown, Pa., and Vickers Electric Division, Vickers, Inc., St. Louis, Mo., recover selenium from the scrap of rectifier manufacturers and spent catalysts and also upgrade commercial-grade selenium into high-purity selenium. In 1953 the Kawecki Chemical Co. initiated selenium recovery from sulfuric acid-production byproducts (*3*).

Domestic production of selenium for the 5 years 1947–51 averaged 514,988 pounds annually. In 1952 primary selenium production was 687,384 pounds, and 66,781 pounds of secondary selenium was recovered. A primary selenium production of 923,887 pounds and secondary production of 97,948 pounds were attained in 1953. Preliminary data indicate a primary selenium production of 705,000 pounds and a secondary production of 125,000 pounds in 1954. An appreciable increase is expected in 1955.

Apparent domestic consumption (producers' domestic shipments plus imports to consumers) of selenium has ranged during the 5 years from about 600,000 to over 1,000,000 pounds, averaging 848,460 pounds annually. Producers' domestic shipments in the period averaged 536,215 pounds and imports to consumers 312,245 pounds. The trend since this period has been toward greater domestic shipments and smaller consumer imports. Apparent domestic consumption of selenium was 802,033 pounds in 1952, 1,125,501 pounds in 1953, and about 1,025,000 pounds in 1954. Demand for selenium and selenium compounds in the 1951–52 period exceeded 1,500,000 pounds and was steadily increasing, with no hope of fulfillment in the near future.

Exports of selenium by producers were at a low level during the 1947–51 period, averaging 27,321 pounds annually. Exports since this period have been at an even lower level and under strict Government control.

Imports of selenium for the 5 years 1947–51 averaged 315,568 pounds a year. A record 529,175 pounds of selenium was imported in 1947, largely from Canada. Selenium imports totaled 123,135 pounds in 1952, 99,865 pounds in 1953, and 194,692 pounds in 1954.

PRICES

The price of commercial-grade selenium has risen gradually from $1.75 a pound in 1947 to $6.00 a pound in 1955. Further increases are possible. Selenium prices in foreign markets are 2 or 3 times higher than those in the domestic and Canadian markets.

RESEARCH

Aside from the previously mentioned search for substitutes for selenium, most research is directed toward increasing the supply of selenium. Selenium producers have completed research on increasing the selenium-recovery rates, and most of the resultant improvements have been incorporated into existing selenium-recovery processes. Many companies have analyzed their byproducts and wastes for selenium content. Very few were high enough in selenium content to warrant extraction.

The Bureau of Mines Salt Lake City Experiment Station has been investigating the selenium content of ores, rock strata, and industrial byproducts for several years. This station has also experimentally recovered selenium from some of the richer and more promising sources, such as the Lysite, Wyo., volcanic tuffs and the vanadiferous black shales of southeastern Idaho and southwestern Wyoming. Effective July 1, 1954, the General Services Administration provided financial support for continuing this research program.

Preliminary reports from the investigation of the commercial feasibility of extracting selenium from selenium concentrating vegetation are encouraging.

OUTLOOK

Present indications are that selenium production will be expanded slightly in 1955–57. Normal industrial demand for selenium is likely to increase more rapidly than selenium-production capabilities, in spite of the widespread trend to design away from the use of selenium and the increased used of selenium substitutes. The selenium shortage can be expected to continue for at least 2 or 3 more years. During 1957–59 an increased selenium production should be possible as a result of the several research programs now in progress. An annual world production of 3 or 4 million pounds of selenium should be feasible as a result of large-scale mining of low-grade deposits and from the farming of seleniferous vegetation.

PROBLEMS

The most important selenium problem at present is the inadequate supply of available selenium. The quantity of selenium produced is governed not so much by the demand for selenium as by the demand for copper. Large quantities of selenium are available from other sources, but these contain only 1 pound or less of selenium per ton of ore. Large-scale operations would be necessary to recover such small percentages of selenium economically. Before industry can be expected to expend several million dollars in such large-scale production facilities, it must be convinced that such an undertaking is feasible and reasonably assured of returning a profit: additional technical and economic studies will be needed. Designing away from selenium and the increasing use of selenium substitutes tends to discourage any new large-scale production of selenium.

An attempt to farm seleniferous vegetation would be an entirely new type of mining never before attempted and attended by a host of pioneering problems.

The worldwide scope of the selenium shortage plus the large price differential between North American selenium and world-market selenium prices combine to cause several import and export problems.

The quantity of selenium that would be consumed if more were available is not known. A survey to evaluate this demand is needed.

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SILVER

By

James E. Bell [1]

SILVER is one of the metals known earliest to man and through the ages has been held in high esteem for monetary use and the arts. In modern times, as silver's monetary use has declined, its application in the arts and industry has increased. Motion pictures would not have been possible without silver; silver solders provide an important safety factor in aviation. The production of silver is trending downward, but stocks are very large.

Summary

Silver was widely mined and used in ancient and medieval times, but production was small compared to that which followed the discovery and development of rich deposits in the Americas after 1493. Production of silver in the United States was insignificant until the Comstock lode and other great deposits in the Western States were discovered after 1859. For many years the United States has been second in silver production, exceeded by Mexico. The total world output since 1493 is approximately 19,700,000,000 ounces, of which 82 percent was produced in the Western Hemisphere and 21 percent in the United States.

Most of the predominant silver deposits in the United States have been depleted, and in recent years the domestic silver output has been derived chiefly as a byproduct of base-metal and gold mining. Normally, a large quantity of silver is recovered in the United States from secondary sources, and for many years there has been a large excess of imports of silver over exports. Large quantities of silver are in circulation, and silver monetary reserves are very large.

The principal uses of silver are as coinage, as a monetary reserve, in the arts, in photography and other industrial applications, and in dentistry and medicine.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

HISTORY

Silver probably was discovered later than gold or copper but was known and valued by mankind before the beginning of history. Ancient slag dumps in Asia Minor and on islands in the Aegean Sea attest that men had learned to mine and smelt lead-silver ores and separate silver from lead as early as the third millennium B. C. The most famous of the ancient silver districts was at Laurium, Greece; it was worked for several centuries in the first millennium B. C., with a total output estimated at over 250,000,000 ounces. Silver was mined widely during the era of the Roman Empire, and most of the silver deposits in what was once the empire were known to the Romans. Silver mines were worked in some countries in Europe during the Middle Ages, but the output was relatively small.

An enormous increase in the world production rate of silver followed soon after America was discovered in 1492, when deposits of silver ores far larger and richer than those previously known in the Old World were opened by the early conquistadors. Famous among the many bonanza silver districts found and developed in the 1500's are Potosí, Bolivia and Guanajuato and Pachuca, Mexico. Potosí, with a total output exceeding 1 billion ounces, was the greatest of all. Bolivia was the leading silver producer to 1700, and Mexico has led during most years since that date.

Production of silver in the United States was insignificant until discovery of the Comstock lode in Nevada in 1859 (*5, 10*).[2] Development of this great deposit and a long series of others in the Western States placed the United States first in silver production from about 1871 to 1900. Since 1900 the United States has been second to Mexico in silver output. Most of the predominately silver deposits in the United States have been depleted, and for many years the greater part of the domestic output has been recovered as a byproduct of ores mined principally for base metals or gold.

The world production of silver from 1493 to 1953 was approximately 19,700,000,000 troy ounces, or over 675,000 short tons. Of this total, 82 percent was produced in the New World; North America contributed 62 percent and South America 20. Mexico produced 35 percent of the world total, the United States 21, Peru 9, and Bolivia 9.

It is estimated that about one-third of the total world output of silver is in circulation as coinage or held by governments for monetary purposes; one-third, including that hoarded, is privately owned; and one-third has been misplaced or dissipated. As of the end of 1952 United States Treasury silver holdings were about 1,938,000,000 ounces, exclusive of more than 410,000,000 ounces released to foreign countries under Lend-Lease agreements that provide for return of the silver. Silver held by banks in the United States and in coins in circulation amounted to approximately 996,000,000 ounces more.

The alltime high in annual world production of silver was approximately 278,000,000 ounces in 1937; for the United States the alltime high was nearly 75,000,000 ounces in 1915. In 1952 the world output of silver was 210,200,000 ounces and that of the United States approximately 39,000,000 ounces.

In the United States 69 percent of the 1952 output of silver was recovered as a byproduct of base-metal ores. The remainder, except for a relatively small quantity recovered as a byproduct of gold mining, both placer and lode, was derived from "dry" silver ores, with values predominantly in silver. Approximately 99 percent of the domestic silver production of 1952 was recovered in the smelting of ores and concentrates (*2*).

GEOGRAPHICAL DISTRIBUTION OF SILVER INDUSTRY

Silver is widely distributed, and some quantity has been found in nearly all countries. During recent years silver has been produced in some 60 countries in all the continents and Oceania. In the United States, silver has been produced in 26 States and Alaska. Approximately 99.5 percent of the domestic production has been mined in the Western States and most of the remainder in Michigan and Missouri. The five leading States, in order of total output, are Montana, Utah, Colorado, Idaho, and Nevada; they supplied 85 percent of the total United States production. The five leading States, in order of output in 1952, are Idaho, Utah, Montana, Arizona, and Colorado; their

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

combined production was 91 percent of the domestic total. Nearly all base-metal ores mined in the United States contain some silver, although the Tri-State zinc-lead ores are an outstanding exception (*2*).

TECHNOLOGY

There are some 60 well-known silver minerals, but the chief minerals of economic importance are the following: Native silver, argentite (Ag_2S), cerargyrite ($AgCl$), polybasite (Ag_9SbS_6), proustite (Ag_3AsS_3), and pyrargyrite (Ag_3SbS_3) (*12*). Silver is most commonly associated with lead, and most copper and zinc deposits contain some silver. Gold in placers and lode deposits usually contain considerable silver. Silver is often associated with cobalt. The silver content of most base-metal deposits decreases with depth.

The observations on geology that follow apply to straight silver deposits and silver-lead deposits, rather than to deposits of base metals that yield silver as a byproduct. Most of the silver of the world has been mined in the western Cordillera of the Americas, where it is associated with intrusive volcanic rocks of Tertiary age. Most silver deposits are formed by hydrothermal action. The deposit types include cavity fillings—fissures, stockworks, and breccias; replacements—massive, lode, and disseminated; and sulfide enrichments. Deposits occur in host rocks of nearly all kinds and in formations of most geological ages. Massive replacements are numerous, but most of the world silver has been won from fissure veins. The unique Cobalt silver district in Canada is particularly interesting. Many narrow fissure veins of relatively small horizontal and vertical extent occur in rocks of pre-Cambrian age. Fully 97 percent of the silver values were found as native silver, and great slabs of nearly pure silver were common. Discovered in the early 1900's, the deposits were virtually exhausted in less than 3 decades, but with a total production of around 375 million ounces—a quantity exceeded by only 3 other districts of the world.

For centuries silver was recovered from ores by cupellation or amalgamation. About 1850 the Augustin process was developed, in which the ore was roasted with common salt to convert the silver values to chlorides that could be dissolved in hot brine and thus removed from the ore. The Augustin and related processes were superseded by the Patera process, in which sodium hyposulfite was used as the solvent, usually after a preliminary chloridizing roast. All processes to recover silver have been replaced by the cyanide process (introduced about 1890) or by concentration and smelting.

Free-milling straight silver ores may be treated by cyanidation or by concentration and cyanidation (*4*). These processes formerly were employed in various districts in the United States but have virtually been discontinued in this country because of depletion of ores suited to them. Silver ores are still being cyanided to some extent, however, in some districts in Mexico and Central America.

The largest operator in the United States treating "dry" silver ore employs straight flotation to recover the metal values. In 1947 the ore averaged about 45 ounces per ton silver and 2.6 percent lead, with minor quantities of copper, zinc, antimony, and arsenic. The concentrate produced averaged 375 ounces of silver and 22 percent lead per ton. Recovery of both silver and lead exceeded 98 percent.

Byproduct silver from base-metal mines follows the ores produced through all their processes of mining, direct shipping or concentration and shipping, and smelting and is ultimately recovered in the precious-metal units of base-metal refineries. Losses of silver values in concentration processes are about the same as those of base-metal values; losses of silver values in smelting are low. The average recoveries of silver from base-metal ores of various classes in the United States (Kansas, Missouri, Oklahoma excluded) in 1952 were as follows: Copper ore, 0.082 ounce per ton; lead ore, 3.561 ounces; lead-copper ore, 1.678 ounces; zinc ore, 0.218 ounce; zinc-lead, zinc-copper, and zinc-lead-copper, 1.512 ounces; total base-metal ores, 0.239 ounce (*3*). In some base-metal mines the revenue from silver in the ores may determine whether the property can be worked economically.

The silver in lead ores is carried down with the lead in smelting and is separated from it by the Parkes process. This process consists of stirring zinc into the molten lead bath, heated to above the melting point of zinc, and allowing the mixture to cool. A zinc crust containing the silver separates out; the zinc is driven off by distillation, and the residue is cupeled to silver bullion. The silver in copper ore is carried down with the copper in smelting processes; and, in the electrolytic refining of copper, the silver separates out into the anode slime, whence it is recovered as silver bullion by smelting (*7*). The silver produced with gold in gold-placer mining or lode-gold mining is recovered in the electrolysis of gold bullion.

Crude-silver bullion, whether produced by the cyanide process or separated from lead by cupellation, or from copper or gold by electrolysis, or recovered from scrap contains gold and/or small quantities of some other metals. Such crude-silver bullion formerly was refined by chemical methods, but more recently most has been refined by electrolysis, with an electrolyte of silver nitrate and nitric acid. The refined silver is cast into ingots that have a purity over 99.9 percent.

BYPRODUCTS AND COPRODUCTS

Gold is usually associated with silver in nature, and gold is a byproduct or coproduct of some silver-mining operations. Lead is a byproduct or coproduct of silver-lead ores, and copper, zinc, antimony, bismuth, and arsenic may also be byproducts. Silver may be present in enough quantity in some base-metal ores to be classed as a coproduct rather than a byproduct. Tin is a coproduct of silver in some Bolivian ores.

SUBSTITUTES AND SECONDARY SOURCES

Aluminum and rhodium are substituted for silver for reflectors and enamels in dental fillings, and tantalum is substituted for plates, sutures, and pins in surgery. The total substitution of other materials for silver is small, however, compared to the total consumption.

A constant flow of secondary silver in the form of new scrap; worn-out or remoded silverware, jewelry, and novelties; and silver compounds from photographic laboratories and chemical plants is returned to the refineries by manufacturers and dealers. According to the Bureau of the Mint, in the United States an average of approximately 37 million ounces of silver was returned from industry annually during 1941–52.

RESERVES

Based on an estimate made as of January 1944 (*12*), the total inferred, indicated, and measured recoverable silver reserves of the United States and Alaska were 763,400,000 ounces; this equals about 20 percent of the total domestic production to the same date. The distribution of the reserves, by States, was as follows: Alaska, 2,000,000 ounces; Arizona, 100,000,000; California, 40,000,000; Colorado, 100,000,000; Idaho, 225,000,000; Montana, 105,000,000; Nevada, 45,000,000; Central States 5,500,000; New Mexico, 13,000,000; Oregon, 1,000,000; South Dakota, 2,200,000; Utah, 120,000,000; Washington, 3,000,000; Eastern States, 1,700,000. Additional reserves in new discoveries of silver ores and silver-bearing base-metal ores have largely offset the production of silver since 1944.

Data on the silver reserves of other countries are nonexistent or unavailable. However, because the known principal silver districts are largely depleted, the silver reserves remaining probably are much smaller than the quantity so far produced. Recoverable world reserves of silver to the order of 5 billion ounces is perhaps a reasonable estimate.

USES

Silver has been used for centuries for coinage and more recently as a monetary reserve in the form of bullion to support paper currency issued by governments and central banks.

Silver has many properties that make it valuable in the arts and industries. It is beautiful in color and has the ability of taking a fine finish. It is highly malleable and ductile and ranks first among the metals in conductivity of electricity and heat. It resists corrosion, especially by weak acids and organic compounds.

For many years the principal nonmonetary consumer of silver has been the silverware industry, mostly in fabrication of tableware from sterling silver. Pure silver is too soft for most uses and is alloyed with 7.5 percent copper to form "sterling silver" of standard grade. Jewelry, insignia, and many novelties are also made of sterling silver.

Second as a consumer of silver is the photographic industry and third the electroplating industry. Of growing importance are the silver solders and brazing alloys, which are made in a wide variety of types containing 10 to 80 percent silver and the remainder copper, zinc, or other metals. Silver brazing alloys are widely used in joining pipes, making electrical connections, and forming mechanical assemblies. Silver alloyed with about 10 percent copper finds much use in electrical contacts; small additions of silver to copper impart hardness to commutator bars.

Silver compounds are used for caustic, astringent, and antiseptic purposes in medicine. Silver has considerable use in dental fillings and in surgery as suture wires and plates.

A new use for silver developed during World War II is the plating of bearings of engines of aircraft and mechanized equipment for high-duty service and long life. Soft lead-silver solder containing about 2.5 percent silver has advantages over soft lead-tin solders and Babbitt metal for some uses; in these uses in effect it is a conserver for tin. Silver in very large quantities was loaned to the Defense Plant Corporation by the Treasury during the War for use as electric-current busbars, releasing copper for other needs.

STATISTICAL CONSIDERATIONS

In the United States the Bureau of Mines gathers, analyzes, and compiles statistics on domestic mine production of silver in cooperation with the geological surveys of some States. The coverage permits the Bureau to publish regular reports on the mine production of silver, by mining districts, counties, and States. Data also are published showing details on the sources of silver with respect to various types of ore and on the portions recovered by the various methods of treatment (*4*). General data on refinery production and the consumption of silver are supplied to the Bureau of Mines by the Bureau of the Mint. Data on foreign trade in silver are supplied by the United States Department of Commerce.

Data on silver production in foreign countries are gathered by the Bureau of Mines by means of questionnaires sent to each country and by consular reports, official publications, business magazines, and company reports.

PRODUCTION, CONSUMPTION, AND FOREIGN TRADE

Data on silver production from various aspects are given in tables 1, 2, and 3.

Data on the consumption of silver in the United States, as supplied by the Bureau of the Mint, are given in table 4. No breakdown of the consumption by uses is available, but it is known that the silverware industry provides the largest market, followed by the photographic industry.

In 1952 the United States imported silver ore, base bullion, refined bullion, and silver coins from some 50 countries. In the same year it exported refined bullion and coins to 20 countries. Table 5 summarizes silver movements to and from the United States from 1948 to 1952, as supplied by the United States Department of Commerce.

TABLE 1.—*Mine production of silver for various periods, 1493–1952* [1]

Period	Number of years	World: Quantity, fine ounces	North America		South America		Europe	
			Quantity, fine ounces	Percent	Quantity, fine ounces	Percent	Quantity, fine ounces	Percent
1493–1600	108	746, 932, 166	90, 410, 000	12. 10	453, 820, 000	60. 76	177, 702, 166	23. 79
1601–1700	100	1, 271, 922, 450	306, 650, 000	24. 11	800, 570, 000	62. 94	89, 702, 450	7. 05
1701–1800	100	1, 832, 768, 759	1, 044, 510, 000	57. 00	598, 040, 000	32. 63	189, 218, 759	10. 32
1801–50	50	1, 064, 261, 495	605, 475, 600	56. 89	304, 680, 000	28. 63	153, 605, 895	14. 43
1851–1900	50	4, 034, 289, 688	2, 646, 679, 759	65. 60	692, 474, 639	17. 16	496, 072, 827	12. 30
1901–25	25	4, 901, 347, 412	3, 665, 881, 929	74. 79	415, 522, 057	8. 48	311, 476, 259	6. 36
1926–50	25	5, 469, 395, 702	3, 707, 641, 538	67. 81	615, 707, 326	11. 28	405, 558, 085	7. 44
1951–52	2	407, 700, 000	230, 310, 000	56. 57	55, 615, 000	13. 66	70, 000, 000	17. 17

Period	Number of years	Asia		Africa		Australasia		Other	
		Quantity, fine ounces	Percent	Quantity, fine ounces	Percent	Quantity, fine ounces	Percent	Quantity, fine ounces	Percent
1493–1600	108	25, 000, 000	3. 35	----------	------	----------	------	----------	------
1601–1700	100	75, 000, 000	5. 90	----------	------	----------	------	----------	------
1701–1800	100	1, 000, 000	. 05	----------	------	----------	------	----------	------
1801–50	50	500, 000	. 05	----------	------	----------	------	----------	------
1851–1900	50	38, 955, 723	. 97	2, 495, 131	0. 06	157, 611, 709	3. 91	----------	------
1901–25	25	171, 316, 187	3. 49	30, 184, 751	. 62	306, 950, 891	6. 26	15, 338	0. 00
1926–50	25	333, 042, 906	6. 11	110, 869, 180	2. 04	290, 554, 024	5. 32	----------	. 00
1951–52	2	12, 500, 000	3. 07	16, 931, 000	4. 15	21, 929, 000	5. 38	----------	------

[1] Merrill, Charles White, Summarized Data of Silver Production: Bureau of Mines Econ. Paper 8, 1930, 58 pp. (Table 2 brought to date.)

TABLE 2.—*Mine production of recoverable silver in the United States, 1942–52, with production of maximum year, and cumulative production from earliest record to end of 1952, by States*

[Fine ounces]

	Maximum production [1]		Production by years											Total production from earliest record to end of 1952
	Year	Quantity	1942	1943	1944	1945	1946	1947	1948	1949	1950	1951	1952	
Western States and Alaska:														
Alaska	1916	1, 379, 171	119, 704	42, 788	13, 362	9, 983	41, 793	66, 150	67, 341	36, 056	52, 638	32, 870	32, 986	20, 078, 339
Arizona	1937	9, 422, 552	7, 064, 467	5, 713, 889	4, 394, 039	3, 558, 216	3, 268, 765	4, 569, 084	4, 837, 740	4, 970, 736	5, 325, 441	5, 120, 985	4, 701, 330	322, 212, 730
California	1921	3, 629, 223	1, 450, 440	609, 075	778, 936	986, 798	1, 342, 651	1, 597, 442	724, 771	783, 880	1, 071, 917	1, 145, 219	1, 099, 658	114, 622, 975
Colorado	1893	25, 838, 600	3, 096, 211	2, 664, 142	2, 248, 830	2, 226, 780	2, 240, 151	2, 557, 653	3, 011, 011	2, 894, 886	3, 492, 278	2, 787, 882	2, 813, 643	747, 984, 031
Idaho	1937	19, 587, 766	14, 644, 890	11, 700, 180	9, 931, 614	8, 142, 667	6, 491, 104	10, 345, 779	11, 448, 875	10, 049, 257	16, 095, 019	14, 753, 023	14, 923, 165	598, 100, 530
Montana	1892	19, 038, 800	11, 188, 118	8, 450, 370	7, 093, 215	5, 942, 070	3, 273, 140	6, 326, 190	6, 930, 716	6, 327, 025	6, 590, 747	6, 393, 768	6, 138, 185	787, 856, 454
Nevada	1913	16, 090, 083	3, 723, 435	1, 620, 280	1, 259, 636	1, 043, 380	1, 250, 651	1, 377, 579	1, 790, 020	1, 800, 209	1, 537, 217	981, 669	941, 195	598, 031, 601
New Mexico	1885	2, 343, 800	676, 170	463, 583	535, 275	465, 127	338, 000	515, 833	537, 674	380, 855	338, 581	443, 267	479, 318	70, 450, 259
Oregon	1941	276, 158	87, 376	10, 523	20, 243	10, 461	6, 927	30, 379	13, 596	12, 195	13, 565	6, 218	4, 037	5, 305, 302
South Dakota	1900	536, 200	186, 937	35, 886	5, 445	26, 564	86, 901	111, 684	94, 693	109, 383	142, 065	139, 590	132, 102	10, 417, 526
Texas	1938	1, 433, 008	672, 781	10, 284	5, 355	23, 265	42, 922	20, 547	3, 065	2, 691	2, 454	1, 381	4, 672	33, 303, 173
Utah	1925	21, 276, 689	10, 574, 955	9, 479, 340	7, 593, 075	6, 106, 545	4, 118, 453	7, 780, 032	8, 045, 329	6, 724, 880	7, 083, 808	7, 310, 665	7, 194, 109	763, 311, 342
Washington	1902	721, 450	369, 038	370, 440	321, 608	281, 444	264, 453	293, 736	375, 831	357, 853	363, 656	334, 948	315, 645	14, 871, 542
Wyoming	1901	21, 400	52		3	31	26	95	11	21		2		74, 821
Total			53, 854, 574	41, 170, 780	34, 200, 636	28, 823, 331	22, 765, 937	35, 592, 183	37, 880, 673	34, 449, 927	42, 109, 386	39, 451, 487	38, 780, 045	4,086,620,625
West Central States: Missouri	1952	517, 432	69, 106	111, 285	92, 243	94, 822	69, 401	93, 600	114, 187	123, 413	236, 273	184, 424	517, 432	5, 431, 326
States east of the Mississippi:														
Alabama	1936	869				1								5, 239
Georgia	1904	1, 500	7					13	3					10, 963
Illinois	1924	8, 891	104	2, 153	2, 437	2, 198	2, 302	1, 790	4, 047	3, 128	2, 001	3, 465	3, 781	155, 554
Maryland	1917	1, 092												2, 595
Michigan	1916	716, 640	61, 674	48, 479	54, 218	21, 863		3, 089						10, 256, 112
New York	1951	47, 568	40, 012	38, 004	25, 238	14, 271	15, 786	22, 409	18, 788	18, 378	32, 628	47, 568	38, 895	567, 788
North Carolina	1906	30, 769	8, 259	7, 169	1, 461									357, 223
Pennsylvania	1942	15, 501	15, 501	13, 095	13, 545	10, 434	7, 887	9, 863	13, 731	10, 827	10, 563	13, 575	9, 247	249, 360
South Carolina	1940	8, 047	5, 064	135										35, 325
Tennessee	1920	110, 719	34, 671	52, 058	45, 907	35, 391	18, 016	79, 147	39, 692	41, 833	39, 958	24, 960	57, 569	3, 321, 509
Vermont	1952	45, 361		2, 721	18, 862	20, 586	35, 275	21, 469	24, 910	27, 446	28, 205	41, 300	45, 361	[2] 292, 743
Virginia	1944	18, 993	1, 793	14, 947	18, 993	1, 300								79, 389
Total			167, 085	178, 761	180, 661	106, 044	79, 266	137, 780	101, 171	101, 612	113, 355	130, 868	154, 853	15, 333, 800
Grand total			54, 090, 765	41, 460, 826	34, 473, 540	29, 024, 197	22, 914, 604	35, 823, 563	38, 096, 031	34, 674, 952	42, 459, 014	39, 766, 779	39, 452, 330	4,107,385,751

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[1] States east of the Mississippi figures are peaks since 1896, except New York and Pennsylvania which are peaks since 1905. The Illinois figure is the peak since 1907. Alaska, California, Nevada, and Oregon are peaks since 1880.

[2] Includes a small quantity for New Hampshire.

TABLE 3.—*World production of silver, 1948–52, by countries* [1] [2]

[Fine ounces]

(Compiled by Pauline Roberts and Berenice B. Mitchell)

Country	1948	1949	1950	1951	1952
North America:					
United States	39, 228, 468	34, 944, 554	42, 308, 739	39, 907, 257	39, 840, 300
Canada	16, 109, 982	17, 641, 493	23, 221, 431	23, 125, 825	24, 375, 853
Central America and West Indies:					
Costa Rica [3]	3, 029	720	215	582	----------
Cuba	185, 216	[3] 157, 411	[3] 221, 779	[3] 172, 318	[3] 163, 211
Guatemala	(4)	81, 502	339, 360	309, 857	371, 679
Honduras	3, 170, 871	3, 431, 614	3, 514, 556	3, 182, 254	3, 703, 912
Nicaragua [3]	216, 802	191, 082	133, 282	141, 764	137, 309
Panama	400	(4)	1, 940	5, 788	----------
Salvador	[5] 216, 342	280, 309	462, 973	352, 102	368, 448
Mexico	57, 519, 703	49, 454, 882	49, 141, 445	43, 797, 734	50, 353, 560
Total	116, 651, 000	106, 184, 000	119, 346, 000	110, 996, 000	119, 314, 000
South America:					
Argentina	1, 201, 900	1, 249, 421	1, 150, 000	1, 253, 879	962, 948
Bolivia (exports)	7, 555, 424	6, 655, 204	6, 558, 751	7, 137, 465	7, 065, 608
Brazil	23, 095	21, 041	21, 155	20, 315	17, 301
Chile	861, 961	799, 685	746, 765	983, 491	1, 246, 356
Colombia	109, 188	106, 590	115, 711	129, 773	123, 175
Ecuador	205, 800	264, 300	273, 200	33, 600	82, 297
Peru	9, 288, 703	10, 609, 648	13, 367, 807	17, 379, 148	19, 179, 525
Total	19, 246, 000	19, 706, 000	22, 234, 000	26, 938, 000	28, 677, 000
Europe:					
Austria	(4)	7, 427	8, 681	5, 466	3, 858
Czechoslovakia	[6] 1, 600, 000	(4)	(4)	(4)	(4)
Finland	167, 615	171, 150	115, 939	157, 275	150, 083
France	494, 414	570, 888	549, 669	546, 550	353, 650
Germany, West	[7] 867, 459	1, 601, 782	1, 637, 116	1, 819, 957	1, 877, 700
Greece	32, 000	----------	----------	64, 300	72, 403
Italy	595, 464	793, 545	850, 998	869, 710	1, 055, 927
Norway	215, 410	170, 399	167, 184	163, 969	176, 186
Portugal	35, 366	31, 958	68, 288	65, 427	67, 697
Spain	339, 396	514, 283	823, 831	735, 908	553, 128
Sweden	1, 137, 943	1, 140, 708	1, 275, 709	1, 145, 890	(4)
U. S. S. R. (estimate)	12, 000, 000	20, 000, 000	24, 000, 000	24, 000, 000	24, 000, 000
United Kingdom	15, 942	13, 996	18, 153	26, 777	(4)
Yugoslavia	1, 504, 237	1, 917, 792	2, 386, 839	3, 032, 008	2, 577, 043
Total (estimate)	19, 500, 000	29, 000, 000	34, 000, 000	35, 000, 000	35, 000, 000
Asia:					
Burma	450, 000	[6] 250, 000	1, 800	280, 720	242, 307
China	(4)	160, 000	320, 000	320, 000	[6] 400, 000
India	12, 797	11, 275	15, 676	17, 180	(4)
Japan	2, 185, 672	2, 887, 265	3, 964, 572	4, 609, 924	5, 288, 707
Korea, Republic of	38, 505	18, 932	1, 222	5, 371	6, 130
Philippines	150, 760	218, 419	216, 034	274, 602	693, 751
Saudi Arabia	67, 819	81, 295	124, 287	109, 912	111, 945
Taiwan (Formosa)	14, 133	17, 148	20, 603	26, 388	(4)
Total (estimate)	3, 000, 000	3, 700, 000	4, 700, 000	5, 700, 000	6, 800, 000
Africa:					
Algeria	29, 739	32, 472	32, 000	9, 600	(4)
Bechuanaland	233	27	39	70	281
Belgian Congo	3, 805, 715	4, 549, 330	4, 459, 951	3, 795, 266	4, 727, 252
French Morocco	487, 598	736, 220	1, 007, 900	1, 569, 000	(4)
Gold Coast (exports)	45, 553	40, 051	43, 317	52, 542	44, 116
Kenya	3, 184	2, 279	2, 586	2, 150	17, 315
Mozambique	616	244	71	96	(4)
Nigeria	4, 270	484	325	200	270
Northern Rhodesia [8]	145, 865	134, 920	173, 304	100, 702	312, 940
Southern Rhodesia	81, 404	84, 495	85, 549	79, 731	81, 356

See footnotes at end of table.

TABLE 3.—*World production of silver, 1948–52, by countries* [1] [2]—Continued

[Fine ounces]

Country	1948	1949	1950	1951	1952
Africa—Continued					
South-West Africa	323, 647	642, 500	843, 737	1, 030, 066	1, 064, 335
Swaziland	124	120	60	18	------------
Tanganyika (exports)	25, 010	27, 631	31, 014	35, 697	35, 900
Tunisia	16, 011	67, 517	73, 947	61, 119	[6] 62, 000
Uganda (exports)	56	42	35	14	(4)
Union of South Africa	1, 170, 951	1, 159, 375	1, 119, 135	1, 162, 588	1, 176, 433
Total	6, 140, 000	7, 478, 000	7, 873, 000	7, 899, 000	9, 032, 000
Oceania:					
Australia:					
Commonwealth	10, 057, 519	9, 849, 213	10, 677, 456	10, 792, 032	11, 256, 742
New Guinea [9]	31, 739	31, 786	30, 399	33, 603	(4)
Fiji	29, 187	29, 755	37, 736	24, 869	(4)
New Zealand	232, 563	232, 599	199, 701	133, 291	(4)
Total	10, 351, 000	10, 143, 000	10, 945, 000	10, 984, 000	11, 362, 000
World total (estimate)[1]	174, 900, 000	176, 200, 000	199, 100, 000	197, 500, 000	210, 200, 000

[1] Silver is also produced in Bulgaria, Cyprus, Hong Kong, Hungary, Federation of Malaya, Indonesia, North Korea, Poland, Rumania, Sarawak, Sierra Leone, and Turkey; production data are not available, but estimates are included in total.

[2] This table incorporates a number of revisions of data published in previous Gold and Silver chapters.

[3] Imports into the United States. Scrap is included in this figure in many instances, most notably in the case of Cuba.

[4] Data not available; estimate included in total.

[5] Exports.

[6] Estimate.

[7] American and British Zones only.

[8] Recovered from an accumulation of refinery slimes.

[9] Year ended May 31 of year following that stated.

TABLE 4.—*Net industrial consumption of silver in the United States, 1943–47 (average) and 1948–52.*

[U. S. Bureau of the Mint]

Year	Silver (fine ounces)		
	Returned from industrial use	Issued for industrial use	Net industrial consumption
1943–47 (average)	44, 635, 252	154, 615, 252	109, 980, 000
1948	23, 897, 173	129, 186, 173	105, 289, 000
1949	22, 660, 459	110, 660, 459	88, 000, 000
1950	45, 257, 340	155, 257, 340	110, 000, 000
1951	46, 650, 905	151, 650, 905	105, 000, 000
1952	25, 038, 076	121, 538, 076	96, 500, 000

TABLE 5.—*Value of silver imported into and exported from the United States, 1948–52.*

[U. S. Department of Commerce]

	Imports	Exports	Excess of imports over exports
1948	$70, 884, 513	$12, 400, 060	$58, 484, 453
1949	73, 535, 694	23, 281, 043	50, 254, 651
1950	110, 035, 107	6, 201, 874	103, 833, 233
1951	103, 468, 510	8, 590, 185	94, 878, 325
1952	67, 296, 379	4, 921, 285	62, 375, 094

SELF-SUFFICIENCY AND STRATEGIC CONSIDERATIONS

Since the beginning of Government aid to silver in 1933, the domestic output of newly mined silver has been bought by the United States Treasury at prices fixed by an amendment to the act. Normally, the domestic supply of silver for industrial needs consists of imports and secondary production purchased at open-market prices; normally, also, these sources are adequate for trade requirements. Silver purchased by the Treasury enters Treasury holdings; most of it becomes a reserve to give backing to silver certificates. Normally, Treasury holdings include silver over that required by statutes for monetary reserves; such excess silver is permitted by law to be sold to industry at the Treasury price for silver, which normally is higher than the open-market price. Some Treasury silver was delivered in 1951 to relieve a tight trade supply (*6*).

Industrial uses of silver applied to military equipment and materials plus special military uses, such as silver in engine bearings, increased the consumption of silver during World War II far above normal quantities. Brazing solders were widely used for joining parts of ships, planes, tanks, guns, bombs, shells, rockets, torpedoes, and many other items; military photography absorbed large quantities of silver.

Silver was controlled by laws and War Production Board regulations during the war. In effect, Treasury silver was eligible for engine bearings, official military insignia, solders and brazing alloys, and identification neck chains; domestic silver was permitted in silverware, jewelry, watchcases, church articles, pens and pencils, mirrors, and other items classed as nonessential; and foreign silver was limited to photographic uses, electrical appliances, medical and dental supplies, and other items with preference ratings.

It seems logical to anticipate that, in an emergency, legislation would be enacted to release silver for military needs from the vast stocks in Treasury holdings. No shortage of silver in the United States is in prospect.

DEFENSE PROGRAM

The Defense Program has led to a considerable increase in the demand and consumption of silver. Data on the consumption by uses are not available, but the increase probably was shared by industrial and military uses and by the arts.

PRICES

The yearly average price for silver showed a fairly regular downtrend from $1.339 per fine troy ounce in 1866 to $0.507 in 1915. World War I caused the price to advance, but the decline was steady from 1923 to 1927. The average price increased about 2 cents an ounce in 1928 and decreased about 5 cents an ounce in 1929. The average yearly price decreased to $0.385 for 1930, $0.287 for 1931, and $0.279 for 1932. These prices were determined by the market. Since 1933, however, the price of newly mined domestic silver has been set by Government action, effected through the coinage law and the seigniorage deduction prescribed by Presidential proclamation and, in recent years, by act of Congress (31 U. S. C., secs. 316c, 316d). The coinage value of silver in the silver dollar is $1.2929+ per fine troy ounce of silver. That value is based on the standard weight of the silver dollar, which (except for the brief period between 1873 and 1878) has been 412½ grains troy of silver since 1837 (5 Stat. 136; R. S. sec. 3513; 20 Stat. 25, 31 U. S. C., sec. 316). Presidential Proclamation 2067 of December 21, 1933 (48 Stat. 1723), authorized the Treasury to acquire for coinage newly mined domestic silver with seigniorage deduction of 50 percent, thus fixing the price of silver at $0.6464 per fine troy ounce. The price thus fixed under the coinage-seigniorage law is known as the "Treasury price." Changes in the Treasury price have been as follows: Increased to $0.7111 on April 10, 1935; increased to $0.7757 on April 24, 1935; decreased to $0.6464 on January 1, 1938; increased to $0.7111 on July 1, 1939. The latest change in the Treasury price was effected by act of Congress (60 Stat. 750, 31 U. S. C., sec. 316d), on July 31, 1946, when the seigniorage deduction was set at 30 percent, thus increasing the Treasury price to $0.90505. The provisions of law (31 U. S. C., secs. 316d, 734b) authorizing the sale of silver from Treasury holdings to public purchasers in effect provides a ceiling on the domestic open-market price of silver. Such sales of silver are now made at the price of $0.90505 per fine troy ounce of silver.

The daily New York, or open-market silver quotation, reported by Handy & Harman is determined on the basis of actual sales of bar silver 0.999 (999/1,000) fine in lots of 50,000 troy ounces or more for nearby delivery, as reported daily by regular suppliers, and is usually one-fourth cent below the price paid for such bar silver as an allowance to suppliers for carrying, delivering, and marketing. In addition to foreign silver, the quotation also applies to domestic silver if such silver enters the New York market.

Average annual New York silver prices from 1931 to 1950 have been as follows, in cents per fine troy ounce:

1931	28.70	1942	38.33
1932	27.89	1943	44.75
1933	34.73	1944	44.75
1934	47.97	1945	51.93
1935	64.27	1946	80.15
1936	45.09	1947	71.82
1937	44.88	1948	74.36
1938	43.22	1949	71.93
1939	39.08	1950	74.17
1940	34.77	1951	89.37
1941	34.78	1952	84.94

Quotations in September 1954 were around $0.85 per ounce.

TAXES

Mines producing silver, and silver refineries and dealers, are subject to property taxes, income taxes, and excess-profit taxes. Dividend payments to stockholders are subject to income taxes. Jewelry and other luxury items made of silver are subject to luxury taxes; in some cities and States they are subject to sales taxes.

TARIFFS

No duties are imposed by the United States on imports of silver in ores, bullion, or coins, but silver manufactured goods are subject to duties ranging from 17½ to 55 percent ad valorem.

RESEARCH

Before World War II the major problem of the silver industry, particularly domestic producers, was the large surplus in silver output, which would have limited price recovery from the depression lows, except for the purchases of silver in the 1930's by the United States Treasury. Research to extend the uses and increase the consumption of silver was carried out on a large scale by the American Silver Research Project sponsored by the principal silver producers and dealers. The work performed under this project and the findings were set forth in a report published in 1940 (*1*). Fields of investigation reported upon include the following: Properties of silver, binary silver alloys, ternary and engineering alloys containing silver, technology of silver, low-temperature bonding of silver, high-temperature bonding of silver, the use of silver in bearings, silver coatings, chemical and evaporated silver coatings, silver in stationary electrical contacts, silver in moving electrical contacts, silver as a catalyst, corrosion resistance of silver and silver alloys, silver as a fungicide.

A broad digest of the conclusions resulting from the investigations follows:

The greatest hopes for extensive new consumption lie in the metallurgical field. Expansion of the use of pure-silver coatings as a corrosion-resistant surface of containers offers the most promise for early development. An ideal solution for increasing silver consumption would be the discovery that a small addition of silver to some widely used metal improves its qualities.

In chemistry, some data suggest that silver may have uses as a catalyst. Photography is the outstanding chemical use for silver, with further expansion probable, but consumption of silver is largely canceled by ultimate scrap recovery. Considered as reagents, no important uses for silver compounds have been found.

The field for silver-bearing electrical contacts is growing. A real invasion of the generator brush field by silver-bearing brushes may be in prospect.

Biological applications are inviting, but consumption possibilities are unknown. The antiseptic properties of silver are familiar, but a vast amount of work needs to be done to permit an intelligent appraisal.

In summary, it is concluded that further intensive research along the lines thus far followed will be needed to achieve greater use for silver.

OUTLOOK

CURRENT TRENDS

Unlike most other metals, the postwar production of silver in the United States and the world has been below prewar levels. Making the same comparison, the net consumption and average prices are higher. There appears to be no reason to question that these situations mark definite trends. The increase in postwar silver consumption was general with all silver products, but the major portion resulted from larger demand for silverware and other luxury items because of widespread prosperity—a condition expected to continue. The increasing proportion of silver being consumed in industry and the arts promises a more dependable demand than when politically controlled monetary uses predominated.

NEW USES AND NEW SOURCES IN PROSPECT

A storage battery employing zinc oxide and silver oxide in an electrolyte of potassium hydroxide, which has a high weight-capacity ratio and rapid discharge rate, has been developed and appears to have potentialities for special uses. No sources of silver additional to those already known are definitely in prospect. However, new developments at base-metal deposits containing byproduct silver now underway and expansion of operations at known deposits will contribute to the silver supply.

FORECAST OF SUPPLIES AND REQUIREMENTS

Continuance of international tension and maintenance of strong industrial activity would sustain the production and consumption of silver at levels somewhat higher than would otherwise be so. No shortage of silver need be anticipated for either the short- or long-term outlook; imports plus secondary production should be ample for all normal domestic requirements.

PROBLEMS

For many years before World War II, the chief problem of the silver-mining industry was a surplus production, but this phase appears to be coming to an end. The purchase of domestic newly mined silver by the Treasury and consequent support of the price of silver have disgruntled some silver fabricators and silverware merchants. Most of the silver output will continue to be derived as a byproduct of base-metal ores and will be affected by the prices of base metals. The fortunes of silver will depend largely on Government policies that determine the extent and price at which silver is used in the monetary system.

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SODIUM COMPOUNDS

By

Joseph C. Arundale[1]

SODIUM COMPOUNDS were known and used by the ancients. Today, as the source materials of a great alkali industry, they are a basic and inextricable part of the intricate texture of modern civilization's requirements for products and services. Their abundance urges greater utilization.

Summary

The two materials considered in this chapter are sodium carbonate and sodium sulfate. Salt (sodium chloride) and other compounds will be discussed elsewhere in this publication.

Sodium carbonate occurs naturally as the mineral trona ($Na_3H(CO_3)_2 \cdot 2H_2O$) and in solution in many saline waters. Sodium sulfate occurs principally as the minerals thenardite (Na_2SO_4), mirabilite (Glauber's salt, $Na_2SO_4 \cdot 10H_2O$), and glauberite ($Na_2SO_4 \cdot CaSO_4$) and in solution in saline waters. Domestic production of these minerals constitutes only a portion of the total output of sodium carbonate and sulfate. These natural sources will be emphasized in this chapter.

The principal domestic deposits occur as dry-lake encrustations and bedded deposits and in brines in the Western States. Sodium carbonate is produced in California and Wyoming and sodium sulfate in California, Wyoming, and Texas.

Sodium carbonate and sulfate are recovered by underground mining of bedded deposits or by fractional crystallization of brines by evaporation and temperature manipulation.

The uses of sodium carbonate include glass, ceramics, soap, detergents, cleaners, water softening, petroleum refining, aluminum production, textiles, pulp and paper, iron and steel, caustic soda, sodium nitrate, and miscellaneous other uses. Sodium sulfate is used in manufacturing kraft paper and paperboard and glass, in processing textile fibers, dyes, and soap, in tanning, in stock tonic and pharmaceuticals, and in other sodium compounds.

Requirements for both sodium carbonate and sulfate are expected to follow the upward trend of industrial activity.

Problems of the industry include some of a technical nature, involving recovery and refining of the products, and competition from manufactured materials.

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[1] Assistant chief, Branch of Construction and Chemical Materials, Bureau of Mines.

BACKGROUND

SODIUM CARBONATE

Sodium carbonate has been known and used since ancient times. Early Egyptians recovered it from incrustations around dried lakes. From early history until near the end of the 18th century seaweeds and marine plants were burned, and sodium carbonate was recovered from the ashes for use in lye, soap, and glass. In reference to this early source, sodium carbonate popularly is called "soda ash."

About 1791 Le Blanc in France developed a commercial process for producing soda ash, using sodium sulfate (made with salt and sulfuric acid), limestone, and coal. This process was widely used in Europe. In the 1860's the Solvay brothers in Belgium developed the process that bears their name and now is used in making most of the world's soda ash.

Anhydrous sodium carbonate does not occur in nature, but the mineral trona ($3Na_2O{\cdot}4CO_3{\cdot}5H_2O$) often is found in saline deposits, and sodium carbonate occurs in a dissolved state in many brines.

DOMESTIC DEPOSITS AND PRODUCTION

The United States has several large deposits of sodium carbonate; however, it is being produced currently only in Wyoming and California. An exploratory oil well drilled near Green River, Wyo., encountered a bed of trona up to 20 feet thick at a depth of about 1,500 feet. Several additional holes drilled in that area indicate a reserve of at least 300 million tons, and the lateral extent of the deposit has not been delineated. Crude trona has been mined from this deposit for several years, and recently a plant has been built to produce refined soda ash from this material.

Many of the western saline deposits and playas contain natural sodium carbonate. Both Searles and Owens Lakes in California have large reserves, and the remainder of the domestic output is produced from these two lakes.

Sodium carbonate recovered from natural sources represents only a small percentage (less than 10 percent) of the total supply of soda ash, as is shown in table 1.

TABLE 1.—*Manufactured sodium carbonate produced*[1] *and natural sodium carbonates sold or used by producers in the United States, 1946–53.*

Year	Manufactured soda ash (ammonia-soda process)[2] (short tons)	Natural sodium carbonates[3]	
		Short tons	Value
1946	4, 284, 231	215, 625	$3, 427, 086
1947	4, 524, 668	293, 051	5, 862, 178
1948	4, 575, 452	[4] 288, 769	[4] 6, 623, 280
1949	3, 916, 016	[4] 200, 496	[4] 4, 163, 714
1950	3, 991, 199	351, 075	7, 543, 769
1951	5, 093, 927	350, 688	8, 368, 037
1952	4, 442, 450	323, 479	7, 828, 033
1953	4, 879, 408	419, 206	10, 627, 460

[1] U. S. Bureau of Census.

[2] Total wet and dry (98–100 percent Na_2CO_3.) Includes quantities used in manufacturing caustic soda and sodium bicarbonate and quantities processed to finish light and finished dense soda ash.

[3] Soda ash and trona (sesquicarbonate).

[4] Exclusive of Wyoming.

TECHNOLOGY

There are several methods of recovering and refining sodium carbonate from brines, depending on the coproducts to be recovered. Three methods now in use are outlined below:

In one process, brine is pumped to towers and carbonated to sodium sesquicarbonate ($Na_2CO_3{\cdot}NaHCO_3{\cdot}2H_2O$) by flue gases from an oil-fired furnace. The crystals so precipitated are filtered off. The sodium sesquicarbonate is dried at a low temperature, using steam-heated air driers, or is calcined to soda ash in a furnace.

In a second process brine is carbonated in wooden towers with limekiln gas. The sodium bicarbonate formed is filtered off by means of rotary vacuum filters, and the bicarbonate is calcined in a rotary drier.

In another process a brine is partly evaporated to precipitate burkeite ($Na_2CO_3{\cdot}2Na_2SO_4$) crystals. The crystals are filtered and redissolved. Salt is added and the solution warmed and concentrated in vacuum crystallizers. The burkeite formed is filtered off. The remaining liquor is cooled to produce crystals of sal soda ($Na_2CO_3{\cdot}10H_2O$). The sal soda is filtered off and melted in its own water of crystallization. The solution is then evaporated; sodium carbonate monohydrate ($Na_2CO_3{\cdot}H_2O$) is recovered and dried in an oil-fired rotary drier.

Sodium carbonate is the only material recovered from the bedded deposits in Wyoming, but from the brine of Searles Lake, salt cake, potash, borax, bromine, and lithium compounds also are recovered.

Domestic facilities for manufacturing soda ash are at Syracuse, N. Y.; Detroit, Mich.; Baton Rouge, La.; Wyandotte, Mich.; Fairpoint, Ohio; Barberton, Ohio; Corpus Christi, Tex.; Saltville, Va.; and Lake Charles, La. All are situated at or near the raw material—salt or brine.

The only important process for manufacturing soda ash is the Solvay or ammonia-soda process. In this process natural brine, or brine formed by feeding water into a salt deposit, is pumped to an ammonia absorbing tower. Ammonia gas is fed into the tower and absorbed in the brine. The ammoniated brine is sent to a carbonating tower. As the brine flows down the tower carbon dioxide is blown through it. The resulting ammonium bicarbonate reacts with salt to form sodium bicarbonate, which is filtered and dried to produce a crude sodium carbonate. The carbon dioxide liberated by heat in the calciner is recovered and reused. Ammonium chloride, which is left in solution after the bicarbonate has been precipitated, is treated with slaked lime and distilled. The ammonium hydroxide so formed is converted by heat to ammonia gas and water and the ammonia distilled off. The ammonia is recovered and recycled.

The crude sodium bicarbonate is converted to a "light ash" (32 to 35 pounds per cubic foot) by washing and drying with recycled soda ash (dry) under controlled temperature. "Dense ash" is made by moistening light ash with a sodium carbonate solution and drying in a rotary kiln to form $Na_2CO_3 \cdot H_2O$ (64 to 67 pounds per cubic foot).

USES

Soda ash is one of the basic industrial chemicals and the most important product of the huge alkali industry. It enters directly into the production of glass, soap, detergents, cleaners, water softening, petroleum refining, aluminum production, textiles, pulp and paper, iron and steel, sodium nitrate, caustic soda, and miscellaneous other chemicals.

The common types of flat, container, and tableware glasses are largely silica and alkali. Addition of soda ash (usually about 15 percent of the batch) causes easier melting and results in a glass that can be blown or formed into shape more readily; for many years this has been the largest single use for soda ash, representing about one-fourth of the total consumption.

Nearly half of the soda-ash supply is used in making caustic soda (sodium hydroxide, NaOH), sodium bicarbonate, sodium nitrate, and a long list of other compounds, which in turn have uses too numerous to list.

Soda ash or caustic soda constitutes nearly 50 percent of some soaps, detergents, and cleansers.

Soda ash is used for removing certain undesirable salts in industrial waters. In the petroleum industry soda ash is used in the sulfuric acid treatment of petroleum derivatives, which improves color and chemical stability.

Soda ash is used in extracting aluminum, chromium, vanadium, radium, and uranium. In ferrous metallurgy it is used as flux and in quenching baths.

In the manufacture of pulp and paper soda ash is used in controlling the causticity of cooking liquor, regenerating recovered alkali, and preparing resin size.

The consumption of soda ash, by end uses, is shown in table 2.

TABLE 2.—*Soda-ash consumption in the United States, 1950–53, by end uses* [1]

[Thousand tons]

	1950	1951	1952	1953
Glass	1, 225	1, 640	1, 410	1, 590
Caustic and bicarb	700	994	766	790
Nonferrous metals	245	333	320	430
Pulp and paper	200	320	305	330
Soap	105	120	115	101
Cleansers [2]	110	142	135	160
Water softeners	100	105	95	120
Textiles	65	56	39	40
Exports	50	152	106	160
Petroleum refining	24	29	31	33
Other chemicals	1, 050	1, 253	1, 180	1, 350
Miscellaneous	151	296	262	226
Total	4, 025	5, 440	4, 764	5, 330

[1] Estimated by Chemical Engineering, March 1954.
[2] Includes modified sodas.

Salt cake, borax, and natural alkali minerals are used to some extent as substitutes for soda ash in glass manufacture. Caustic soda or potassium carbonate may be substituted in some applications, but both are more expensive and would be substituted only if soda ash were not available.

Most countries of the world have a requirement for alkali. In some less industrially advanced countries it may be used only in glass or soap. There is a considerable international trade in soda ash; Germany, France, United Kingdom, and Kenya are the principal exporters. The first three produce manufactured soda ash; natural soda ash is recovered from the brine of Lake Magadi in Kenya. United States exports are small and imports almost negligible, except in times of temporary shortages. The bulk of exports goes to Mexico and Brazil.

Dense soda ash is quoted at $1.75 per 100 pounds, paper bags, carlots, works; light soda ash is quoted at $1.65 per 100 pounds, same basis.

SODIUM SULFATE

The principal sodium sulfate minerals found in nature are thenardite (Na_2SO_4), mirabilite (Glauber's salt, $Na_2SO_4 \cdot 10H_2O$), and glauberite ($Na_2SO_4 \cdot CaSO_4$). Sodium sulfate also occurs in several double salts and complex minerals. Many natural waters, including sea water, contain sodium sulfate; but in the United States commercial or potentially commercial deposits are confined to the dry areas of the West, where it occurs as surface salts or saline concentrations.

DOMESTIC DEPOSITS AND PRODUCTION

Important domestic deposits are as follows:

Arizona —Thenardite in Yavapai County.
California —Mirabilite in the brine and thenardite in the solid salts of Searles Lake; mirabilite in Soda Lake, San Luis Obispo County; thenardite-bloedite ($Na_2SO_4 \cdot MgSO_4 \cdot 4H_2O$) deposit in Imperial County; Glauber's salt in Dale Lake.
Nevada —Thenardite in Rhodes Marsh, Mineral County.
New Mexico —Dona Ana and Eddy Counties.
North Dakota—A series of lakes in the northwest corner of the State (a few extend into eastern Montana).
Oregon —Several saline deposits in the southeastern part of the State.
Texas —Bedded deposit and Soda Lake near Monahans and bedded deposits in the Panhandle.
Utah —Mirabilite in Great Salt Lake and the salt beds surrounding it.
Washington —Many small lakes and saline deposits.
Wyoming —A chain of lakes from near Laramie to north of Casper.

Although many of these occurrences have not been thoroughly explored and no accurate estimate of reserves is available, in aggregate, probably hundreds of millions of tons of sodium sulfate is available. Some of these deposits are being replenished by runoff water trapped and evaporated in basins. Sodium sulfate is being produced from only a few of these sources. Production from others has proved uneconomic by reason of the nature or location of the deposit.

The tonnage of natural sodium sulfate sold or used by these producers is shown in table 3.

In addition to domestic occurrences, similar deposits are known to occur in many foreign countries. The most important to the United States are the saline deposits of Saskatchewan Province, Canada. A considerable tonnage of sodium sulfate has been imported from Canada.

TABLE 3.—*Production and sales of sodium sulfate by producers in the United States, 1946–53*

Year	Production, manufactured [1] and natural (short tons)			Sold or used by producers (natural only)	
	Salt cake (crude)	Glauber's salt (100 percent $Na_2SO_4 \cdot 10H_2O$)	Anhydrous refined (100 percent Na_2SO_4)	Short tons [2]	Value
1946	527, 746	167, 153	122, 573	198, 781	$1, 695, 413
1947	693, 517	202, 285	134, 969	257, 294	3, 329, 094
1948	668, 246	184, 744	169, 018	265, 862	4, 248, 613
1949	537, 843	156, 634	136, 276	186, 223	2, 733, 853
1950	561, 395	185, 626	184, 254	186, 537	2, 199, 336
1951	707, 388	219, 942	233, 666	(3)	(3)
1952	662, 373	177, 929	202, 813	236, 825	3, 217, 000
1953	734, 024	204, 159	219, 937	248, 230	3, 340, 760

[1] U. S. Bureau of the Census.
[2] Includes Glauber's salt converted to 100-percent Na_2SO_4 basis.
[3] Bureau of Mines not at liberty to publish figures separately.

Sodium sulfate is produced and marketed in several forms. Crude salt cake (Na_2SO_4) usually is 92 to 99 percent pure; Glauber's salt is the decahydrate of sodium sulfate ($Na_2SO_4 \cdot 10H_2O$); and anhydrous sodium sulfate is the refined and dehydrated product. The term "sodium sulfate" will be used when referring to all types.

TECHNOLOGY

There are several methods of producing and processing these products. Some bedded deposits have been mined by underground room-and-pillar methods or in open pits. The product can be sold without purification for some purposes or can be further processed. Sodium sulfate can be recovered and purified by evaporating natural sulfate brines, or brines prepared by dissolving impure crystals in water, either by solar evaporation where conditions of climate are suitable or by artificial evaporation.

The brine from Searles Lake, which contains a number of salts in solution, is pumped to a triple-effect evaporator. Heating and evaporation of the brine cause precipitation of burkeite (double salt of sodium carbonate and sulfate) and some sodium chloride and lithium-sodium phosphate. These salts are removed by elutriation. From the remaining liquor, potassium and boron compounds are recovered in later steps. Sodium chloride is separated from burkeite by elutriation equipment acting as hydraulic classifiers. Lithium-sodium phosphate is recovered by flotation. The burkeite precipitate is filtered. The burkeite filter cake is pulped and leached for free sodium carbonate.

The burkeite solids are dissolved in water and cooled to precipitate Glauber's salt. These crystals are mixed with sodium chloride, and anhydrous sodium sulfate is produced by dissolving the salt in the water of hydration of the Glauber's salt. This product is filtered and dried in rotary driers. Some operations have taken advantage of climatic conditions and the inverse solubility of sodium sulfate. In such operations natural brines are exposed to warm, dry air during the day or the summer season and to lower temperatures during the night or the winter season. This results in concentration of the brine and precipitation of mirabilite (Glauber's salt) at temperatures below 33° C. Evaporation can be carried to completion and the crystals harvested and dried.

Manufactured sodium sulfate is produced with coproduct hydrochloric acid by the reaction of salt (NaCl) and sulfuric acid in Mannheim furnaces. It also is recovered in substantial quantities as a byproduct in the manufacture of rayon, phenol, bichromates, boric acid, lithium carbonate, cellophane, and formic acid.

Sodium sulfate enters the market as crude salt cake, Glauber's salt, or anhydrous sodium sulfate. Each is preferred for certain uses, although some substitution is possible. The largest single use for sodium sulfate is in the manufacture of kraft paper and paperboard. In this process wood is cooked in a digester with caustic soda, sodium sulfide, and other chemicals. The sodium sulfide is produced by heating sodium sulfate with the wood. The cooking dissolves the material which binds the wood fibers, which subsequently are used to form kraft wrapping paper and boxboard. The sodium sulfide is reusable, but losses in the cycle require addition of new sodium sulfate at the rate of about 180 pounds per ton of paper.

USES

Sodium sulfate is used in producing various chemical compounds. In addition to sodium sulfide, they include barium sulfate and sodium-aluminum sulfate.

As much as half of the alkali in glass batches can be introduced as sodium sulfate, although standard formulas usually call for about 5 percent for fining and scum prevention. Excessive use increases the required temperature of the melt and injures the furnace refractories.

Sodium sulfate is used in processing rayon, wool, and other textile fibers, in dyes, coal tar, soap, leather tanning, stock tonic, pharmaceuticals, and numerous other miscellaneous uses.

Crude salt cake, anhydrous sodium sulfate, and Glauber's salt can be used interchangeably in some applications, but each is preferred for certain purposes. For example, anhydrous sodium sulfate is used in glass because of the absence of water and impurities. Salt cake is used in making kraft pulp principally because it is cheaper.

Glauber's salt is quoted at $45 per short ton, crystals, bags, carlots, works, freight allowed. Anhydrous sodium sulfate, bags, carlots, delivered, is quoted at $2.50 per 100 pounds. Salt cake, bulk, works, 100 percent Na_2SO_4 basis, is quoted at $24 per ton.

OUTLOOK

Sodium carbonate and sodium sulfate in their various forms are basic industrial chemicals and enter into a multitude of products and services. Therefore, requirements are expected to reflect the upward trend expected in industrial activity. However, this can be said of sodium carbonate with more confidence than of sodium sulfate. Some specific probable increases include:

(1) Some of the new wood-pulping plants built in the past few years utilize soda ash rather than salt cake in the cooking process.

(2) The expanding synthetic detergent industry is using more phosphates made with soda ash, and these phosphates are also finding wider use in water treatment.

(3) Shortages of tin in emergencies may force the use of more glass containers and thereby increase the use of both sodium carbonate and sodium sulfate.

The supply situation is somewhat confused. Production facilities for both manufactured and natural alkalies have been expanded. One of the largest uses for soda ash is for conversion to caustic soda. New chlorine plants are largely of the electrolytic type, with caustic soda as a byproduct. Disposal of increasing supplies of such material undoubtedly has released soda ash for other purposes. This trend also has affected the production of hydrochloric acid from Mannheim furnaces and thereby reduced supplies of byproduct salt cake from this source.

PROBLEMS

One of the principal problems of the natural sodium carbonate and sodium sulfate industries is competition from manufactured material. As regards natural sodium carbonate, soda ash made by the Solvay process is the competitor. The raw materials salt, limestone, and coal are widely distributed, and Solvay plants can be established near centers of consumption. Natural soda ash can compete only within a limited market area, except during periods of nationwide shortage. The same is true of sodium sulfate. Manufactured and byproduct material is made nearer the more important industrial areas than the natural sodium sulfate occurrences in the Western States.

Certain technical problems affect the efficiency of recovery of the natural products and their quality; they are largely specific problems relating to a particular deposit. Continued research is needed on the technology of recovery of sodium compounds and possible byproducts.

Since sodium-compounds resources are ample, an obvious problem is the development of new and expanded uses for them.

Detailed statistical and economic data are needed to guide industry and Government agencies.

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STEEL

By

James C. O. Harris [1]

OF ALL the metals in use none is as important as steel; there is scarcely an article in daily use that is not made from it or by it. The quantity of steel produced, whether measured for the United States or for the world as a whole, is about 20 times that of all other metals combined. One-eighth of the labor force of the United States is engaged directly in producing, transporting, and distributing iron and steel products. Approximately 25 percent of the total freight in the United States consists of raw materials used in the production of steel and the products of the iron and steel industry.

Summary

The history of steelmaking in the United States has seen successive developments of processes that, through economic and technical advantages, have dominated production at any given time. This chapter presents an account of these developments, including size, location, concentration, and capitalization of the industry. Transportation, prices, controls, production, and distribution are also discussed.

The 74.9 million net tons of pig iron produced in 1953, along with 77 million tons of ferrous scrap, contributed to the production of 111.6 million net tons of steel ingots and steel for castings by ingot producers (*2*).[2]

Type of furnace:	*United States production, 1953 million net tons*	*Percent*
Open hearth—basic	99.8	89.4
Open hearth—acid	.7	.6
Bessemer	3.8	3.4
Electric [1]	7.3	6.6
Total	111.6	100.0

[1] Includes small quantity of crucible steel.

In addition to the above, 16.4 million net tons of castings (90 percent iron castings) was produced by foundries and steel plants (see p. 809).

The production of alloy-steel ingots and castings totaled 10 million net tons, 66 percent of which was produced in open-hearth furnaces and the remaining 34 percent in electric furnaces.

The use of steel in making finished products is universal enough to enable the allocation of steel to be one of the chief means of controlling the distribution of many other raw materials in emergency periods.

Since the United States would not be self-sufficient in an emergency, the raw materials for ferrous alloys and additive metals are being stockpiled because of their importance to the steel industry.

The annual capacity for steel production was expanded from 99 million net tons in 1950 to 124 million tons by the end of 1953, largely with the assistance of Necessity Certificates issued by the Defense Production Administration and Office of Defense Mobilization.

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[1] Commodity-industry analyst, Bureau of Mines.

[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

According to the Research and Development Board of the Department of Defense, only an estimated 3 percent of the 4 billion dollars spent in 1953 for scientific and industrial research in the United States was devoted to research on the primary metals, and steel took only a small part of this. Expenditures for research have heretofore been small because of the large supply of ores and coke of high quality and uniform grade, but a current decline in the quality of domestic ore and coke and increasing reliance on foreign ores will require increasing attention to technologic research. Such technical advancement is necessary to maintain the strong competitive position of the United States in steel, to reduce the consumption rates for all materials per ton of steel ingots produced, to improve the utilization of ore and coke of lower quality, to find possible substitutes for less ample materials and materials of foreign origin, and to improve methods and facilities for producing steel.

BACKGROUND

HISTORY OF STEEL INDUSTRY

Before the invention of the Bessemer-converter process in 1856 only two methods for making steel—the cementation and the crucible processes—had been developed. History tells us that both of them were certainly known and practiced by the ancients. The Wootz steel of India, the famous blades of Damascus, Syria, and the equally famous steels of Toledo, Spain, were all made by the crucible process or a combination of the cementation and crucible processes. During the Middle Ages both methods appear to have been lost to civilization. The cementation process was revived in Belgium about 1600, and Benjamin Huntsman of England rediscovered the crucible process in 1742. The first recorded steel production in the United States was the 917 tons of "blister" steel made in 1810 from bar iron. In 1832 the Girrard Bros. established a crucible-steel works at Cincinnati. By 1867 three Bessemer steelworks were operating in the United States, and the 3,000 tons of steel that they produced may be considered the beginning of the American steel industry. Although this process bears the name of Henry Bessemer of England, with a patent issued in 1856, Henry Kelley had experimented with the process as early as 1847 in the United States (*8*). The next great step in the creation of the steel industry was invention of the open-hearth process by Karl Wilhelm Siemens of England in 1868. Although Bessemer-steel production exceeded open-hearth until 1908, open-hearth steel production provides about 90 percent of the total today. Next to the open hearth in the production of steel is the electric-arc furnace. The earliest instance of application of electricity to metallurgy is a patent granted on March 16, 1853 to the French chemist, Pichon. The first practical application of electric melting of steel was made by Sir William Siemens in England in 1879. Following Siemens' method came the French Heroult electric-arc furnace in 1888–92. The production of electric-furnace steel has increased each year since it was introduced into this country about 1905–10. Some have predicted that the electric-furnace portion of total carbon-steel production may increase from the 1953, 3.7-percent figure to 30 percent by 1975.

SIZE AND ORGANIZATION OF STEEL INDUSTRY

The iron and steel industry, including rolling mills, is comprised of 255 companies, employing 650,000 salary and production workers, with an annual payroll of $3.2 billion (*2*, *3*).

These companies have 260 blast furnaces, 1,223 steel furnaces, and a total estimated investment of approximately $8 billion.

There is a high degree of integration to achieve economies in transportation of materials and in heating and shaping, with resulting savings in fuel, power, and labor costs. Most single companies own their mining, pig iron, steel, and fabricating facilities. In 1948, only 7 percent of the pig iron produced was consumed by the companies that did not own a pig-iron source (table 1).

TABLE 1.—*Organization of the iron and steel industry, 1948*

	Number of companies	Pig-iron capacity (percent)	Steel capacity (percent)	Finished hot-rolled product capacity (percent)
Fully integrated	19	93	90	88
Semi-integrated	59	----------	10	9
Nonintegrated	163	----------	----------	3
Merchant furnace	16	7	----------	----------
Total	257	100	100	100

SOURCE: Alderfer, E. B., and Miehl, H. E., Economics of American Industry. McGraw-Hill Book Co., Inc., New York, N. Y., 1950, p. 30

DEGREE OF GEOGRAPHIC CONCENTRATION OF PRODUCTION

With the United States furnishing almost half of the world output of steel, it is particularly significant that over 90 percent of the United States steel output is produced in a comparatively small area in the northeastern section of the country. Within this area are two heavy concentrations. One in the Pittsburgh-Youngstown region, including parts of Pennsylvania and Ohio, which supplies 32 percent of the total output, and another in the Chicago region, which supplies 20 percent of the total. Although the percentage for the latter includes all of Illinois and Indiana, the large majority of the steel plants of this region are in the vicinity of South Chicago. To illustrate more clearly the degree of concentration, Kentucky, West Virginia, Minnesota, and Missouri, which are included in American Iron and Steel Institute districts for these areas, are omitted.

RELATION OF TRANSPORTATION TO PLANT LOCATION

The value of all raw materials and most of the products of the iron and steel industry is low in proportion to their weight. It is therefore true that the cost of transportation, more than any other single factor, determines the location of producing plants in relation to sources of raw materials and consuming markets. The development of excellent rail, truck, and water transportation in the United States has made possible the movement of materials and products of the iron and steel industry over considerable distances at relatively low cost.

Ohio, western Pennsylvania, Indiana, and Illinois produce about 60 percent of all steel manufactured in the United States. Steel plants in this area obtain their ore from the Lake Superior mines. Pittsburgh has the advantage of being close to a steel market, with coal, scrap, and fluxing limestone close at hand. In 1954 the total transportation cost (including loading and unloading) for iron ore for Pittsburgh was $5.13 per long ton. In addition to the above, the markets in these areas are protected from competition with foreign steel by the prohibitive cost of rail transportation from ports of entry.

The Atlantic coast mills (Morrisville, Pa., and Sparrows Point, Md.) use principally imported ores and distribute the bulk of their products via ocean transportation. Coal and fluxing stone are available to these plants from sources less than 300 miles away. These plants also have the advantage of being in a favorable position to compete in foreign markets.

Plants in the Birmingham, Ala., district have the special advantage of coal, limestone, and iron ore within a 25-mile radius. Because of the relatively low grade ore in the Birmingham district, facilities have been constructed at Mobile, Ala., to reload foreign ore for transportation to Birmingham. The increased transportation cost of such high-iron-content ores over local ores will be offset in part by increased pig-iron production, with accompanying reduction in unit consumption of coke. Favorable production plus the ready availability of inland waterway and rail facilities connecting the area with Mobile puts Birmingham in a position to supply coastal and export markets as well as some northern and southwestern markets by rail.

Before World War II only limited quantities of steel were produced in the West, the bulk of the steel for Pacific coast consumption being shipped from eastern mills by intercoastal vessels. During the war two integrated plants were constructed in the West, chiefly to supply steel for war industries on the coast. These plants (at Geneva, Utah, and Fontana, Calif.) are supplied with raw materials, except coal, from local sources. Both plants are using high-volatile Utah coal mixed with low-volatile coal from Oklahoma and Arkansas.

RELATIVE FREIGHT COSTS

The low American freight rates that now apply to large quantities of raw materials presumably reflect the economies of bulk transport in the high-weight-capacity cars used for this traffic. A comparison of these low rates with the higher (and still rising) freight charges for finished steel helps to explain the tendency for new steel works in the United States to be located increasingly according to markets instead of, as formerly, being governed by the cost of assembling raw materials.

PRICES AND COSTS IN THE IRON AND STEEL INDUSTRY

HISTORY OF BASING-POINT SYSTEM

Adoption of the basing-point system greatly influenced the course of iron and steel prices. This system of price quoting was used for all iron and steel products except carbon-steel rerolling ingots and steel rails, which were quoted f. o. b. mill.

Under the basing-point system a base quotation was given for an article (usually the lowest priced variety of the article) in a given area; this standard was used for delivery of a minimum specified quantity at a point designated by the industry as the price-basing point for the area. Extras were added to cover smaller orders, and specifications more difficult to meet than those covered by the base price. The "extra book" also included deductions applying to specifications less rigid than those covered by the base price. Extras and deductions, arrived at by agreement among producers, were uniform for each specification. For delivery to destinations other than the basing point, the cost of transporting the article from the basing point to the point of delivery specified in the order was added.

Until 1924 Pittsburgh was the only basing point, and products were quoted on a Pittsburgh-plus basis. At that time the Federal Trade Commission ordered certain companies to cease using the Pittsburgh-plus system, and from that period on a greater number of basing points was established and the system extended to cover more products.

On April 26, 1948, a decision by the United States Supreme Court, against cement companies using the multiple-basing-point system caused the steel industry to desert this practice. The industry now uses an f. o. b. mill price, as it did before 1880.

IRON AND STEEL PRICES IN 1953

Composite finished steel	$0. 04518 per lb.
Weighted average value of finished steel	. 06762 per lb.
Pig-iron composite	55. 96 per net ton (*11*).
Steel scrap, Heavy-Melting, composite.	39. 52 per long ton (*11*).

The composite finished steel price, as quoted from the Iron Age, is computed on a base of the following 10 carbon-steel products: Hot-rolled bars, structural shapes, plates, rails, pipe, wire, and hot- and cold-rolled sheets and strip.

The weighted average value of finished steel quoted above was computed for all grades and all products using values published by the United States Department of Commerce.

Scrap prices fluctuate more widely than those of pig iron, although scrap is pig iron's competitor. The drop in scrap prices in 1929–32 was 54 percent against a 24-percent drop for pig-iron prices in the same period. Scrap prices are sensitive and reflect changes in general business activity. However, high prices have small effect on total supply when the limit is near. As experienced during World War II, the quantity of marketable, high-grade, steelmaking scrap obtainable at any given time is limited.

COST STRUCTURE OF STEELMAKING

The end product value (*5*) is the sum of all four components of the cost structure—cost of raw materials, fuel (and other forms of energy), wages, and depreciation and other overhead. An approximate distribution of cost is given in table 2. This table represents a hypothetical steel works, but it is unlikely that any actual works will have the particular distribution shown because of the wide variations in conditions which exist.

The total labor component generally represents about 25 percent of the total cost of making a heavy steel product, part of which is not included in table 2. The table compares cost of delivery of steel products but does not show the additional transport component in the cost of materials and fuels. These components of cost, so closely interlocking, must be shown separately, before the problems of the industry can be analyzed. In each component costs could be cut, but only by increasing expenses somewhere else, such as by using cheaper ore at the expense of greater coke consumption.

TABLE 2.—*Approximate distribution of costs (including taxes and return on capital) in making steel in the United States, 1950* [1]

	Percentage of grand total	Percentage of sub-total
Ore, scrap, and other materials	14. 0	43
Fuel (mainly coke)	12. 5	38
Labor	1. 5	4
Other	5. 0	15
Total for iron	33. 0	100
Iron	(33. 0)	41
Scrap, ferroalloys, and other materials	29. 5	37
Fuel	2. 5	3
Labor	4. 5	6
Other	10. 5	13
Total for steel ingots	80. 0	100
Ingots used	(80. 0)	
Circulating-scrap credit	=12. 0	
Fuel	1. 0	
Labor	11. 0	
Transportation (delivery)	8. 0	
Other	12. 0	
Total for heavy steel products	100. 0	

[1] Other materials include limestone, oxides, refractories, etc. Fuel includes power and allows for gas credits. Labor includes direct and indirect productive and maintenance labor only and represents about three-quarters of the total cost of labor in a works. *Balance includes all other items of cost, taxes, profits ploughed back and dividends. The values are derived.* Source: Anglo-American Council on Productivity, Iron and Steel, Productivity Team Report. June 1952, p. 11.

EXPENDITURE FOR EXPANSION AND IMPROVEMENT

In 1953 over $1 billion was spent by steel companies to expand and improve their facilities. In the past 8 years—1946–53—over $5.6 billion has been invested. This figure exceeds the capital investment of the entire industry as of 8 years ago. The following lists the year-by-year (1946–54) expenditures by the industry for expansion and improvement:

Year:	*Dollar expenditures*
1954	[1] 775, 000, 000
1953	1, 000, 000, 000
1952	1, 170, 000, 000
1951	1, 041, 000, 000
1950	513, 000, 000
1949	510, 000, 000
1948	583, 000, 000
1947	496, 000, 000
1946	291, 000, 000

[1] Advance estimate.

SOURCE: Steel Facts, February 1954.

This capital investment brings the steel-industry capacity to 124,330,410 net tons as of January 1, 1954, an increase of 35.3 percent over the 1946 capacity of 91,890,560. Steel-expansion cost since 1946 is estimated at $173 per net ton of capacity. Some ingot capacity has been developed for as little as $16 per ton, although a new, completely integrated mill may run as high as $300 per net ton of capacity.

To encourage the iron and steel industry to reach a capacity of 120 million net tons (a goal that has been surpassed) the Defense Production Authority granted applications for accelerated tax amortization. The granting of such a Certificate of Necessity permitted the applicant to charge as an expense all or part of the expansion cost within a 5-year period in computing profits for purposes of taxation.

By April 1954 DPA and ODM had approved a total of $5.9 billion in Certificates of Necessity for the iron and steel industry—20 percent of the value of such certificates issued to all industries combined; however, this entire amount is not subject to the accelerated tax amortization—only that portion deemed vital for defense (60 to 75 percent). These applications referred to expansion of capacity of steelworks and rolling mills ($2.7 billion), blast furnaces ($1.0 billion), and iron-ore projects ($1.1 billion); 1,325 project certificates have been issued to this industry, including coal and coke, ferroalloys and iron ores, foundries, pipe mills, and lake and ocean transportation relating to the iron and steel industry.

Under present conditions operations can now drop to 74 percent of capacity and still produce as much steel as 100-percent operations would have produced in 1946.

COST OF BUILDING A STEEL MILL

New steel-plant construction costs several times as much (per ingot ton of capacity) as it did before World War II. According to statements by several executives at a hearing before the Wage Stabilization Board in 1952, the original cost of 1 company over a long period averaged $77 per ingot ton compared with an estimated cost exceeding $200 in 1952.

United States Steel's new Fairless plant at Morrisville, Pa., cost $250 to $275 per ingot ton of capacity. This figure includes all finishing facilities for a fully integrated plant.

COST OF INDUSTRY

In data covering 94.5 percent of the iron and steel industry for 1953 (105.5 million tons of ingots and steel for castings), involving 53 companies employing 623,351 workers, the net billing value of products shipped (including other services and other income) amounted to $13.2 billion. The total expense (including taxes) amounted to $12.4 billion, and the remaining $800 million was divided between dividends and undistributed profits. Included in the expenses was $54.0 million of interest and charges on long term debt (*2*).

Expenses involved more directly in production were:

	Million
Employment costs	$4,485
Materials, supplies, freight and other services, etc	6,097
Depreciation, depletion and amortization	620
Taxes—Federal, State, and local	1,200

Some of the labor was involved in rolling mills, etc., and the billings include shipments of finished-steel products. Thus a direct relationship of the expenditures to the tonnage of steel for ingots and castings would not be applicable.

Relating the expenditures to billings, employment costs represented 34.0 percent; materials, supplies, freight, etc., represented 46.2 percent; and depreciation, depletion, and amortization represented 4.7 percent. Taxes represented 9.0 percent.

Capitalization as of December 31, 1953, was comprised of:

	Million
Earned surplus	$3,579
Capital surplus	530
Preferred stock	702
Common stock	1,970
Total stock and surplus	6,781
Total long-term debt	1,327
	8,108

TABLE 3.—*Employment costs* [1]

[Million dollars]

Year	Wages and salaries	Social security and other employment benefits	Pensions	Total
1943	2,551	75	41	2,667
1944	2,624	64	41	2,729
1945	2,318	54	42	2,414
1946	1,921	42	20	1,983
1947	2,383	54	27	2,464
1948	2,730	74	28	2,832
1949	2,489	71	41	2,601
1950	2,852	124	175	3,151
1951	3,495	149	185	3,829
1952	3,474	142	173	3,789
1953	4,093	165	219	4,477

[1] These data are from consolidated financial reports of companies producing more than 90 percent of industry's output of steel ingots.

SOURCE: American Iron and Steel Institute, Annual Statistical Report.

TAXES

Federal taxes (*3*) on the 1953 income of the iron and steel companies (representing 53 companies) was $1.0 billion, or 7.6 cents on each sales dollar.

Local, State, and miscellaneous taxes incurred by the iron and steel companies in 1953 were over $192 million.

The total of Federal, State, and local government taxes paid by these companies was over

$1.2 billion, exclusive of social security and similar employee benefits, or 9 cents on each sales dollar.

GOVERNMENT CONTROLS

To meet both civilian and military demands during the Korean emergency, it was necessary to allocate steel among the users in a manner that would be most beneficial to the defense efforts of the United States. Steel was one of the metals of the Controlled Materials Plan; the other two were aluminum and copper. The Controlled Materials Plan was derived from the concept that, since steel, copper, and aluminum were used so universally in manufacturing end products, to control these was to control indirectly all other materials which are used along with these metals. Regulations issued, specific to iron and steel, were as follows:

M-1. *Steel;* governed the steel mills as to production of steel-mill products. Among other things, it established the lead times required for placing orders for the different steel-mill products.

M-6. (later M-6A). *Steel Distributors;* governed the receipt and distribution of steel-mill products by distributors.

M-20. *Iron and Steel Scrap;* as originally issued, established a 60-day inventory limitation on steel scrap. Later, it was amended to permit the National Production Authority to direct the shipment of scrap held by scrap dealers to selected users. The regulation also contained certain technical provisions, such as segregation of different types of scrap.

M-64. *Used Rails, Used Axles, and Used Cast-Iron Car Wheels;* required, subject to certain exceptions, distribution of unserviceable axles to bar mills and forge shops, distribution of unserviceable rails to rerolling mills and foundries, and distribution of scrap cast-iron wheels for use in producing new cast-iron wheels.

M-80. *Iron and Steel, Alloying Materials and Alloy Products;* required all processors and melters to prepare and submit to NPA melting or processing schedules and reports on their inventories of these materials. Revisions of the order were issued from time to time governing allocation or use limitation of the individual alloying material.

M-92. *Automobile Wreckers*; required auto wreckers to report inventories of motor vehicles and loose scrap and limited their acceptance of delivery of motor vehicles or car-units.

All above controls were revoked by November 1, 1953.

TARIFF

Pig Iron.—Ever since 1909 tariff provisions on pig iron have applied, except for a period between 1913 and 1922 when it entered duty free:

Per long ton:

$2.50	Act of 1909, par. 118.
Free	Act of 1913, par. 518.
.75	Act of 1922, par. 301.
1.125	Presidential proclamation under Tariff Act of 1922, section 315 (a), effective March 25, 1927.
1.125	Act of 1930, par. 301.
.75	Trade agreement with United Kingdom, effective Jan. 1, 1939, this reduction being applicable only to iron containing 0.04 percent or less of phosphorus; higher phosphorus iron remains dutiable at $1.125 per long ton.
.60	Effective since June 6, 1951.

In the Act of 1930, as amended, there is provision for additional duty rates on the alloy content of iron: (Whether due to this duty or not, imports of alloyed pig iron are very small.)

Per lb.:
$0.50—Vanadium content, over 0.1 percent.
.50—Tungsten content, over 0.2 percent.
.35—Molybdenum content, over 0.2 percent.
.015—Chromium content, over 0.2 percent.

Iron and Steel Scrap.—The duty on ferrous scrap is $0.375 per ton; however, it was suspended March 14, 1942, to June 30, 1949, again on October 1, 1950, and, with yearly suspension since 1950, it will be duty-free until June 30, 1955.

SECURITY AND SELF-SUFFICIENCY

Attacks on the German steel industry during World War II have proved that plants, furnaces, and mills of this industry can survive considerable bombing, using World War II standards. The German iron, steel, and coal industries were still able to operate at 92 percent of capacity when the Allies took over in 1945. German steel production buckled only when the Allies began bombing supply lines. Supply lines are, literally, the life lines of the industry, since the many and vast quantities of materials necessary for steel production must be transported from various sources to the plants and the finished products shipped to consuming industries. The atom bomb could be much more destructive to the plant itself, but since the supply lines are so vital the main problem still lies in means of protecting them in this country.

Steelmaking facilities here are not particularly vulnerable to enemy attack because of distances between plants. Only a small fraction of the steelmaking capacity is in any one target area. New technologic developments may cause further decentralization.

A serious threat to the security of the steel industry would lie in the curtailment, by hostile action, of iron-ore movement. The United States would not be able to depend on imports of African, South American, and Swedish ore, the three main sources of overseas supply. Therefore, two of the major problems are practical utilization of low-grade domestic ores and protection and improvement of American supply lines. Completion of the St. Lawrence Seaway will afford a safer and cheaper route for the ores recently developed in Labrador. Cargos in that river could be more readily protected from submarine attack than those in the open sea.

In addition to iron ore and its transportation, fuels, fluxes, scrap, other ores, additive alloys, and power are essential to the steel industry. There are adequate domestic supplies of coking coal, oil, and gas to take care of the needs of the steel industry, as well as fluxes, except fluorspar, which is included in the National Stockpile. With a sound scrap program the supply of this important raw material should be sufficient for the needs of the industry. The prices of scrap and pig iron and their relative availability will determine the balance between the use of these raw materials.

Adequate procurement and stockpiling of critical and strategic ores and additive alloys are well underway. Supplies of manganese, chromium, and many other useful and rare ores are now fairly secure (*23*). Public utilities supply about two-thirds of the total power consumed by the steel industry. Therefore, steel and power industries must progress together.

PRODUCTION AND DISTRIBUTION OF IRON AND STEEL

Imports constitute only a small part of the total United States supply of pig iron, scrap, and steel. Neither does the iron and steel industry rely on foreign markets to take any appreciable part of its output (tables 4 and 5).

Almost all of the vast American output of pig iron and steel—approximately 42 and 43 percent, respectively, of the world's production—is consumed in the United States.

TABLE 4.—*Production, imports, and exports of pig iron and steel, 1946–53*

[Thousand net tons]

Year	Pig iron			Steel [1]	
	Production	Imports	Exports	Production	Exports [2]
1946	44,842	14	96	66,603	3,012
1947	58,329	33	40	84,894	4,207
1948	60,055	219	7	88,640	3,245
1949	53,413	100	81	77,978	3,518
1950	64,587	796	7	96,836	2,567
1951	70,274	1,067	7	105,200	2,764
1952	61,313	374	14	93,200	3,271
1953	74,901	590	19	111,610	2,680

[1] Imports, nominal.
[2] Including shapes, etc.

SOURCE: American Iron and Steel Institute.

TABLE 5.—*Consumption, imports, and exports of ferrous scrap, 1946–53*

[Thousand net tons]

Year	Consumption			Imports	Exports
	Purchased	Home	Total		
1946	23,350	26,134	49,484	58	142
1947	29,285	31,579	60,864	71	170
1948	32,544	32,420	64,964	481	212
1949	25,171	29,166	54,337	1,151	299
1950	33,376	35,525	68,901	778	217
1951	37,871	38,857	76,728	417	245
1952	34,186	34,837	69,023	154	353
1953	[1] 33,398	[1] 43,733	[1] 77,131	172	317

[1] Total consumption is in actual figures. Breakdown of "home" and "purchased" scrap is based on purchased scrap receipts (net) and home scrap produced.

TABLE 6.—*Production of pig iron, by States, in 1953*

[Thousand net tons]

Pennsylvania	20,767
Ohio	15,148
Indiana	8,350
Illinois	6,582
Maryland, West Virginia	5,696
Massachusetts, New York	4,958
Alabama	4,663
Colorado, Utah, California	3,867
Michigan, Minnesota	3,169
Kentucky, Tennessee, Texas	1,701
Total	74,901

SOURCE: American Iron and Steel Institute.

TABLE 7.—*Production of steel, by States, in 1953*

[Thousand net tons]

Massachusetts, Rhode Island, Connecticut	490
New York	5,924
Pennsylvania	31,054
New Jersey, Delaware, Maryland	6,156
Virginia, West Virginia, Kentucky, Tennessee	4,188
Georgia, Alabama	4,514
Ohio	21,749
Indiana	13,853
Illinois	9,154
Michigan, Minnesota	5,973
Missouri, Oklahoma, Texas, Colorado	3,408
Utah, Washington, Oregon	2,273
California	2,874
Total	111,610

SOURCE: American Iron and Steel Institute

Pig iron and ferrous scrap enter directly into the manufacture of steel ingots, steel castings, and iron castings. In 1953 the following were produced and shipped; figures are in thousand net tons.[3]

Iron and steel castings are generally machined for accurate compliance with specifications, and then used in the manufacture of end products.

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[3] SOURCE: Bureau of the Census.

Production of steel ingots	[1] 111,482
Production of steel castings	[1] 1,840
Shipments of steel castings by foundries (carbon)	1,277
(alloy)	557
Shipments of iron castings:	
Gray iron	13,708
Malleable	971

[1] These figures do not agree with American Iron and Steel Institute because of different coverage—AISI does not canvass steel-casting producers that do not make steel ingots.

TABLE 8.—*World production of pig iron (including ferroalloys), and steel in 1953*

[Thousand metric tons]

(Showing countries producing over 1 million tons)

Country:	*Pig iron*	*Steel*
Australia	1,864	2,082
Austria	1,321	1,284
Belgium	4,217	4,445
Canada	2,869	3,733
China [1]	2,400	2,350
Czechoslovakia [1]	2,781	4,200
France	8,759	9,996
Saar	2,382	2,684
Germany:		
Federal Republic	11,654	15,420
Soviet Zone [1]	1,068	2,177
Hungary [1]	760	1,500
India	1,805	1,531
Italy	1,309	3,452
Japan	4,653	7,662
Luxembourg	2,722	2,659
Poland [1]	2,299	3,600
Sweden	1,004	1,764
Union of South Africa	1,223	1,241
U. S. S. R.[1]	28,000	38,500
United Kingdom	11,354	17,892
United States	70,025	101,250
All other countries	4,531	6,578
Total	169,000	236,000

[1] Estimate.

In 1953 steel was distributed by the following market classifications:

	Thousand net tons	*Percent*
For converting and processing	3,514	4.4
Forgings (other than automotive)	1,445	1.8
Bolts, nuts, rivets and screws	1,342	1.7
Warehouses and distributors	14,879	18.6
Construction, including maintenance	9,918	12.4
Contractors' products	3,324	4.1
Automotive	14,664	18.3
Rail transportation	4,788	6.0
Shipbuilding and marine equipment	872	1.1
Aircraft	161	.2
Oil and gas drilling	756	.9
Mining, quarrying, and lumbering	324	.4
Agricultural	1,233	1.5
Machinery, industrial equipment, and tools	4,328	5.4
Electrical machinery and equipment	2,112	2.6
Appliances, utensils, and cutlery	2,046	2.6
Other domestic and commercial equipment	2,086	2.6
Containers	6,051	7.5
Ordnance and other military	2,691	3.4
Shipments of nonreporting companies	938	1.2
Export	2,680	3.3
Total shipments	80,153	100.0

The United States consumes more steel per capita than any other country in the world. In 1951 American consumption per capita was 1,347 pounds, while Canada—the second largest consumer—used 805 pounds per person.

TECHNOLOGY

This section presents the production of pig iron, the preparation of ferrous scrap, and the utilization of these materials (with other materials) in the production of carbon and alloy steels.

In 1953 the manufacturers of steel ingots and steel for castings consumed 90.4 percent of the total pig iron, 82.4 percent of the total purchased scrap, and 85.1 percent of the total home scrap. The remainder of each of these materials was consumed chiefly in foundries in the manufacture of castings.

PROPERTIES OF IRON AND STEEL

The following table is the composition of a common grade of pig iron (northern pig iron for the basic open-hearth process) and that of a commonly used steel (AISI 1020), in percentages:

	Basic pig, open hearth	*Steel (AISI 1020)*
Carbon	[1] 4.00	0.18–0.23.
Manganese	1.01–1.50	.30– .60.
Phosphorus	.40 max	.040 max.
Sulfur	.05 max	.050 max.
Silicon	1.00 max	[2] .10 max.

[1] Not specified.
[2] May or may not be specified.

The properties of the metal are modified very substantially in the course of manufacture of steel from pig iron, reflecting the changes effected in composition of the metal.

Pure iron is a moderately soft, malleable, grayish-white metal that melts at 1,535° C. (2,795° F.) and boils at 3,000° C. (5,430° F.). It has a specific gravity of 7.87, and its electrical conductivity is about one-sixth that of copper. The variety of uses to which iron and steel are put depends on the great change in the physical properties of iron caused by the presence of small percentages of certain other elements.

Iron is in close contact with coke in the blast furnace and when drawn off to be cast into pigs has 3.50 to 4.50 percent of carbon disseminated through it in the form of graphite flakes and iron carbide. It is the iron carbide that makes pig iron hard and brittle. Pig iron has many other impurities, whose nature and quantity depend on the material charged into the blast furnace. Some pig is used as cast iron to make parts that do not require high tensile strength and are not subject to shock, but over 75 percent of the pig iron produced is used for making steel.

Steel is primarily a refined alloy of iron and carbon and is malleable in the as-cast condition. The carbon content may vary from approximately 0.01 to 1.7 percent; however, nearly all steel contains less than 1 percent of carbon. To the special or alloy steels, one or more elements are added to impart special properties. The most common of these elements are manganese, chromium, nickel, molybdenum, tungsten, vanadium, silicon, and copper. The special properties sought include hardness, stiffness, ductility, resilience, tensile strength, compressive strength, resistance to fatigue, resistance to abrasion, resistance to corrosion, and the ability to hold a cutting edge at high temperature.

MATERIALS USED BY THE IRON AND STEEL INDUSTRY

In 1953, the iron and steel industry consumed the following to produce 111.6 million net tons of ingots and steel for castings (*2*):

Iron ore	million net tons	137
Coke	do	71
Fuel oil	billion gal	2. 4
Natural gas	billion cu. ft	224
Electric power	billion kw.-hr	40. 7
Limestone and other fluxes	million net tons	40
Scrap	do	77
Copper	thousand net tons	43. 9
Lead	do	16. 9
Tin	do	37. 6
Zinc	do	286. 5
Manganese	do	582. 6

PIG-IRON PRODUCTION

A relatively small part of the total of purchased scrap consumed enters into the making of pig iron—approximately 5 percent.

In making pig iron (*6*) a blast furnace is charged with iron ore, coke, scrap (purchased and home), mill scale and mill cinder, sinter, and limestone. It requires approximately 2 tons of iron-bearing materials to produce 1 ton of pig iron, and, in addition, approximately ½ ton of limestone and 1 ton of coke.

A modern blast furnace (fig. 1) is a circular, brick-lined steel shell, sometimes over 100 feet high and with a hearth diameter as large as 28 or 29 feet. Some of the larger furnaces now in operation are capable of a daily production of 1,400 to 1,800 tons of pig iron; however, a capacity of 600 to 700 tons is more usual.

Adjacent to each furnace are 3 or 4 stoves for preheating the air before it is blown into the furnace. These stoves are cylindrical, about 22 feet in diameter and 100 feet in height. The interior of the stoves is filled with refractory brick arranged in a checker pattern; this permits easy passage of the gas and provides a large area of exposed brick surface. In operation the checkerwork is first heated by burning blast-furnace gas in burners the bottom of the stoves. When the checkerwork is hot the burners are turned off, and air is blown through the stove. During passage through the hot brickwork the air is heated to 1,000° to 1,700° F., before it is delivered into the blast furnace.

Iron ore, coke, and limestone (dolomite in some localities) are charged at the top of the furnace by means of a skip car running on an inclined track. The oxygen in the hot air blown through the furnace tuyères burns the coke mostly to carbon dioxide. The carbon dioxide is reduced to carbon monoxide by hot coke, and the metallic oxides are reduced mainly by the carbon monoxide to form metal and carbon dioxide. This carbon dioxide is reduced to carbon monoxide as the gases advance up the shaft of the furnace to the point where the coke is not hot enough for further reduction of the carbon dioxide. Direct reduction of iron ore by carbon also occurs at temperatures above 940° F. The intense heat created by the burning coke melts the iron, which trickles down through the charge and collects in a pool on the furnace hearth. At the same time limestone is calcined by the heat of the furnace; the resulting lime combines with most of the impurities in the ore and coke and forms a molten slag that drips to the hearth, where it floats on the heavier iron.

The slag is drawn from the furnace at frequent intervals, while the iron is tapped 4 or 5 times every 24 hours in "casts" averaging 150 to 350 tons, according to the size of the furnace.

For each ton of iron the furnace also produces about half a ton of slag and 6 tons of gas. The gas passes out at the top and then through cleaners, where most of the dust is removed. Part of the gas is burned in the stoves, as mentioned above, while the remainder is used to produce power for blowing air into the furnaces and to generate electricity. Many steel plants now use this gas to replace part of the coke-oven-gas requirement for the coking process.

Slag may be discharged directly into slag pits adjacent to furnaces or hauled away to slag pits or dumps in large pots mounted on railway trucks. It can be allowed to cool naturally before further processing or quenched with water or steam to produce a granulated or foamed material.

Granulated slag is used mainly in manufacturing cement and for road fill. Crushed and sized air-cooled slag is used as ballast in road building, in concrete, and in similar applications as replacement for crushed stone and in the manufacture of mineral wool and roofing granules. Foamed slag is used almost exclusively in manufacturing lightweight concrete blocks.

Iron tapped from the blast furnace is impure,

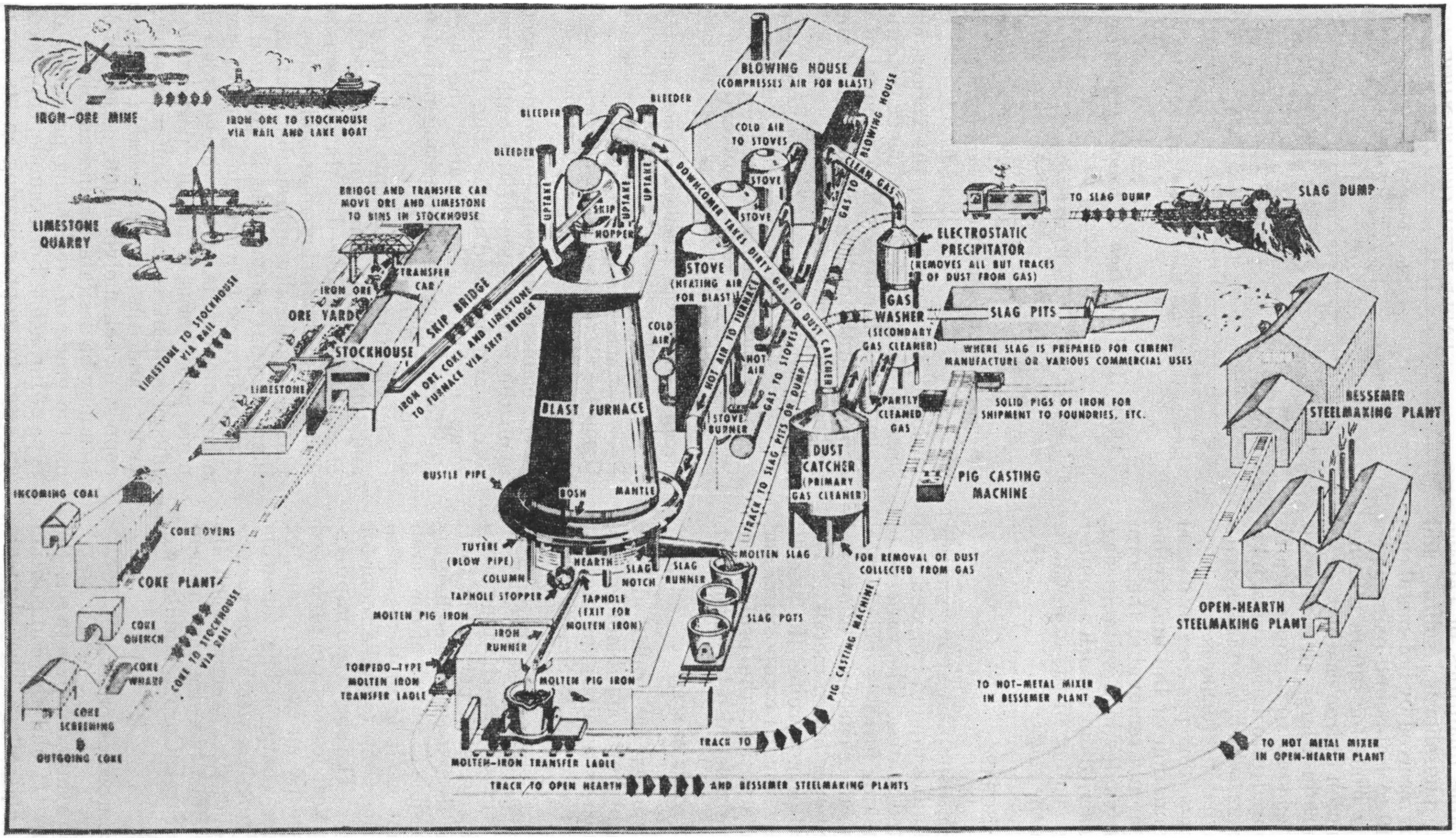

FIGURE 1.—Modern Blast Furnace.

(This drawing is entirely schematic and not to scale.)

This is not an engineering drawing. The various pieces of equipment are not drawn to the same scale. The sketch is intended only to show the major external features of a blast furnace and its auxiliaries, the sources of raw materials, and disposition of the products.

(Courtesy, United States Steel Corp.)

containing considerable quantities of carbon, silicon, manganese, phosphorus, sulfur, and other elements. It is received by hot-metal cars and then cast into 40 to 100-pound bars or pigs (hence the name pig iron) for future refining, for use in the production of iron castings, or, as is the usual procedure in a large steel plant, it is hauled in the molten state to the open-hearth department and poured into large, refractory-lined, hot-metal mixers, which keep it hot until it is needed in steelmaking furnaces.

The process of converting pig iron into steel involves partial elimination of the undesirable elements by oxidation and removal in the form of gas or as molten oxides in a suitable slag. This process can be carried out in such a manner that only a small percentage of the iron will be oxidized and lost in the slag.

SCRAP

Iron and steel scrap supplies approximately 50 percent of all metallics used in iron and steel furnaces in the United States. The varieties of scrap used in steelmaking furnaces include steel scrap and iron scrap, either carbon or alloy, and synthetic scrap (Bessemer blown metals). The essential characteristics governing the usefulness of scrap are its chemical composition and physical form. In addition, the problem of availability and price must also be considered.

PHYSICAL FORMS OF SCRAP

Scrap ranges from grades called Heavy Melting, such as ingot butts and bloom crops, to Light Melting, such as turnings and borings. The Bureau of Mines now has 13 classifications of scrap.

The density of scrap is related directly to steel production. Furnace-scrap-charging boxes that give high scrap weights per cubic foot, using dense material, decrease furnace-charging time and make it possible to put all of the scrap in at one time (that is, it is unnecessary to melt down part of the scrap to make room for the complete charge). The above two improvements will minimize heat losses from prolonged charging times caused by an excessive number of boxes per furnace charge. The following tables give the charging-box weight for various grades of scrap (solid steel weighs 489.6 lb. per cu. ft.), and the effect box size has on scrap density at Armco Steel Corp., Middletown, Ohio.

TABLE 9.—*Weight of scrap*

Scrap	No. of boxes (20.5 cu. ft. box capacity)	Weight, lb.	Wt. per box, lb.	Wt. per cu. ft., lb.
Mixed steel [1]	3, 906	5, 124, 300	1, 312	64. 0
Hydraulic bundles [1]	2, 845	3, 645, 100	1, 281	62. 5
Turnings	562	651, 600	1, 159	56. 5
Coils and thin bars	716	1, 701, 200	2, 376	115. 9
Bloom butts	593	2, 472, 500	4, 338	211. 6
Shear bundles	1, 020	2, 779, 800	2, 724	132. 9

[1] Salted when small scrap is available.

SOURCE: AIME, Open-Hearth Proceedings: 1952, p. 76.

TABLE 10.—*Relation of weight per cubic foot to size of box*

Box size, cu. ft.	*Wt. per box, lb.*	*Wt. per cu. ft., lb.*
20. 5	1, 698	82. 8
25. 0	2, 659	76. 0
45. 0	3, 060	68. 0

SOURCE: AIME, Open-Hearth Proceedings: 1952, p. 77.

CHEMICAL COMPOSITION OF SCRAP

There is a direct relationship between the composition of scrap (*1*) and the composition of the steel bath and slag. The elements present, except sulfur, may be divided into four groups, according to their oxide stability and chemical reactions among the various elements present under basic oxidizing conditions in open-hearth furnaces. The removal of carbon depends on the degree of oxidation of the bath.

The four groups are:

Group 1.— Elements almost completely taken up by the slag: Silicon, aluminum, titanium, zirconium, boron, and vanadium.

Group 2.— Elements distributed between the slag and metal: Manganese, phosphorus, sulfur, and chromium.

Group 3.—Elements remaining almost completely in solution in the steel: Copper, nickel, tin, molybdenum, cobalt, tungsten, and probably arsenic and antimony.

Group 4.— Elements almost completely eliminated from slag and metal: Zinc, cadmium, and lead.

The elements in group 1 have no direct effect upon the composition of the bath, since they oxidize and are held in the slag. High-silicon scrap is undesirable because the high-silica slag formed during the melting period attacks the furnace banks.

The effect of the elements listed in groups 2 and 3 depends on furnace practice and the kind of steel being produced. Manganese is usually desirable, except in low-metalloid grades. Phosphorus is generally undesirable, but the maximum amount permissible is governed by the type of furnace practice, the grade of steel, and other components of the charge (for example, more phosphorus is usually permissible if the silicon content is low). Sulfur, arsenic, antimony, and tin are undesirable. Chromium, copper, nickel, molybdenum, cobalt, and tungsten may be desirable or undesirable, depending on the grade of steel being produced and the type of furnace. For example, copper could be an asset for copper-bearing sheets for roofing but a detriment to tin plate having a 0.06 maximum copper specification. The elements in group 4 have no effect on the slag or bath, since they are eliminated during the melting and refining period, but they may be detrimental to the furnace and auxiliary equipment. Zinc oxide may clog tubes in waste-heat boilers. Excessive zinc in the charge also tends to shorten silica-brick life in roof and end zones. Lead has been known to melt out from the charge at temperatures below its volatilization point and run down through the crevices between bricks. The channels thus formed make room for liquid steel to follow through the bottom when melted.

The distribution of these elements in the electric furnace is the same under basic oxidizing conditions but will vary with a reducing condition. Acid-lined furnaces will not be discussed, because they supply less than 1 percent of United States steel production.

COMMERCIAL ASPECTS OF SCRAP

In addition to the above physical and chemical classification, there is another broad classification of iron and steel scrap into "home" and "purchased." Home or recycled scrap is produced within the steel plants and circulated back to the furnace for remelting. The bulk of home scrap is evolved by cropping blooms, slabs, plate, and strip during the hot-rolling process. Purchased scrap refers to all types obtained outside the steel industry. It is further subdivided into "prompt" and "all others." Prompt industrial scrap is the waste or byproduct of fabricating iron and steel products. The all-other industrial scrap comprises products and parts that are worn out, discarded because of failure, obsolete, or discarded for any other reason.

QUANTITY OF SCRAP USED IN STEELMAKING FURNACES

The scrap portion of steel-furnace charges (open hearth and electric) varies from 100 percent to zero, the balance being pig iron or blown metal (partly refined metal from a Bessemer converter). A Bessemer converter consumes very little scrap unless oxygen is used. Table 11 gives a typical open-hearth charge and table 12 the expected metallic recovery with various types of charges.

KINDS OF STEEL AND PROCESSES EMPLOYED

There is no definite line of demarcation between the generic terms "steel" and "iron", using the term "iron" in the sense of elemental iron or of pure commercial iron products. In alloys containing less than 0.10 percent carbon the terms frequently are used interchangeably and somewhat loosely. (Pig iron and cast iron contain up to 4.5 percent of carbon.) Steels may contain alloying constituents other than carbon. The general classifications of steel are carbon, alloy, stainless and heat resisting, and tool and die *(11)*.

Steel may be made in an open-hearth furnace, a Bessemer converter, or an electric furnace. It is the result of metallurgical reactions that take place in these furnaces. The process is one of refining pig iron and scrap by oxidation

TABLE 11.—*Typical charge for 150-ton open-hearth furnace*

Constituent	Practice 1		Practice 2		Practice 3		Practice 4	
	Pounds	Percent	Pounds	Percent	Pounds	Percent	Pounds	Percent
Scrap	141, 900	45. 2	110, 200	31. 5	63, 000	18. 0	5, 300	1. 5
Hot metal	160, 500	51. 2	210, 000	60. 0	245, 000	70. 0	280, 000	80. 0
Fe in ore	11, 400	3. 6	29, 800	8. 5	42, 000	12. 0	64, 700	18. 5
Total metal	313, 800	100. 0	350, 000	100. 0	350, 000	100. 0	350, 000	100. 0
Charge ore	21, 500	6. 9	46, 000	13. 1	65, 000	18. 5	100, 000	28. 5
Limestone	23, 000	7. 3	19, 000	5. 4	17, 500	5. 0	16, 500	4. 7

TABLE 12.—*Typical ingot yields for various types of charges*

Type of charge:	*Normal range of ingot yield percent of metallic charge*
Scrap and coke (under 20 percent iron)	92–94
Scrap and pig or scrap and hot metal (70 to 55 percent scrap)	90–92
Fifty-fifty (55 to 45 percent scrap)	89–91
High hot metal (under 45 precent scrap)	86–89
Scrap and blown metal (30 precent scrap)[1]	86–89
Hot metal and blown metal (duplex)[1]	84–87

[1] Yield for blown-metal practices includes conversion loss in Bessemer.

SOURCE: AIME, Basic Open-Hearth Steelmaking: 1951, pp. 266, 277.

and slag reactions, whereby the quantities of carbon and silicon are lowered, while the content of other elements is increased or decreased, depending on the kind of steel desired.

Steel can be made using pig iron only; however, steel scrap normally constitutes a considerable portion of the charge, depending on the relative availability of steel scrap and pig iron.

Once steel is made there is always the serious problem of corrosion. Corrosion may be engendered by exposure of metals to the atmosphere or contact of steel with corrosive chemicals, soil, dissimilar metals, etc. Various types of corrosion-resistant steels are produced to combat specific types of corrosion, but these steels are considerably more expensive than carbon steels. Some types of corrosion are best combatted by applying inhibitive coatings, such as tin, zinc, or other metals, or various types of organic and inorganic coatings, including paints, lacquers, etc.

KINDS OF STEEL

Steels that derive their properties mainly from carbon, with no guaranteed or specified minimum alloy content, are classed as carbon steels. Although there are no generally accepted ranges or limits of carbon content for low-, medium-, or high-carbon grades they may be classified as hypoeutectoid or hypereutectoid steels. The hypoeutectoid steels are those in which the carbon content is below the eutectoid value of about 0.80 percent carbon and the hypereutectoid steels are those with carbon content above that value. It is also impossible to make a classification by application, however, they may be classified in accordance with the method of manufacture such as acid open hearth, basic open hearth, electric, or Bessemer. This classification may be extended further to include the method of deoxidation used—for example, mechanically capped, rimmed, semikilled, or fully killed. Properties of these steels are modified by heat treatment to control microstructure and mechanical properties.

Steel containing alloying elements intentionally added to impart desired mechanical or chemical properties not otherwise obtainable with iron and carbon alone is known as alloy steel.

Steel is classified as alloy when the maximum of the range given for the content of alloying elements exceeds 1 or more of the following: Manganese—1.65 percent, silicon—0.60 percent, and copper—0.60 percent; or in which a definite range or definite minimum quantity of any of the following elements is specified or required within the limits of the recognized field of constructional alloy steels: Aluminum, boron, chromium—up to 3.99 percent and cobalt, columbium, molybdenum, nickel, titanium, tungsten, vanadium, zirconium, or any other alloying element added to obtain a desired alloying effect.

STAINLESS AND HEAT-RESISTING STEELS

"Stainless and heat-resisting steels" is a generic term. It does not designate any one steel but rather a whole family of steels whose common characteristic is their comparatively high chromium content. Stainless steel basically is an alloy of chromium and iron containing over 10 percent chromium. When chromium is added to iron in such quantities it imparts throughout the entire body of the metal remarkable resistance to corrosion and heat. Other elements may be added to the iron-chromium combination to obtain steels with special characteristics. The most important of these is nickel, which is used to produce 18–8—that is, 18 percent chromium, 8 percent nickel, balance iron. Other useful additions are columbium, manganese, molybdenum, phosphorus, selenium, silicon, sulfur, titanium, and zirconium, all of which result in modifications for special services or fabrications.

The stainless steels may be grouped according to chemical composition and response to heat treatment as follows:

(1) Ferritic steels—nonhardenable steels containing 11.5 to 27 percent chromium, with generally low carbon contents (ranging from less than 0.08 to a maximum of 0.35 percent).

(2) Martensitic steels—hardenable (by quenching) steels containing 11.5 to 18 percent chromium, with varying amounts of carbon (0.15 percent maximum to 1.20 percent maximum).

(3) Austenitic steels containing 16 to 26 percent chromium, 6 to 22 percent nickel, and 0.03 percent maximum to 0.20 percent maximum carbon—hardenable without quenching.

Two of the most widely used heat-resistant steels not mentioned above, although they are included with the martensitic type, are AISI types 501 and 502, containing 4 to 6 percent chromium. These steels have good strength at elevated temperatures and are corrosion-resistant rather than stainless.

Ferritic stainless steels are generally better than the martensitic type and as good as the austenitic type in resistance to corrosion; however, because of their lower ductility they are not used to replace the austenitic type. These steels are magnetic and can be hot- and cold-worked but usually undergo excessive grain growth during prolonged exposure at elevated temperatures. There is no tendency toward intergranular corrosion (*4*).

Martensitic stainless steels are magnetic; can be cold-worked without difficulty, especially when the carbon content is low; show satisfactory resistance to weather, to water, and to some chemicals; can be machined satisfactorily;

have good toughness; and are easily forged or otherwise hot worked. The corrosion resistance is usually increased by hardening which is accomplished by quenching from the recommended temperatures (*4*).

Austenitic stainless steels are not magnetic unless severely cold worked. They can be hot-worked and cold-worked if proper allowance is made for their rapid work hardening. Except for surface hardening their alloys cannot be hardened by heat treatment. They are extremely shock-resistant and difficult to machine unless they contain sulfur or selenium. These steels can be welded without difficulty; and, if the welding operation is controlled properly, the resistance to corrosion is not impaired. These steels have the best high-temperature strength and resistance to scaling of the stainless steels; however, they are subject to intergranular corrosion at temperatures between 800° and 1,600° F. unless the structure is stabilized to prevent the presence of carbides at or near grain boundaries which are vulnerable to corrosive mediums. This may be done by the addition of titanium, (4 to 6 times the percentage of carbon) or columbium (8 to 10 times the carbon content), which combine with carbon and prevent the formation of chromium carbides. The corrosion resistance of the austenitic stainless steels is usually better than that of the other two groups (*4*).

The most important recent development in the stainless steel industry is the increased use of austenitic stainless steels containing manganese instead of nickel as a major alloying element. These are being used as a substitute for AISI type 301 (18 percent chromium, 8 percent nickel), because of the current shortage of nickel. Manganese, like nickel, has the property of producing an austenitic structure in steel, although it takes twice as much to do it. Steel containing 15 percent chromium minimum, 16.5 percent manganese, 0.10 percent carbon maximum, and 1 percent nickel is now being used as a substitute for type 301 stainless.

TOOL AND DIE STEELS

Tool and die steels form another classification of steel. Essentially they are steels used in making tools for cutting, machining, shearing, stamping, punching, and chipping. These steels may be of the plain carbon or the alloy types, which must be of high quality which is usually attained by special methods of processing.

BESSEMER PROCESS

The Bessemer process was the first of the three processes used commercially in large-scale production of steel. A Bessemer converter is a pear-shaped vessel 10 feet or more high, open at the top, lined with acid refractory material, and mounted on trunnions, so that it can be tilted to receive the hot metal and to pour the steel when a "blow" is finished (fig. 2).

Molten pig iron, 15 to 40 tons at a time, is poured into the converter, and a blast of air is introduced from the bottom through a number of small openings in a specially designed base of refractory material. As the air is forced through the liquid metal, carbon and some of the undesirable elements contained in the pig iron are oxidized or burned out. These impurities escape in the form of gas or are removed with the slag. The extent of blowing is determined by the grade of steel being produced. Ferroalloys may be added to the converter or the Bessemer teeming ladle. The teeming procedure is similar to that employed in open-hearth shops.

The Bessemer process is a rapid method of steelmaking. Each charge requires about 20 minutes. The operation is one of the most spectacular of all steelmaking operations because of the brilliant display of flames and smoke emitted from the converter during the "blow."

Although the preponderance of steel made in the United States is produced in the open hearth, the Bessemer process occupies an important position for producing steel for special purposes, such as free-machining steels, and steels for certain types of wire and some grades of pipe, especially pipe made by the butt-weld process (*6*).

In addition to the steel directly cast from the Bessemer converter, a considerable quantity is transferred molten to tilting open-hearth furnaces for further refining. This is known as the duplex process.

The replacement of part of the air with oxygen in converters (acid and basic) offers great possibilities as a means of increasing, by duplexing, the production rate of stationary open-hearth furnaces as well as electric furnaces. Molten metal refined by surface blowing with high-purity oxygen in a solid-bottom, basic-lined converter (Brassert or Austrian Linz-Donawitz process) could also be used in duplexing. In fact, the low-nitrogen-content steel made by the Brassert process could replace many open-hearth grades.

OPEN-HEARTH PROCESS

Approximately 90 percent of the steel made in the United States is produced in open-hearth furnaces (fig. 3). The annual ingot capacity of the United States steel industry, as reported by the American Iron and Steel Institute, was 124.3

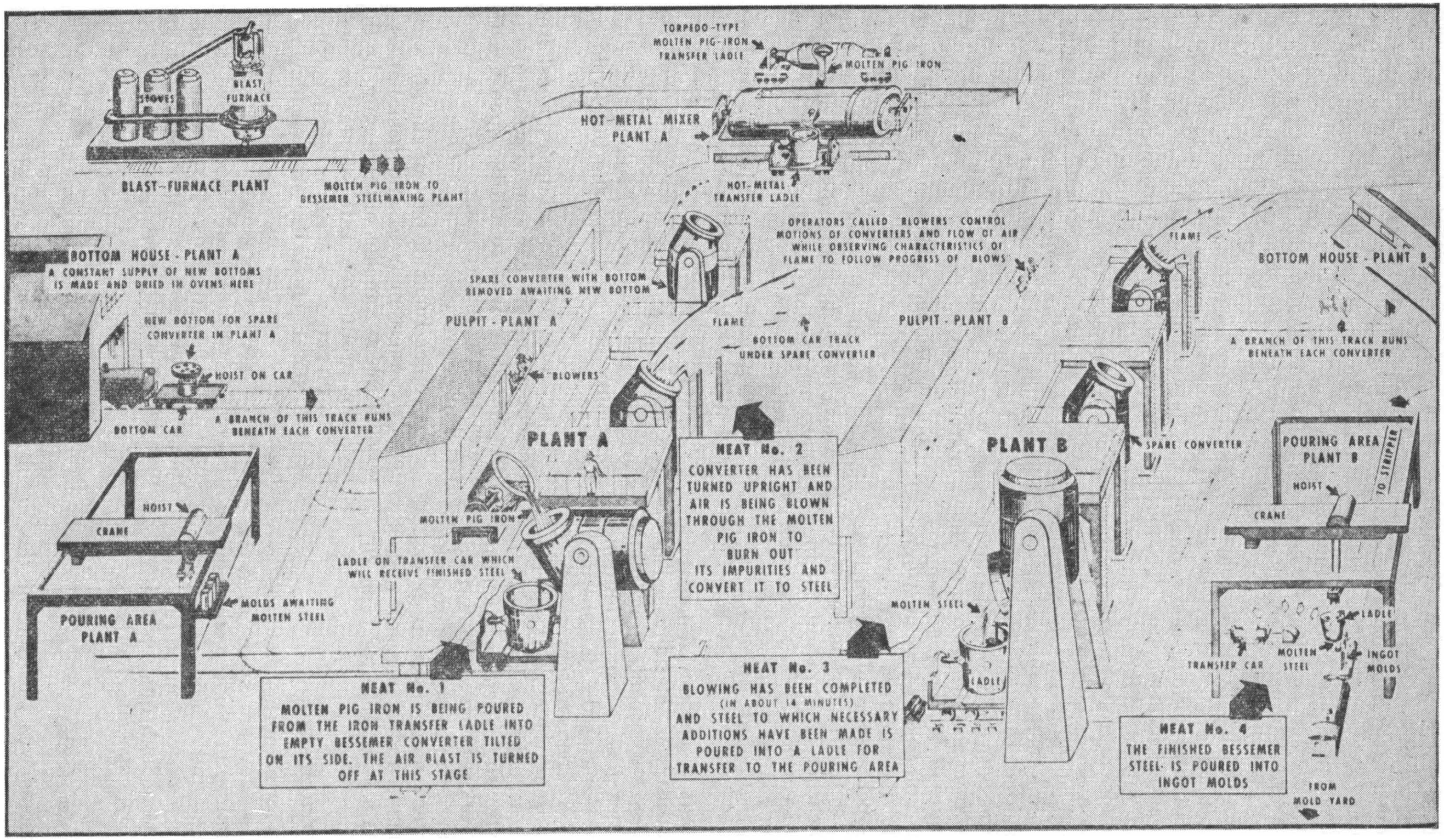

FIGURE 2.—The Bessemer Converter.
(This drawing is entirely schematic and not to scale.)

This is not an engineering drawing. The various pieces of equipment are not drawn to the same scale. The sketch is intended only to show the major external features of two separate Bessemer steelmaking plants and their auxiliaries, the positions of converters at various stages of blows, and disposition of finished steel.

(Courtesy, United States Steel Corp.)

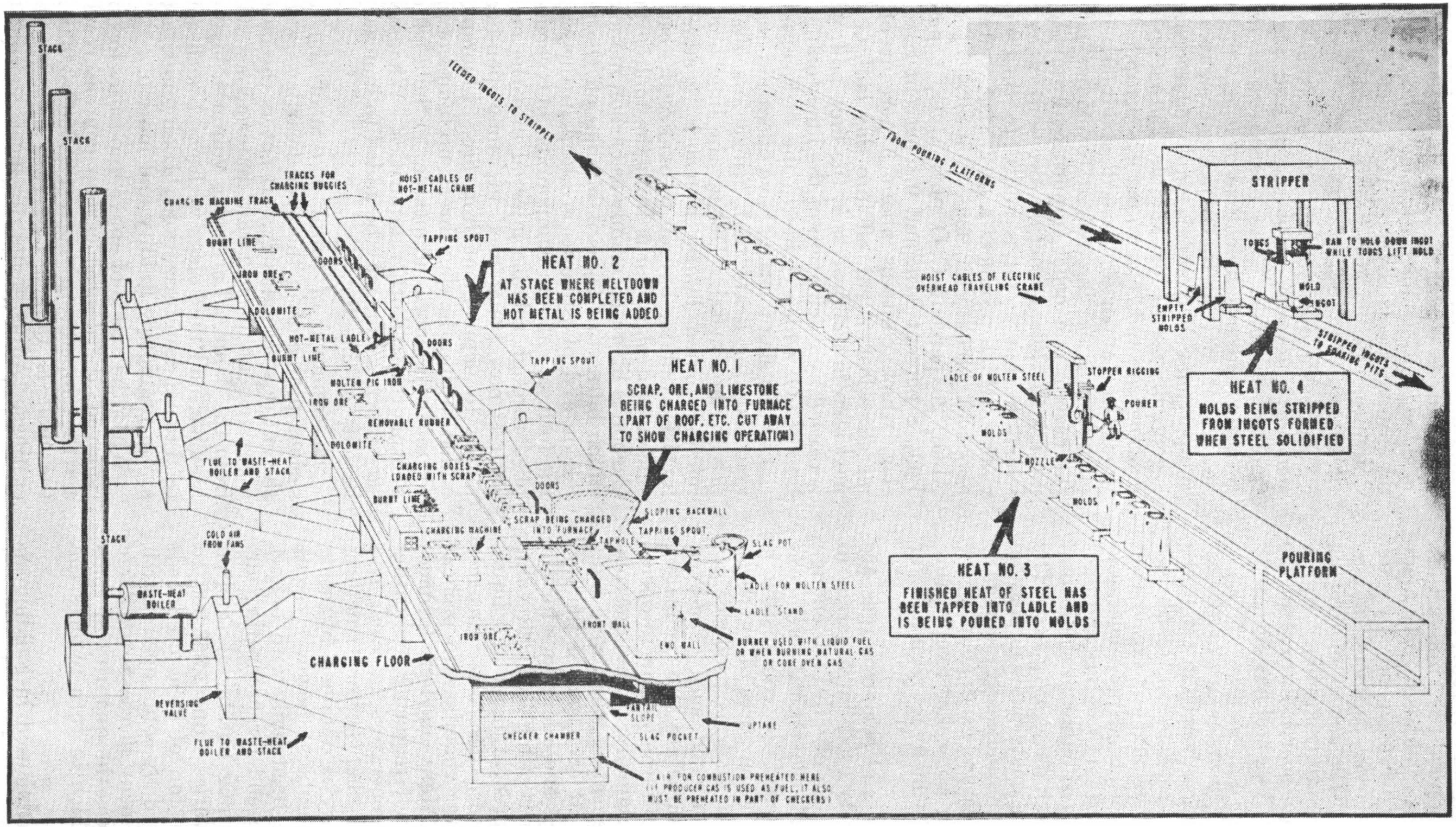

FIGURE 3.—The Open Hearth.

(This drawing is entirely schematic and not to scale.)

This is not an engineering drawing. The various pieces of equipment are not drawn to the same scale. The sketch is intended only to show the major external features of an open hearth and its auxiliaries, the sources of raw materials, and the disposition of its products.

(Courtesy of United States Steel Corp.)

million net tons as of January 1, 1954, of which open-hearth furnaces supplied 109.1 million tons.

A modern open-hearth plant is usually made up of several furnaces arranged side by side. The trend is toward larger furnaces ranging from 200 to 275 tons with a cost of over $4 million per furnace, including auxiliary equipment. The largest open-hearth furnace in the world is in Weirton, W. Va., and has a capacity of 550 tons per heat.

Modern open-hearth furnaces are of the reverberatory type, having a hearth in the shape of a large oval dish. At each end of the furnace are ports (where the fuel and air meet) connected to the refractory checkers-regenerators by flues. Through a system of valves and flues the checkers are connected either to the incoming air supply or the stack for waste gases. The utilization of heat from waste gases is possible by these valves, which automatically change the direction of the flow of air and gases about every 15 minutes to maintain a checker temperature of 1,800°–2,200° F., which is required for efficient operations.

Oil, tar, natural gas, and coke-oven gas are the most common fuels used in America in open-hearth furnaces. To obtain high enough flame temperature and economical fuel consumption, the air supplied for combustion is always preheated, and when gases of low British thermal unit values, such as producer gas or blast-furnace gas enriched with coke-oven gas, are used as fuel, these are also preheated. Since a large number of furnaces now in operation were designed originally to use gaseous fuels that required preheating, they were built with two regenerators at each end of the furnace, a small one for gas and a large one for air. When using gaseous fuels of high calorific value that do not require preheating or the liquid fuels such as oil and tar, these older furnaces employ both regenerator chambers to preheat air for combustion. Many modern furnaces designed to use fuels not requiring preheating still incorporate 2 regenerators at each end because it is believed that better structural strength of the masonry is obtained by using 2 relatively small chambers; however, some modern furnaces have been built with only 1 large chamber at each end for preheating air.

Fuel consumption using hot metal (molten pig iron) and scrap varies from 2,850,000 to 4,800,000 B. t. u. per ton of steel produced. Oxygen, now being used by some open-hearth plants for flame enrichment, increases the flame temperature and reduces the quantity of combustion air needed, which in turn reduces the heat loss in flue gases.

The refractory material used in building an open hearth is about the same for both acid and basic except for the hearth. Both types use silica-brick roofs, which will withstand temperatures up to 3,050° F. A typical basic hearth is made up of insulating fire clay and magnesite or chrome brick covered with magnesite and dolomite. In the acid hearth, silica brick and silica sand are used.

The basic open-hearth charge usually consists of limestone, steel scrap, and pig iron (molten or cold). The charges used in the United States usually consist of 40 to 70 percent hot metal and 30 to 60 percent scrap. If a 100-percent cold charge is employed, more scrap is generally used—for example, 15 to 25 percent cold pig iron and 75 to 85 percent scrap. When the hot-metal portion exceeds about 55 percent, enough iron ore is charged to oxidize part of the carbon and other impurities from the hot metal. A violent reaction (ore boil) takes place when the hot metal is added to this type of charge, which produces a flush slag containing about 30 percent iron, 22 percent SiO_2, 3 percent Al_2O_3, 20 percent CaO plus MgO, 1 percent P_2O_5, 8 percent Mn, and 0.1 percent S. This slag, which contains about 5 percent of the metallics lost, is run off into slag ladles. The next step in the process is the lime boil, which is caused by calcination of the limestone. The lime (CaO) which comes to the surface, makes the second slag for the above operation and the first slag for heats that utilize less than 55 percent hot metal. During the lime boil and refining period lumpy iron ore is added to furnish available oxygen to oxidize the carbon, silicon, and phosphorus from the molten bath. Carbon is liberated mainly as carbon monoxide, which burns to CO_2 as it bubbles off the bath. Silicon and phosphorus are held in the slag in the form of calcium compounds. Oxygen supplied through a lance injected into the molten bath is now being used in some open-hearth plants to supplement iron ore. Fluorspar is added to thin the slag (break up lumps of lime) to make it more adaptable to hold silicon, phosphorus, and other impurities. When necessary, burned lime may be added to increase the basicity of the slag.

The removal of sulfur is one of the most serious problems in the open hearth as well as the entire steel industry. It enters the bath as a constituent in most raw materials as well as from the flame. Large slag volumes, high slag basicity, and fluid slags with good bath action tend to aid desulfurization. High temperatures aid in promoting fluidity and motion.

When the excess of above impurities has been removed, the desired carbon content reached, and the temperature is correct (2,850°–3,000° F.), the melt is tapped and cast into ingots or castings. Adjustments to meet specification are made by adding ferroalloys to the furnace before tapping and to the ladle during tapping.

In the acid process elimination of sulfur and phosphorus is not possible. The basic slag, essential to the removal of these impurities, cannot be used in the acid-lined furnace without injuring the acid (silica) lining. Oxidation of the carbon is accomplished in the same manner as in the basic process. To produce good steel with the acid process, it is necessary therefore to use raw materials with low impurity.

ELECTRIC-FURNACE PROCESSES

Virtually all known types of steel can be made in an electric furnace. The products include the entire range of stainless steels, high-alloy steels, high-speed and other alloy tool steels, low-alloy constructional steels, and plain carbon steels. Two types of electric furnaces are used in steelmaking—the arc type and the induction type.

Electric-Arc Furnace.—All arc furnaces (fig. 4) are of the tilting design and vary in size from a few hundred pounds to 200 tons. The furnace proper is round or elliptical, and the current for heating is led to the bath through carbon electrodes that pass through the roof. Both acid and basic hearths are used. The basic process, which is the most widely used, utilizes a bottom of magnesite or dolomite and sidewalls and roof of either basic or silica brick. The acid electric furnace, with a bottom of ground ganister and silica sidewalls and roof, is employed principally by steel foundries and forging shops. Since the majority of electric-arc furnaces used in the United States are of the basic type, this type only will be discussed. Table 14 gives the growth of electric furnaces with respect to size, power input, charging methods, and production rates.

The high scrap charge (98 percent scrap, 2 percent pig iron—United States average) of the electric furnace is selected according to the grade of steel being produced and charged either through doors, as in the open hearth, or top-charged, if the furnace is equipped with a swing-top roof. Ferroalloys and virgin alloys not easily oxidized are usually charged into the furnace before melting down. The usual desired melt carbon is 0.15 to 0.25 percent higher than the finished steel. Ore, cinder, roll scale, and lance-injected oxygen are used to lower the carbon. Oxygen is also used as an aid in melting and during the refining period to increase temperature.

The electric-arc furnace, with proper slag manipulation, can be operated to permit highly efficient recovery of alloys. Oxidizing and reducing slags may both be used in the same heat, depending on the composition of the

TABLE 13.—*Consumption of pig iron and scrap per ton of steel ingots produced in open-hearth furnaces, in 1953*

State	Number of plants	Ingots produced (thousand net tons)	Total scrap and pig-iron consumption (thousand net tons)	Metallic charge per ton (tons)
Massachusetts	1	490	521	1. 063
Rhode Island	1			
Connecticut	--------			
New York	3	5, 772	6, 246	1. 082
New Jersey	1	5, 687	6, 005	1. 056
Delaware	1			
Maryland	1			
Pennsylvania	31	28, 805	32, 001	1. 111
West Virginia	1	3, 648	4, 160	1. 140
Kentucky	2			
Georgia	1	4, 321	4, 803	1. 112
Alabama	3			
Ohio	17	17, 571	19, 314	1. 099
Indiana	4	13, 818	15, 378	1. 113
Illinois	7	7, 735	8, 688	1. 123
Michigan	2	4, 979	5, 542	1. 113
Minnesota	1			
Missouri	1	3, 088	3, 390	1. 093
Oklahoma	1			
Colorado	1			
Texas	2			
Utah	1	4, 557	4, 874	1. 070
Washington	1			
California	6			
Total	91	100, 471	110, 922	1. 104

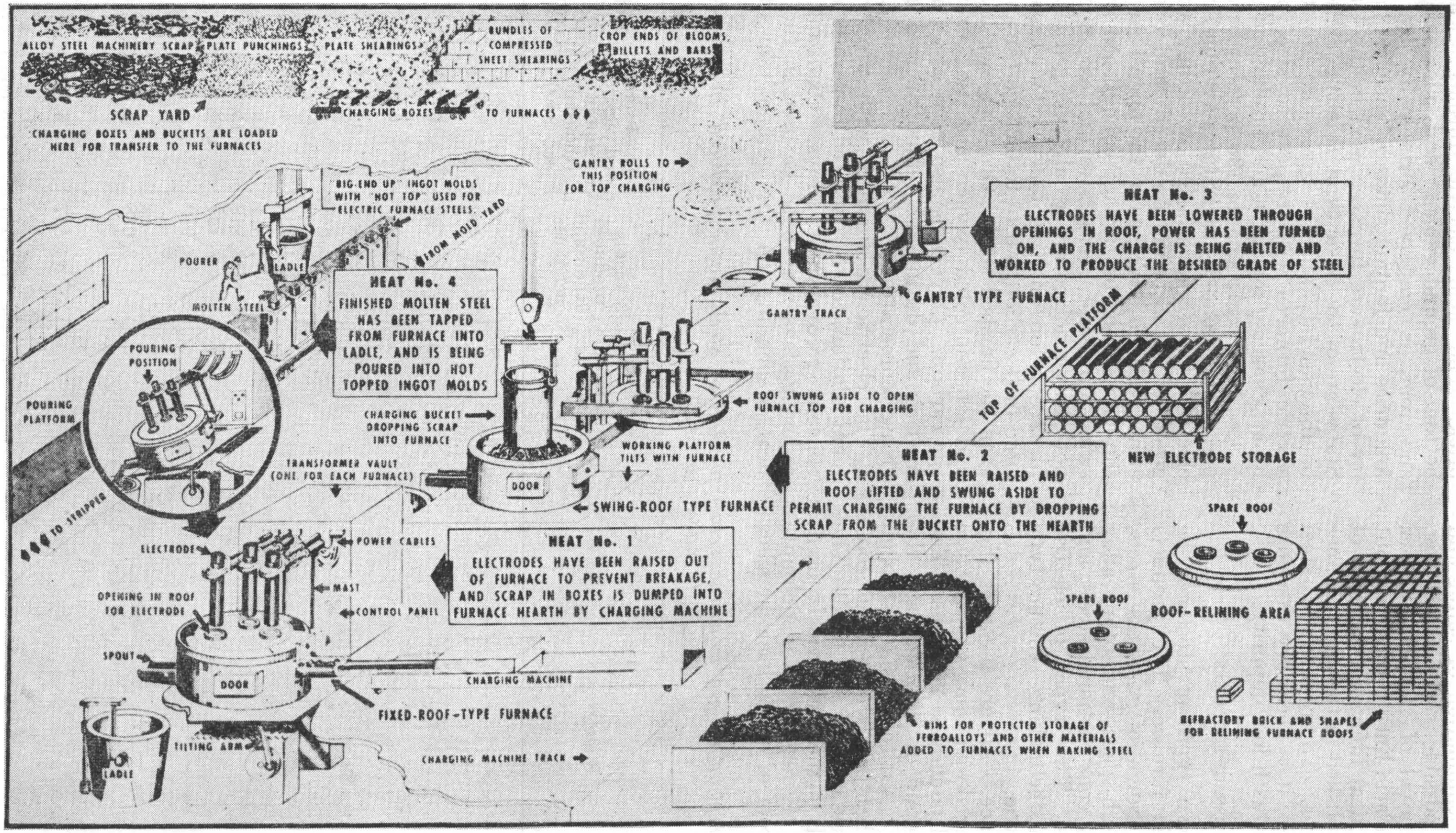

FIGURE 4.—The Electric-Arc Furnace.

(This drawing is entirely schematic and not to scale.)

This is not an engineering drawing. The various pieces of equipment are not drawn to the same scale. The sketch is intended only to show the major features of a plant making steel in electric furnaces, the arrangement of furnaces, sources of raw materials, and disposition of the product.

(Courtesy, United States Steel Corp.)

TABLE 14.—*Comparison of size, power input, and charging method for electric furnaces* [1]

(Reported by C. D. King, vice president, United States Steel Corp.)

	1918	1925	1940	1950	1954 [2]
Diameter of shell	11 ft	16 ft.–6 in	20 ft	20 ft	24 ft.–6 in.
Size of heat net tons	7	40	90	90	200.
Time of heat, tap-to-tap	8:00	10:00	9:30	5:36	8:42.
Rate of production per hour net tons.	0.875	4.0	9.5	16.0	23.0.
Capacity of transformer kv.–a.	2,500	10,000	12,000	20,000	25,000.[3]
Size of electrode	17-in. carbon	18-in. graphite	20-in. graphite	20-in. graphite	24-in. graphite.
Power consumption kw.-hr.	675	570	570	520	475.
Type of charging	Door	Door	Door	Top	Top.
Slag practice	Double	Double	Single	Single	Single.

[1] All data pertain to 100-percent cold charges.
[2] Estimated capacity of plant currently being constructed.
[3] Can be increased to 33,000 kv.-a. with heat exchangers.

SOURCE: Jour. Metals, April 1954, vol. 6, No. 4, p. 465.

charge and the analysis of the product. Limestone, lime, silica sand, crushed coke, aluminum, and fluorspar are materials used for slag constituents. The elimination of elements is the same as in the open hearth under basic oxidizing conditions. Sulfur may be removed from the metal with a reducing slag containing over 2 percent calcium carbide. In the presence of chromium the removal of carbon is difficult because of the chemical affinity chromium has for carbon. Carbon may be absorbed by the metal while chromium is being oxidized and going into the slag. To oxidize carbon in a charge containing a high percentage of stainless steel scrap requires an excess of oxygen and extremely high temperatures (over 3,200° F.). Oxygen supplied through a lance into the molten bath is now being used to a large extent to replace iron ore. Considerable chromium is oxidized along with the carbon, and the resultant slag will contain a high percentage of chromium oxide. Chromium oxide may be reduced and returned to the metal by the addition of lime and crushed ferrosilicon.

The control of hydrogen in stainless steel is a frequent problem of many producers. Hydrogen may come from moisture in the charge materials, leaks in the furnace-cooling system, or high humidity in the atmosphere. Argon or helium gas and dry air are effective for purging hydrogen from the bath.

Within the past few years improvements in electric-furnace design, electrical facilities, wider use of the top-charging techniques, and changes in steelmaking practices have resulted in ever-increasing production rates at costs comparable with those of the open hearth for carbon steels. The carbon-steel portion of total electric-furnace steel production has increased from 15 to 20 percent during World War II to 52 to 60 percent for 1947–53. This increase, plus the fact that electric steelmaking capacity increased from 2.6 million net tons in 1942 to 10.5 million tons by the end of 1953, shows the trend toward carbon-steel production in the electric furnace.

A Battelle Memorial Institute study shows that the capital cost of electric furnaces is only 60 percent of the cost of open hearths, that the cold-charge process (scrap and pig iron) favors the electric furnace in cost, and, with the 50-percent hot-metal-50-percent scrap process, the annual return with the electric furnace equals or exceeds that of the open hearth. Other advantages of the electric furnace are:

(1) Flexibility (they can be put into production or withdrawn at will while the roofs and other brick work of open hearths tend to fall in).

(2) The annual down time for electric furnaces is about 15 days compared with 30 days for open hearths.

(3) The metallic yield is about 2 percent greater than that of the open hearth.

(4) The electric furnace gives greater control of sulfur.

(5) The swing-top roof permits much faster charging.

(6) The use of induction stirring will give the electric furnace an additional advantage in the control of slag-metal reaction, better mixing of alloys, and uniform temperature.

In favor of the open hearth is the fact that most of these furnaces in the United States are either comparatively new or have been modernized. It is therefore unlikely that these furnaces will be taken out of service within the next 20 to 25 years; however, much of the future expansion will probably be in electric furnaces because of their lower investment cost.

Electric Induction Furnace.—No electrodes are required in an induction furnace where, as the name implies, the heating current is induced in the metallic charge itself. Due to the action of the induced current the metal bath is in constant circulation, and the swirling motion assists in removing undesirable nonmetallic particles which would appear as inclusions in the finished steel. The charge is carefully selected to produce the composition desired in the finished steel. Additions are mainly limited to small amounts of ferroalloys as final deoxidizers. Induction furnaces range in capacity from a few pounds up to 3 tons or more. Special steels and alloys, superior in quality to other methods of manufacture, may be produced in induction furnaces by melting in a vacuum or under pressure in an inert gas atmosphere. In such an operation the furnace and mold are enclosed in an airtight container, which can be evacuated or put under pressure. When the melting is complete the entire assembly is tilted to pour the molten metal into the mold. The container is not opened for recharging until the metal has solidified to prevent oxidation and absorption of atmospheric gases. However, units are now under construction that will operate continuously without breaking the vacuum; while one charge is being melted and poured in a vacuum chamber another is introduced through an airlock.

Although arc, resistance, and gas methods have been used for vacuum melting, only the induction furnace has been used for steels on any sizable scale.

ALLOY STEELS

In 1953, 10,328 thousand net tons of alloy steels, including stainless, was produced in the United States. This figure includes 70,880 tons of steel for castings.

Grade:	*Ingots (thousand net tons)*
Carbon-Boron	21
Nickel	31
Molybdenum	502
Manganese	361
Manganese-Molybdenum	291
Chromium	1,722
Chromium-Vanadium	62
Nickel-Chromium	165
Chromium-Molybdenum	1,179
Nickel-Molybdenum	195
Nickel-Chromium-Molybdenum	1,618
Silico-Manganese	105
Other alloys	2,956
	9,208
Stainless Steel	1,049
Total	10,257

RESEARCH BY INDUSTRY, INSTITUTIONS, AND GOVERNMENT

Technologic research by iron and steel companies is small compared with that in other industries; this has been due largely to the large supplies of uniform and high-quality iron ores and coke in the United States. However, the quality of these has been deteriorating rapidly. Manufacturers of plant equipments have been a valuable source of technologic research in iron and steel (*5*).

A substantial increase in expenditures for re-

TABLE 15.—*Principal metals used by the steel industry*

Metal	Reason for use	Typical applications
Aluminum	Removes gases and impurities; aids surface hardness.	Seldom more than a trace remains, except in nitrided steel.
Chromium	Small amounts improve hardening qualities; more than 10 percent prevents rust.	Tools; machinery parts; stainless and heat- and acid-resisting steels.
Cobalt	Holds cutting edge at high temperatures. Improves electrical qualities.	High-speed cutting tools; permanent magnet steel.
Copper	Retards rust	Roofing and siding sheets, plates.
Lead	When mixed with tin, forms a rust-resisting coating for steel. Small amounts alloyed with steel improve machinability.	Sheet steel for roofing, auto gasoline tanks, etc ; machinery parts.
Manganese	Small amounts remove gases from steel; 1 to 2 percent increases strength and toughness; 12 percent imparts great toughness and resistance to abrasion.	Small amounts present in all steels; 1 to 2 percent used in rails; 12 percent or more for frogs and switches and dredge bucket teeth.
Molybdenum	Increases strength, ductility, and resistance to shock.	Tools; machinery parts; tubing for airplane fuselage.
Nickel	Increases toughness, stiffness, strength, and ductility. In large amounts resists heat and acids.	Tools; machinery parts; stainless steels; heat- and acid-resisting steels.
Tin	Forms corrosion-resisting coating on steel	Sanitary cans; kitchenware.
Tungsten	Retains hardness and toughness at high temperature.	High-speed cutting tools; magnets.
Vanadium	Increases strength, ductility, and resiliency	Tools; springs; machinery parts.
Zinc	Forms corrosion-resisting coating on steel	Galvanized roofing and aiding sheets; wire fence; pails, etc.

SOURCE: American Iron and Steel Institute, Steel Facts, February 1947.

search in mining, metallurgy, and geology seems to be overdue, whether by private firms, universities, or Government (*19*).

The expenditures for technologic research in the primary metals industries form only an estimated 3 percent of the total $4 billion spent in 1953 for scientific and industrial research (Research and Development Board, Department of Defense). This is very small expenditure compared with those for comparable research for other industries.

Government research has filled some of the gap. In iron and steel this adds to technology in the direction of conservation in the use of materials, more effective use of facilities, better steels etc.

OUTLOOK

In the United States choice of the several steelmaking processes has been simple until recent years. For high-quality alloy steels—including stainless—the electric furnace has been largely used; and for carbon steels—the open hearth. New processes and modifications of those in use today present an outlook for change. The turbohearth and the Brassert processes, recently introduced into the United States, may be used to supplement present open-hearth and electric-furnace steelmaking facilities. Stationary open hearths probably will make greater use of Bessemer-blown metal. Electric furnaces will continue to be supreme in the field of high-alloy steels and doubtless account for a larger percentage of carbon-steel production. The use of induction stirring may be widely adopted. There will undoubtedly be a substantial increase in the production of special steels and alloys by melting in a vacuum and with controlled atmospheres. The lance-injection method using gases and solid materials, now being developed by the Bureau of Mines, covering decarburization, recarburization, desulfurization, resulfurization, and dephosphorization, may find its way into many steel plants. This innovation will aid in the production of open-hearth and electric-furnace steels by reducing heat times and cost, by raising quality, and by improving steel-plant atmospheric conditions.

POSSIBLE EFFECTS OF ATOMIC POWER ON STEEL INDUSTRY

At present two major effects of atomic power on the steel industry of the United States seem possible (*21*). The first is that, as a result of the availability of low-cost atomic electricity, electric-furnace production of steel from scrap might become important in certain major steel consuming centers which today produce far less steel than they consume or often produce no steel whatsoever. The second is that the use of atomic power in conjunction with the hydrogen process for reducing iron ore might move iron reduction to the iron-ore site. Both effects are important mainly because they imply the use of smaller scale units and a much wider degree of geographic decentralization in steel production than are common in industrialized nations today. In neither instance do large cost reductions appear likely as a result of the change.

It is possible that, with low-cost commercial atomic power the trend in the steel industry toward larger equipment, plants, and ownership organizations might be halted and perhaps reversed. The feasibility of exploiting relatively small scale ore deposits scattered throughout the United States would tend to greater geographical diffusion of steel production than now exists.

DEMAND FOR STEEL

The metals competing with steel have made only relatively slight impact; the demand for steel increases. The outlook is for steel to maintain its dominant position in use.

PROBLEMS

Significance of minor elements derived from ores, coke, scrap, fuel, and fluxes, such as S, P, Mn, Cr, Sr, As, Mo, Cu, Ni, and Co, in making and shaping of steel.—A number of these elements may be desirable or undesirable depending on the use to which the product is to be put. Some cannot be removed, others may be removed but their removal is time consuming, and still others are harmful to the furnace lining. Residual copper, nickel, and tin in carbon steel are increasing yearly, principally as a result of small quantities introduced in scrap.

High material-consumption rate per ton of steel ingots produced.

(1) There is a 5 to 15 percent loss of metallic materials in steelmaking furnaces from oxidation.

(2) Much manganese is lost by not deoxidizing molten steel before the manganese addition in making certain steels. High sulfur in steel, which is detrimental to hot-working properties, is counteracted mainly by additional manganese.

(3) There is a considerable loss to the slag of chromium and other oxidizable additive metals.

Other materials-technology problems.—There is need for developing better techniques for:

(1) Using available scrap more efficiently, particularly the lower grades.

(2) Reducing the quantity of hydrogen that enters steel, mainly through raw materials, causing blisters and brittleness.

(3) Better balancing of raw materials to reduce the volume of slag, to obtain an optimum V-ratio (bases to acids) and viscosity.

(4) Meeting the sulfur content of coke, which is increasing in some localities. Moreover, the sulfur content of fuel oils used in the open hearth has increased.

(5) Reducing open-hearth and converter-dust losses.

Although the American steel industry has made great advancements in the modification and improvement in steelmaking and processing equipment, there is still a need for improvement in some instances.

(1) The low thermal efficiency and short life of open-hearth furnaces are partly attributed to inadequate refractories.

(2) Inadequate handling of materials results in a noncontinuous operation and increased cost.

(3) It takes 1 to 2 hours to charge scrap into a modern open hearth through doors compared with 15 to 20 minutes for a moderately large top-charge electric furnace.

(4) Low scrap density per charging box increases furnace-charging time. There has been an increased use of lighter steel products, which increases the quantity of light scrap.

Air and stream pollution from steel plants are problems in many areas.

(1) Smoke, flue dust and compounds of sulfur and fluorine discharged from furnace stacks are harmful to animal and plant life.

(2) Spent acids, phenol, cyanides, waste oils, and suspended solids are harmful to marine life. Suspended solids may also be detrimental to navigation.

There is natural deterioration of steel caused by atmospheric oxidation and other corrosive mediums.—Much steel is lost annually by corrosion. Remelted corroded scrap yields a low metallic recovery.

Overspecification.—This is a serious problem in the present steel technology. Large quantities of the critical alloying additions could be saved by proper specifications. Overspecification is largely attributable to inadequate knowledge of the effect of the various elements available for alloying with steel. Detailed and precise information on the effect of the various less common alloying additions, if properly disseminated, would help to alleviate this wasteful practice.

Expenditures for technologic research in iron and steel.—Research by American iron and steel companies is relatively small compared with that of some foreign companies or with that of other American industries.

Research in all primary metals in 1953 amounted to approximately $120 million, or only 3 percent of the $4 billion spent for all scientific and industrial research during 1953. This may be contrasted with total 1953 billings of $11.6 billion for products shipped by the iron and steel industry alone; purchases of $6.4 billion of materials, freight, and services; and depreciation, depletion, and amortization of $620.1 million.

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STONE

By

Oliver Bowles [1]

ROCKS are the raw materials of which the world is made. Rock when applied to commercial use is called stone. Stone production is a widespread industry in the United States. Every State produces some kind of stone in some form.

Summary

The industry has two main branches—dimension stone and crushed and broken stone. The term "dimension stone" is applied to blocks or slabs of natural stone that are cut to definite shapes and sizes. It includes cut, carved, sawed, and roughhewn blocks of building stone, roofing slabs, cut and polished memorial stone, paving blocks, curbing and flagging, and many special products, such as tubs, sinks, tanks, steps, baseboard, and floor tile. The term "crushed and broken stone," on the other hand, includes irregular fragments that result from crushing; fine rock powders produced by various grinding processes; and irregular large fragments (riprap) used for shore and harbor protection and for spillways at dams.

The mining or quarrying of dimension stone must be done with great care to avoid shattering or damaging the blocks in any way. It also involves careful selectivity to maintain uniformity of texture and color and freedom from blemishes and other defects. The mining or quarrying of crushed and broken stone, on the other hand, is designed to break the rock into fragments with heavy charges of explosives. Preparation of dimension stone for market involves sawing, splitting, surface dressing, carving, and polishing. Crushed-stone preparation involves reduction by impact or rolls, screening, and possibly washing. Dimension stone is used for constructing buildings or as a facing stone or trim on buildings and for making bridges, retaining walls, and various other structures, such as monuments or statues. Crushed stone is used as concrete aggregate, for railroad ballast and highway construction, and in many chemical, metallurgical, and industrial processes.

It may be observed from the preceding paragraph that these two branches have scarcely a single element in common, in mining, milling, or use. Producers of dimension stone may at times market spalls, fines, granules, and terrazzo chips as byproducts; less commonly, producers of crushed stone may sell blocks of building stone. Except for these unusual overlappings the two branches are virtually unrelated. Accordingly they are considered herein as separate industries.

Slate is a branch of the dimension-stone industry, but the quarrying, manufacture, and uses of this product differ so widely from those of the other branches that it is considered separately herein. Accordingly, the stone industries comprise three branches namely: (1) Dimension stone (except slate); (2) slate; and (3) crushed and broken stone.

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[1] Commodity-industry analyst, Bureau of Mines.

DIMENSION STONE (EXCEPT SLATE)

GENERAL REVIEW

The term "dimension stone," as used throughout the following discussion, excludes slate.

Dimension stone is one of the most ancient and widely used building materials. In the United States it is produced in over 41 States. The principal varieties are limestone, sandstone, granite, and marble. The chief uses are in building and highway construction and for memorials.

Stone is difficult to work. It is not well adapted to mass-production methods. The proportion of waste is large, and because of its density transportation costs are heavy. Accordingly, delivered prices are high, and the principal problems of the industry relate to improved efficiency of machines and methods and to utilization of waste, whereby costs may be reduced.

BACKGROUND

Dimension stone is produced by about 375 companies scattered over 41 States and 1 Territory. Total production is valued at about $50 million annually. Quarrying and dressing blocks of stone are among the most ancient of the mining industries. In the United States the use of stone for construction began with the earliest settlements. Stone was available in so many places and furnished such a durable and attractive construction material that its production attained large proportions many years ago. However, circumstances over which the industry had no control led to a decreasing demand, beginning in the early 1890's. Two important events in industrial evolution had profound deterrent influences on the welfare of the building-stone industry. One was the invention of the steel-skeleton-frame skyscraper-type building (the first was erected in 1885). Before that time the walls of a stone building were entirely of stone and carried the burden of the superstructure. With introduction of the structural steel frame, stone (even in buildings still classed as stone structures) occupied a subordinate place. The modern stone building, so-called, is one having only a thin veneer of stone. Stone is sold by the cubic foot, and the number of cubic feet required for a modern building is only a fraction of the quantity required for a stone structure of equal size 60 years ago.

A second factor destined to become even more far-reaching than the skeleton-frame structure in its adverse effects on the use of stone was the advent of concrete, which began to assume importance about that same date; reinforced concrete, in fact, became an essential part of the skeleton-frame structure, but that was only one of its many structural uses. Concrete, a stonelike material that could be poured in place or could be cast or molded into any desired shape, seemed to have such decided advantages over stone, both in cost and convenience, that 45 or more years ago it began to replace stone extensively. Stone is, however, still preferred by many builders.

In 1916—the first year in which the quantity of building stone sold was recorded—sales totaled over 25 million cubic feet; in 1952 such sales were less than 18 million cubic feet.

Other branches of the dimension-stone industry have likewise suffered from competition with concrete. It has replaced flagstones in sidewalk construction and has been substituted widely for curbstones and paving stones on city streets. This narrowed market has posed many problems for stone producers.

Dimension stone in some form, as mentioned previously, is produced in over 40 States. The sedimentary stones—limestone and sandstone—prevail in the Middle West, particularly in Indiana and Ohio. Granites are produced chiefly in New England, the southern Appalachians, and Minnesota and Wisconsin. Marble production is centered in Vermont, Georgia, Alabama, Missouri, and Tennessee.

Although rocks are the most abundant of all material things, only in certain spots do they have the uniformity of texture and color and the absence of blemishes, irregularities, and seams that render them suitable as dimension stone. Rock structures, such as joints, streaks and irregularities of prospective areas, require careful and systematic study.

The quarrying and milling of dimension stone have been described in considerable detail in several Bureau of Mines publications (*4*, *6*). Later information is given in The Stone Industries (*5*), and in Industrial Minerals and Rocks (*3*), 1949.[2]

A notable advance in granite quarry practice since the publication of these reports is the introduction of detachable carbide drill bits. Their use has increased greatly the speed of drilling, and reduced costs. Fabrication is conducted in large mills equipped with a wide range of precision machines.

Byproducts consist of waste blocks, "roughbacks" (ends of blocks), and spalls or fines that may be sold in limited quantities as riprap or crushed stone. Occasionally waste limestone or marble is used for lime manufacture.

The most widely used substitutes for building stone are concrete, terra cotta, brick, and lumber. As stone generally is more costly than any of these substitutes, it is used chiefly for high-class buildings where its architectural adaptability, permanence, and dignity may be capitalized. Granite and marble are used extensively as memorial stones; and in this field of use they have very little competition, although bronze and synthetic substitute—cast stone—are also competitors. Stone is reused at times, but the quantities involved are relatively small.

Although deposits suitable for dimension stone are numerous, those having the requisite qualities are commonly confined to restricted areas. Nevertheless, in the major producing districts the reserves are evidently adequate for many years' operation. No definite reserve figures are available.

USES

The major use of dimension stone is for building purposes. This includes exterior and interior wall construction, piers, columns, caps, steps, sills, trim, wainscoting, railing, flooring, sanitary enclosures, mantels, baseboard, retaining walls, arches and other ornamental structures. A large market has been developed in recent years for "split stone," "strip stone," ashlar and rubble as veneer in residences and other smaller-type buildings. It is also used for memorial stones and for paving blocks, curbing, and flagging. Marble and other ornamental stones are used for making table tops, lamp bases, book ends, ashtrays, and various novelties.

Statistics of the industry are compiled annually by the Bureau of Mines (*9*). Figures for imports and exports are compiled by the Bureau of Mines from data collected by the United States Department of Commerce.

The production and consumption of dimension stone are virtually synonymous, as stocks are moderate and more or less constant. Tables 1 and 2 show the production of dimension stone, by kinds and uses, over a recent 5-year period.

TABLE 1.—*Production of dimension stone, by kinds, 1948–52*

[Short tons]

	1948	1949	1950	1951	1952
Granite	639, 180	485, 860	554, 990	531, 857	518, 838
Basalt	58, 230	35, 370	37, 340	32, 951	57, 996
Marble	82, 700	101, 720	89, 290	87, 191	89, 051
Limestone	607, 130	679, 800	807, 590	806, 842	786, 757
Sandstone	204, 250	225, 590	265, 500	291, 253	352, 230
Miscellaneous stone	80, 120	90, 090	104, 490	111, 595	91, 458
Total	1, 671, 610	1, 618, 430	1, 859, 200	1, 861, 689	1, 896, 330
Total value.	$48, 024, 119	$51, 746, 125	$58, 464, 183	$60, 169, 852	$57, 020, 764

TABLE 2.—*Production of dimension stone, by uses, 1948–52*

[Short tons]

	1948	1949	1950	1951	1952
Building stone	1, 257, 400	1, 255, 050	1, 475, 900	1, 489, 850	1, 519, 459
Monumental stone	306, 770	257, 510	243, 960	234, 753	230, 490
Paving blocks	3, 210	1, 960	3, 740	3, 270	2, 256
Curbing	62, 950	59, 420	78, 040	82, 481	84, 177
Flagging	41, 280	44, 490	57, 560	51, 335	59, 948
Total	1, 671, 610	1, 618, 430	1, 859, 200	1, 861, 689	1, 896, 330

Substantial quantities of stone enter foreign trade. Imports are valued at 1¼ to 1½ million dollars a year. Most of the imports consist of marbles from Italy, Spain, France, Belgium, Portugal, and England. Granite (chiefly for memorials) is imported from Finland, Sweden, and Canada. Travertine is imported from Italy and onyx marble from Mexico. Because of low wage scales in some foreign countries increasing sales of imported cut and polished stone are forecast by some producers. Exports, which are much smaller than imports, are shipped chiefly to Canada.

The United States may be regarded as self-sufficient insofar as quantities of building and ornamental stones are concerned but not from the standpoint of satisfying public demand for variety. There is a popular demand for special types of yellow, red, veined, and brecciated marbles that are not produced in America and for certain granites that have no western world counterparts. No satisfactory substitute for the popular Italian travertine has ever been produced in quantity on this side of the Atlantic.

PRICES

Prices of stone are relatively high compared with those on other building materials because stone is difficult to work and requires many hand operations. As indicated in table 3, prices have not in general advanced out of proportion with those of other materials.

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this section.

TABLE 3.—*Price history of dimension stone, 1930–50 (5-year periods), 1951–52*

[Value per cubic foot]

	Building stone				Memorial stone	
	Granite [1]	Marble	Sandstone [1]	Limestone [2]	Granite [3]	Marble
1930	$4.83	$3.67	$2.44	$1.90	$3.75	$3.71
1935	3.33	4.63	1.57	1.26	2.70	5.07
1940	4.31	4.25	1.83	1.07	2.80	5.45
1945	2.85	5.93	1.32	1.44	3.16	7.15
1950	8.48	8.65	2.32	2.55	5.60	9.72
1951	7.94	8.50	2.53	2.82	5.60	9.25
1952	7.52	8.67	2.52	2.59	5.76	9.34

[1] Dressed.
[2] Finished.
[3] Rough and dressed.

TARIFF

The United States Tariff Act of 1930 established an import duty of 60 percent ad valorem on manufactured granite (including paving blocks) and 25 cents per cubic foot on unmanufactured. The duty on marble, breccia, and onyx in rough blocks was 65 cents per cubic foot and on sawed or dressed, over 2 inches thick, $1 per cubic foot. Slabs and paving tile were dutiable at 8 to 13 cents per superficial foot, with an additional charge of 3 cents if rubbed and 6 cents if polished. Mosaic cubes were dutiable at ¼ cent per pound plus 20 percent ad valorem if loose and at 5 cents per pound plus 35 percent ad valorem if attached to paper or other material. Manufactured articles wholly or in chief value of marble were dutiable at 50 percent ad valorem. On unmanufactured sandstone and limestone the duty was 15 cents per cubic foot and on hewn, dressed, or otherwise manufactured stone 50 percent ad valorem. Unmanufactured travertine was dutiable at 25 cents per cubic foot.

The rates established by the Tariff Act of 1930 have been modified considerably by trade agreements. In the trade agreement with Sweden, effective August 5, 1935, the duty on granite paving blocks was reduced from 60 to 40 percent ad valorem. The duty on all hewn or dressed granite (including paving blocks) was reduced to 30 percent ad valorem in the Finnish agreement, effective November 2, 1936. The duty was reduced further to its present rate of 15 percent ad valorem pursuant to concessions made under the General Agreement on Tariffs and Trade (GATT), effective June 6, 1951. The duty on unmanufactured granite, which had been reduced from 25 to 20 cents per cubic foot under the Swedish agreement, was further reduced to 12½ cents under the Finnish agreement. It was reduced to its present rate of 10 cents per cubic foot pursuant to concessions made under GATT, effective April 30, 1950.

The duty on onyx, in blocks rough or squared, was reduced from 65 to 32½ cents per cubic foot in the trade agreement with Argentina, effective November 15, 1941, and with Mexico, effective January 30, 1943. The duties on all other forms of marble, breccia, and onyx were reduced pursuant to concessions made under GATT. In blocks rough or squared only: Black marble was reduced from 65 to 32½ cents per cubic foot, effective January 1, 1948; other marble and breccia reduced to 32½ cents per cubic foot, effective May 30, 1950. Sawed or dressed, over 2 inches thick: Black marble reduced to 50 cents per cubic foot, effective January 1, 1948; other marble and breccia reduced to 50 cents per cubic foot, effective May 30, 1950; and onyx reduced to 50 cents per cubic foot, effective November 17, 1951. The duties on slabs and paving tiles were reduced from 8–13 cents to 4–6½ cents per superficial foot; the additional charge of 3 cents if rubbed was reduced to 1½ and the additional charge of 6 cents if polished to 3. These reductions in duties were effective May 30, 1950. The duty on loose mosaic cubes was reduced from ¼ cent per pound plus 20 percent ad valorem to ⅛ cent per pound plus 10 percent ad valorem; that on attached cubes was reduced from 5 cents per pound plus 35 percent ad valorem to 2½ cents per pound plus 17½ percent ad valorem, effective May 30, 1950. The duty on manufactured marble was reduced from 50 to 25 percent ad valorem, effective January 1, 1948.

The duties on travertine, sandstone, and limestone were also reduced pursuant to concessions made under GATT. The duty on unmanufactured travertine was reduced from 25 cents to 12½ cents per cubic foot, effective May 30, 1950. On unmanufactured sandstone and limestone the duty was first reduced from 12½ to 10 cents per cubic foot, effective January 1, 1948, and further reduced to 7½ cents per cubic foot, effective May 30, 1950. On hewn, dressed, or otherwise manufactured stone the duty was reduced from 50 to 25 percent ad valorem, effective May 30, 1950.

TRANSPORTATION

As stone is a heavy product, weighing 160 to 170 pounds per cubic foot, transportation expense adds greatly to the delivered cost, and the freight rate becomes an important factor in competitive marketing. Nevertheless, stone is commonly shipped long distances to satisfy architectural demands for certain textures and colors.

RESEARCH

Research by industry has been confined primarily to improvements in machine design and methods for fabricating stone. The more important of these developments include the use of diamond-toothed saws and the introduction of hard alloys for saw teeth and planer tools. Sandblast carving was an important advance. The larger companies also conduct continuing research on the durability of their products, the adaptability of stone for interior or exterior work, color stability, etc.

One company in Indiana conducted tests on quarrying limestone with wire saws; and, although the results of the tests indicated a substantial saving in cost (*16*), the industry has not adopted the equipment generally in its quarries. Two companies have conducted extensive research on the use of wire saws in quarrying and milling granite. These companies now regard wire saws as having passed the research stage and attained the status of a productive tool. One marble company carried on similar research in 1927 and 1928 and again in 1932 and 1933, but the results were less promising (*1*, *2*).

Relatively little research on dimension stone has been conducted by institutions. Government research has been centered mainly in the Bureau of Mines and the National Bureau of Standards. The Bureau of Mines conducted extensive studies on efficiency of methods and equipment many years ago (*4*, *6*, *7*), but such work has been virtually dormant during recent years. The National Bureau of Standards has conducted research on the physical and chemical properties of stone, its durability, and the development of equipment and methods for measuring durability (*11–14*). The Bureau of Mines has cooperated in some of this work (*11*). The Federal Geological Survey has made a detailed study of the natural features of Indiana limestone in their relation to commercial grading (*15*).

OUTLOOK

The prospects for future expansion of the dimension stone industries are not widely believed to be bright. However, some producers feel that a larger market for memorial stones is in prospect and that the merits of natural building stone are attaining wider recognition by architects, builders, and the public. Stone is encountering increasing competition from the newer building materials, such as glass blocks and architectural concrete. Aluminum and other metals are being used to some extent in fields formerly occupied by stone. Stone will continue to be favored for large, dignified structures, but there is little promise of expansion in new fields of use. New sources of supply will undoubtedly be developed if the materials find favor with architects and builders, but such developments are likely to be at the expense of the old-established centers of supply.

PROBLEMS

The principal handicaps of stone in the competitive field are its difficult workability, its lack of adaptation to mass-production methods, and its heavy weight per cubic foot. These factors contribute to the high delivery cost of the finished product. Accordingly the principal problems of the industry relate to methods of reducing the cost of quarrying and manufacture.

Quarry methods vary according to the nature of the rock. Primary cuts in limestone, sandstone, and marble quarries are usually made with channeling machines, which are cumbersome and costly. The development of more efficient methods and machines is a major problem. Primary cuts are made in granite quarries by discharging light charges of black blasting powder in drill holes or by a process of drilling and broaching, which consists of drilling a line of holes close together and broaching out the webs between them. The former process results in a high proportion of waste, and the latter is slow and costly; hence the development of better methods is important. Industry has made some progress in this direction, particularly in experimental use of wire saws (*8*, *10*). Many limestone and sandstone fabricating mills are served by wire saws for block trimming and subdivision, but few companies use them for making primary cuts in quarries. This is a promising field of research.

Industry has made notable progress in the use of diamond-toothed saws, hard-alloy tools, and sand-blast carving in stone-fabricating mills. Wire saws are used in some sandstone, limestone, granite, and marble mills, and their wider employment in both old and new applications is a problem for continuing research.

The principal polishing agent used on marble and granite is putty powder, the chief constituent of which is tin oxide. Because of the high price of tin and its scarcity at times, the development of a substitute polishing agent merits attention.

Waste utilization is an ever-present problem at dimension-stone quarries and mills. The problem varies with the location. For quarries and mills situated near large centers of population where aggregates and other forms of crushed and broken stone are in demand the problem is simpler than in remote regions where byproduct outlets are small. In addition to the exploitation of ordinary byproduct uses of waste, such as riprap, crushed stone, stone sand, agricultural limestone, metallurgical limestone, stone for chemical and industrial uses, railroad ballast, poultry grit, sand from sandstone quarries, etc., a promising field for research is the separation and utilization of the individual components of such rocks as granite.

Statistics as now collected need improvement through development of more complete data, either on a rough-block or finished-product basis, or possibly determination of conversion factors. Producers, consumers, and Government agencies concerned with dimension stone, slate, and crushed and broken stone all need reliable technical, statistical, and economic information.

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SLATE

GENERAL REVIEW

Slate is a rock possessing a cleavage that permits it to be split into thin slabs having many important uses. Roofing slates, steps, window sills, baseboard, blackboards, electrical panels, and flagging are typical products. Slate has many competitors in the construction field; and, although it has superior qualities for many applications, such as churches, museums, art galleries, schools, and college buildings, production costs are so high that it maintains its position with difficulty.

The major problem of the slate industry is to discover ways in which costs may be reduced, because thereby it could offer roofing and other high-quality products on a competitive basis with alternate materials, a condition that would promote a healthier state in the industry and would benefit the public at large. Costs can be reduced by eliminating and utilizing waste and by improved technology; also in other ways, such as are common to industry in general.

BACKGROUND

The term "slate" is applied to a fine-grained rock that has a more or less perfect cleavage permitting it to be readily split into thin, smooth sheets. The term includes materials that differ widely in color and have a considerable range in chemical and mineralogical composition. Slates have resulted from intense metamorphism acting upon sedimentary deposits consisting largely of clay.

In both production and consumption, slate is predominantly a domestic industry; neither imports nor exports are large. The only significant imports are those of school slates, bulletin boards, and blackboards. There are normally 80 to 95 slate-quarry operations in the United States. The industry has two branches, which are not only virtually independent of each other but to some extent competitive. One branch (which is generally regarded as the slate industry in its proper sense) comprises the preparation of roofing slates and millstock of various kinds. The other branch comprises chiefly the preparation of slate granules used on composition roofing, which in a measure competes with roofing slate. The total sales of each branch average 6 to 8 million dollars a year.

The granule branch is not an industry in itself; it is a portion of a much larger industry, including the manufacture of granules from other raw materials, such as quartz and traprock. Slate granules comprised only 28 percent of the total granule production in 1952. Granule production is in reality part of the crushed-stone industry. Although slate-granule manufacture could be regarded as a phase of dimension-slate production because waste slate can be used for making granules, as a matter of fact only minor quantities of waste slate are so used. The larger units of the slate-granule industry have their own independent quarries and utilize materials unsuitable for making slate roofing or millstock. For these reasons, the first branch of the industry mentioned heretofore is considered herein as the slate industry, although incidental reference will be made to slate granules as a possible field of waste utilization.

The slate industry is centered in four principal locations: (1) Lehigh and Northampton Counties, Pa., in the neighborhood of Bangor, Pen Argyl, Wind Gap, and Slatington; (2) Monson, Maine; (3) the New York-Vermont area near Granville, N. Y., and Poultney, Vt.; and (4) Buckingham County, Va. The Peach Bottom district near Delta, Pa., was an important producer in past years. Arkansas, California, and Georgia have produced flagging and other products at times.

Slate is produced mainly from open-pit quarries, although a few underground mines are or have been worked. The technique of quarrying slate from the parent ledges and manufacturing it into finished products is covered in detail in Bureau of Mines Bulletin 218, The Technology of Slate (pp. 27–81) (*2*).[3] The significant features are the generally costly methods of separating blocks at the quarry face, the employment of hand methods in manufacturing roofing slate (methods that have changed very little in the past 150 years), the use of obsolete milling equipment, and the high proportion of waste. These are the factors most intimately related to the problems of the industry, and will be discussed in some detail in later sections.

The principal byproducts of the industry are slate granules and flour, but only small quantities of such products are made from slate waste. Other actual or potential byproducts are lightweight aggregate, slate-lime brick, and slate flour as a filler in paints and other products.

Slate has to meet competition from many sources. There is no application for which some other material may not be used. As a roofing material, it competes with wood shingles, sheet metal, asbestos-cement shingles, ceramic tile, and various types of composition roofing. In the form of blackboards it competes with plaster, various synthetics, porcelain, steel, glass, and special painted surfaces. For electrical panels, such materials as marble, soapstone or plastics are alternative products. For baseboard, steps, wainscoting, floor tile, and other structural products it competes with marble, granite, travertine, sandstone, soapstone, terrazzo, and cement products.

Slate is so enduring that roofing slates are reused at times on successive structures, but such reuse is proportionally small.

No systematic exploration of slate reserves has been made in the United States. From the geology of the Pennsylvania formations, as described by Behre (*1*), the deposits of commercial slate are evidently extensive enough to maintain the industry at current rate of production for many years. The reserves in New York, Vermont, Maine, and Virginia are also extensive, although unmeasured.

The principal uses of slate are for roofing, blackboards, school slates, electrical panels, and structural products such as mantels, hearths, floor tile, steps, risers, flagging, skirting board, wainscoting, window sills, and lavatory slabs. A new application in the construction industry is the use of slate blocks for wall veneer on residential buildings. The colored slates of the New York-Vermont area are particularly adapted to such use. Slate is also used for making billiard and other table tops, vats, sinks, vaults, laundry tubs, laboratory equipment, and mapping-table tops. Waste slate is used at times for making roofing granules and as flour for filler in paints, linoleum, hard rubber, and other products.

Statistics of the domestic slate industry are compiled annually by the Bureau of Mines (*4*) and are adequate for current purposes. Data on imports and exports are assembled by the Bureau of Mines from figures compiled by the United States Department of Commerce. Foreign statistics are available for some countries, notably Great Britain, but have no special significance in the United States in view of the limited scope of international trade.

The production and consumption of slate are virtually synonymous, as stocks are moderate and more or less constant. Table 4 shows the production by major uses for a 10-year period, and table 5 production of millstock, by kinds, for 1951 and 1952. The domestic industry has adequate facilities to supply all demands for slate, although temporary shortages may develop at times for certain classes or types owing to unusual and unforeseen demands, as, for instance, an abnormal call for roofing slate after a hurricane. Slate is predominantly a peacetime industry. Wars and rumors of wars tend to depress rather than stimulate production.

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[3] Italicized figures in parentheses refer to items in the bibliography at the end of this section.

TABLE 4.—*Slate sold by producers in the United States, by uses, 1943–52*

Year	Roofing		Millstock		Other [2] (value)	Total value
	Squares [1]	Value	Square feet	Value		
1943	96, 220	$841, 750	2, 644, 140	$938, 368	$166, 231	$1, 946, 349
1944	89, 090	802, 179	2, 041, 210	715, 689	203, 090	1, 720, 958
1945	101, 300	976, 122	2, 107, 780	742, 345	253, 273	1, 971, 740
1946	146, 790	1, 982, 928	2, 371, 820	1, 032, 584	403, 990	3, 419, 502
1947	170, 590	3, 094, 780	2, 549, 080	1, 444, 835	537, 705	5, 077, 320
1948	218, 650	4, 566, 056	2, 541, 250	1, 600, 019	700, 477	6, 866, 552
1949	181, 490	3, 759, 564	2, 741, 040	1, 727, 649	912, 503	6, 399, 716
1950	197, 570	4, 098, 842	3, 180, 600	2, 130, 430	1, 342, 053	7, 571, 325
1951	205, 120	4, 357, 412	3, 168, 540	2, 127, 387	1, 522, 911	8, 007, 710
1952	145, 640	3, 067, 513	2, 725, 660	2, 049, 895	1, 469, 396	6, 586, 804

[1] Slate required to cover 100 square feet of roof.
[2] Slate used for walkways, stepping stones, and miscellaneous uses.

TABLE 5.—*Millstock—production by uses, 1951–52*

Uses	1951		1952	
	Square feet	Value	Square feet	Value
Electrical slate	326, 090	$470, 179	311, 710	$519, 619
Structural and sanitary slate	1, 250, 810	836, 492	1, 360, 880	896, 093
Grave vaults and covers	12, 880	10, 681	8, 960	7, 103
Blackboards and bulletin boards	1, 133, 770	667, 011	[1] 922, 860	[1] 553, 509
Billiard-table tops	207, 490	131, 081	121, 250	73, 571
School slates	237, 500	11, 943	(1)	(1)
Total millstock	3, 168, 540	2, 127, 387	2, 725, 660	2, 049, 895

[1] A small quantity of school slates is included with blackboards and bulletin boards.

PRICES AND COSTS

The initial purchase price of slate is higher than that of some other roofing materials; but, because of low upkeep, its cost over the life of a building is quite favorable. It competes largely on the basis of its lasting qualities and architectural adaptability. The high prices of slate products are occasioned by high production and delivery costs. Slate occurs generally in relatively narrow, steeply dipping beds that are costly to quarry; moreover, the proportion of waste is high, ranging from 70 to 90 percent of gross production. Furthermore, manufacturing processes demand much hand labor. Prices of slate have advanced greatly through the years. A square of slate—the quantity required to cover 100 square feet of roof—sold for $3.00 in 1887. In 1952 it sold for $21.06. The average value per square foot of millstock in 1907 (the first year for which figures are available) was 10 cents; in 1952 it was 75 cents.

TRANSPORTATION

As slate production is confined to 4 principal centers, distribution is an important element in marketing, and, as slate is a heavy product, the expense of distribution adds greatly to the delivered cost. Fortunately, the production centers are within reasonable distance of populous areas where the chief markets exist.

RESEARCH

The slate industry is relatively barren in research. Modern equipment has been introduced to some extent in the mills. Circular saws equipped with detachable hard-alloy teeth are used widely. Recently diamond saws have been introduced. Forklift trucks have facilitated the handling of roofing slate and flagging. Thus the industry has made some progress in adopting more efficient tools developed elsewhere but has itself conducted little or no fundamental research. Institutional research in recent years has been confined mainly to studies at State College, Pa., on the possible use of slate waste for making lightweight aggregate and other byproducts. The results of the tests have been published (*8*). This was followed by a time study of unit processes as a guide toward increased efficiency of operation (*7*).

Research was conducted by the Bureau of Mines in conjunction with the industry and the National Slate Association beginning in 1926. This involved introduction of a new type of equipment—the wire saw—that greatly improved slate quarrying in the Pennsylvania district. The research program and its results are described in detail in Bureau of Mines

Technical Paper 469, the Wire Saw in Slate Quarrying (*3*).

A later research project of the Bureau of Mines was devoted to possible utilization of slate waste for making lightweight aggregate (*6*).

OUTLOOK

Current trends in the industry are not encouraging. Sales are maintained at a fairly constant level but have not advanced in consonance with increases in the volume of building construction, even though slate is used chiefly as a building material. Such a condition indicates the inability of slate to meet the prices of some other types of roofing. Although reserves are ample, development of additional quarries is not anticipated at present. The research that has been done on new uses has indicated some possible new markets; but these have not yet been fully exploited and do not seem to promise a large-volume market.

PROBLEMS

Waste constitutes a major problem of the slate industry. This problem was presented in detail in Bureau of Mines Bulletin 218, The Technology of Slate (pp. 81–107); and, although this report was published in 1922 (*2*), the presentation is, with few exceptions, as applicable today as then. Although waste slate was found to have several advantageous uses, there has been no significant increase in these during the 30 years since the report was published. During the 4 years after Bulletin 218 was published (1923–26), waste utilization made insignificant progress, and the conclusion was reached that waste reduction was a more profitable field for research—a conclusion that appears sound today; accordingly, in 1926 the Bureau of Mines attacked this phase of the report. The development and use of wire saws, mentioned in the Background of this report, were important steps in that direction; however much remains to be done in adapting the equipment to new conditions.

The use of jet or sand-blast cutting in awkward corners, where wire saws cannot be used, could be investigated. A reappraisal of the entire problem of waste reduction and waste utilization is in order in the light of current conditions.

Methods of making roofing slates are virtually the same today as 150 years ago. A mechanical slate splitter employing a rapidly vibrating chisel blade was invented nearly 40 years ago. A demonstration with a small machine in New York City was encouraging, but no further steps were taken. Development of mechanical slate splitting is a present-day research problem.

The adaptation of wire saws to use in slate mills is a promising field for experimentation. Where slate blocks are cut ready for splitting into roofing slates or sawed to dimension size for millstock, circular saws are ordinarily used. Wire saws are employed extensively in trimming marble, limestone, and sandstone blocks in mills or yards, and it seems reasonable that they could be used to advantage in slate mills as substitutes for circular saws. In view of the reported advantages of diamond circular saws, their wider use, and the development of improved designs are problems of current interest.

A detailed study of unit operations might reveal ways in which techniques could be improved and costs reduced. The results of the time studies by Pennsylvania State College indicate the operations that offer the most promise for research on improvements in machine design and methods of operation (7).

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CRUSHED AND BROKEN STONE

GENERAL REVIEW

Crushed-stone production is widely distributed in the United States. Plants exist in nearly every State, and the output in 1952 was valued at over $410 million at the point of production. The principal varieties of stone utilized are limestone, traprock, granite, and sandstone. The volume of production has increased over sixfold since 1905 because of the large demands for concrete aggregate, roadstone, agricultural limestone, metallurgical limestone, and a great many other applications.

Crushed stone is prepared by mass-production methods, and prices remain reasonably low. Ever-increasing efficiency in machines and processes is apparent; nevertheless, there is room for improvement in equipment and methods, and the chief problems of the industry relate to development of improved efficiency of operation and more complete utilization of waste.

BACKGROUND

Crushed and broken stone is produced in the United States by approximately 1,700 companies, with quarries widely distributed in almost every State. The total production in 1952 exceeded 300 million short tons, valued at 410 million dollars at the point of production. These figures exclude the large quantities of limestone used for making cement and lime.

Unlike the dimension-stone industries that reached their heyday between 1900 and 1925 and then declined, the crushed-stone industry has experienced a notable and consistent growth from a relatively small industry many years ago. In 1905 the total sales of stone for roadmaking, railroad ballast, and concrete aggregate amounted to less than 28 million tons; in 1952 it exceeded 209 million tons. Two principal factors were responsible for this remarkable growth. First, there was the enormous demand for highway construction stimulated by the advent of the automobile age; the second factor was invention of portland cement and the spectacular growth in the use of reinforced concrete for highways, bridges, public works, and buildings. The demand for crushed stone to be used for aggregate in these structures increased correspondingly.

Over 70 percent of the crushed stone produced consists of limestone, which has a much greater variety of uses than other rocks. Large quantities are used for furnace flux, as agricultural limestone, and in a great variety of chemical and industrial applications, such as alkali, glass, paper, and sugar manufacture. About 10 percent of the total consists of basalt (traprock), about 8 percent granite, and the remainder sandstone, marble, and miscellaneous rocks. Concrete and road metal are the major uses, consuming 50 to 65 percent of the total. The crushed- and broken-stone industry includes, in addition to crusher products, large quantities of heavy, irregular blocks (riprap) used for jetty and seawall construction and for preventing beach or shore erosion. Pennsylvania, Ohio, Illinois, Michigan, and New York are the largest producers of crushed and broken stone.

Methods of quarrying rock for cement manufacture were described in considerable detail many years ago in Bureau of Mines Bulletin 160 (*1*)[4]; the methods and equipment used are similar to those employed in producing crushed stone today. Later data on quarrying and preparing crushed stone are given in The Stone Industries (*3*), and in the chapter on Crushed Stone in Industrial Minerals and Rocks (*6*). Briefly, the process consists of stripping all soil or inferior rock from the surface of a deposit, blasting the rock with heavy charges of explosives in deep drill holes, loading with power shovels, conveying the rock to crushers, and sizing the crushed product with various types of screens. The object is to obtain crushed-stone fragments in various standard sizes. Generally a minimum of fines is desired. The fines may be sold as byproducts; however, at some quarries finely divided materials are the principal products.

The most widely used substitutes for crushed stone are sand, gravel, and blast-furnace slag.

Rocks suitable for the principal crushed-stone uses are widespread and abundant. High-grade limestones suitable for metallurgical and chemical use are less widely distributed; and although reserves are ample for an indefinite period, near-surface deposits are reaching or have reached exhaustion in many localities.

STATISTICS

Statistics of the industry are compiled annually by the Bureau of Mines (*5*). Import and export data are compiled by the Bureau of Mines from figures collected by the United States Department of Commerce.

Imports of crushed stone are small. They consist chiefly of quartzite from Canada and chalk and whiting from Europe. Exports are negligible. The United States is self-sufficient insofar as crushed and broken stone is concerned.

Production and consumption of crushed stone are virtually synonymous, as stocks are

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[4] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

TABLE 6.—*Production of crushed and broken stone, by major uses, 1948–52*

[Short tons]

Year	Concrete and road metal	Railroad ballast	Metallurgical	Agricultural	Other	Total
1948	121, 542, 170	18, 180, 990	34, 901, 940	20, 941, 530	28, 297, 150	223, 863, 780
1949	124, 367, 210	17, 054, 180	30, 752, 320	21, 482, 910	28, 751, 520	222, 408, 140
1950	147, 107, 670	18, 614, 040	35, 969, 820	19, 348, 820	29, 213, 500	250, 253, 850
1951	168, 766, 088	21, 368, 552	39, 929, 957	19, 400, 610	34, 223, 935	283, 689, 142
1952	187, 264, 075	21, 383, 068	34, 908, 815	21, 152, 208	35, 141, 684	299, 849, 850

moderate and more or less constant. Table 6 shows the production, by major uses, for a recent 5-year period.

PRICES

Prices of crushed stone are relatively low, averaging $1.00 to $1.37 per short ton at the point of production during recent years. Certain products, however, may sell as high as $2.50 to $4 per ton. Constantly improving efficiency of equipment and methods, together with near-capacity output, permits maintenance of prices at a comparatively low level even in recent years, when labor and machinery costs have increased greatly.

TRANSPORTATION

As stone is a heavy product, transportation costs constitute a substantial part of the delivered price; hence, the success of a crushed-stone enterprise depends greatly upon its availability to consuming centers. Between 50 and 55 percent is conveyed from quarry to destination by truck and 30 to 35 percent by rail. Water transport is of great importance in some localities, notably from the quarries of northern Michigan and in the New York City metropolitan area.

RESEARCH

Research by industry includes studies of improvements in drilling, including the use of hollow steel and detachable bits. Such work is commonly carried on cooperatively with equipment manufacturers. Some companies have devoted much study to the advantages of delayed-action blasting. Other inquiries have led to substitution of motortruck for rail haulage in quarries, to wider use of diesel power, to substitution of vibrating for revolving screens, to the use of ball breakers to reduce secondary blasting, and to many improvements in plant design. The National Crushed Stone Association has conducted extensive research on testing and grading.

Government research has been conducted by the Public Roads Administration on various aspects of crushed stone used in highway work, and by the Bureau of Reclamation, the United States Corps of Engineers of the Army, and others on durability and other properties of aggregates used in mass concrete construction. Much work on the development of standard methods of testing has been done by the American Society for Testing Materials and the American Association of State Highway Officials.

The Bureau of Mines made a detailed study many years ago of the relative efficiency of various types of equipment and methods used at quarries producing stone for cement and lime manufacture, and the data assembled are applicable in many respects to the quarrying and preparation of crushed stone (*1*, *4*). Later the Bureau reviewed the economic problems involved in crushed-stone production (*2*). A comprehensive study of underground limestone mining was conducted by the Bureau of Mines in 1925 and 1926 (*8*). Thereafter, the Bureau studied unit quarry costs in their relation to efficiency of operation (*9*, *10*). The Bureau of Mines has carried on extensive research on explosives, drilling, blasting, loading, and rock haulage which relate directly or indirectly to stone quarry and mine operations. A notable example is work in the oil-shale mine at Rifle, Colo.

IRON-BLAST-FURNACE SLAG

Because of the many similarities to crushed stone in its processing technology, uses, problems, and outlook commercial iron-blast-furnace slag is included in this chapter.

Although small quantities of iron-blast-furnace slag have been used for centuries, the major commercial development has taken place in the last 40 years. Increasing slag-disposal problems encountered by producers of iron and the growing demand for aggregate in iron-producing areas both were contributing factors. Utilization of the slag happily solves both problems.

The slag produced as a byproduct of iron blast furnaces consists principally of lime, silica, alumina, and magnesia. Processors treat the molten slag in various ways, depending on the

TABLE 7.—*Iron-blast-furnace slag processed in the United States, 1944–48 (average) and 1949–53, by types*

(National Slag Association)

Year	Air-cooled					
	Screened			Unscreened		
	Short tons	Value		Short tons	Value	
		Total	Average per ton		Total	Average per ton
1944–48 (average)	14, 171, 915	$13, 730, 537	$0. 97	566, 408	$256, 550	$0. 45
1949	17, 769, 330	21, 090 445	1. 19	727, 595	372, 727	. 51
1950	20, 047, 844	24, 444, 231	1. 22	1, 005, 436	639, 499	. 64
1951	23, 276, 692	29, 531, 983	1. 27	1, 732, 969	969, 975	. 56
1952	21, 056, 846	27, 501, 892	1. 31	1, 364, 463	749, 375	. 55
1953	24, 021, 624	32, 677, 948	1. 36	845, 311	581, 083	. 69

Year	Granulated		Expanded		
	Short tons	Value [1]	Short tons	Value	
				Total	Average per ton
1944–48 (average)	1, 022, 560	$108, 564	731, 383	$1, 313, 643	$1. 80
1949	1, 885, 428	416, 632	1, 199, 026	2, 698, 908	2. 25
1950	2, 168, 365	647, 665	1, 704, 388	3, 749, 463	2. 20
1951	2, 249, 281	888, 644	2, 068, 492	4, 917, 091	2. 38
1952	2, 507, 604	1, 041, 835	1, 970, 463	4, 581, 107	2. 32
1953	3, 358, 910	1, 250, 450	2, 285, 758	5, 557, 813	2. 43

[1] Excludes value of slag used for hydraulic cement manufacture.

product desired. It can be allowed to cool naturally, forming so-called "air-cooled slag"; it can be granulated by sudden chilling in water; or it can be foamed or expanded—that is, converted to cellular masses—by applying controlled quantities of water.

As shown in the accompanying tables, each type of slag has special applications. Air-cooled slag is crushed and used (screened or unscreened) primarily as concrete aggregate, road-base material, and railroad ballast. Granulated slag requires little or no crushing; its major uses are as road fill and in hydraulic cement manufacture. Expanded slag is crushed and sized for use mainly in lightweight concrete blocks.

The rapid growth of the slag industry is indicated by the increased tonnage processed—from 12,405,922 tons in 1944 to 30,511,603 tons in 1953.

Iron-blast-furnace slag competes with crushed stone, sand and gravel, cinders, expanded clay and shale, and pumice. The use of slag in its various applications is based on physical

TABLE 8.—*Air-cooled iron-blast-furnace slag sold or used by processors in the United States, 1953, by uses*

(National Slag Association)

Use	Screened		Unscreened	
	Short tons	Value	Short tons	Value
Aggregate in:				
Portland cement concrete construction	2, 351, 990	$3, 413, 602	-------	-------
Bituminous construction (all types)	6, 040, 107	9, 007, 418	-------	-------
Highway and airport construction [1]	7, 753, 146	10, 894, 480	533, 998	$363, 644
Manufacture of concrete block	616, 958	853, 246	-------	-------
Railroad ballast	5, 204, 602	5, 349, 918	4, 400	3, 300
Mineral wool	469, 112	642, 552	-------	-------
Roofing (cover material and granules)	449, 677	908, 825	-------	-------
Sewage trickling filter medium	82, 679	142, 308	-------	-------
Agricultural slag, liming	6, 643	10, 790	-------	-------
Other uses	1, 046, 710	1, 454, 809	306, 913	214, 139
Total	24, 021, 624	32, 677, 948	845, 311	581, 083

[1] Other than in portland cement, concrete, and bituminous construction.

TABLE 9.—*Granulated and expanded iron-blast-furnace slag sold or used by processors in the United States, 1953, by uses*

(National Slag Association)

	Granulated		Expanded	
	Short tons	Value	Short tons	Value
Road fill, etc	1,311,311	$804,158	----------	----------
Agricultural slag, liming	89,355	123,805	----------	----------
Manufacture of hydraulic cement	1,413,291	(1)	----------	----------
Aggregate for concrete-block manufacture	302,953	209,987	2,220,117	$5,381,934
Aggregate in lightweight concrete	----------	----------	63,180	168,233
Other uses	242,000	112,500	2,461	7,646
Total	3,358,910	[2] 1,250,450	2,285,758	5,557,813

[1] Data not available.

[2] Excludes value of slag used for hydraulic cement manufacture.

and chemical properties, economy, or availability compared to competitive products.

The unit values of slag in 1953 ranged from $0.46 per ton to $3.11 per ton.

Iron-blast-furnace slag is produced in 15 States, but most of the material is processed in the steel centers of Ohio, Pennsylvania, and Alabama. Slightly over 50 percent of the commercial slag is conveyed from plant to destination by truck; most of the remainder is transported by rail.

The National Slag Association has conducted research to improve use technologies and increase the number of applications. Results of its studies are published.

OUTLOOK

Although the advent of concrete had a depressing effect upon dimension-stone production, it had the opposite effect on the activities in the crushed-stone industry, because concrete construction requires aggregates. With some exceptions, crushed stone follows the general trend of industrial activity. Stone, particularly limestone, has a great variety of uses. The relative importance of these uses fluctuates from year to year, but no striking changes are in prospect. New sources of supply will undoubtedly be developed as older sources are depleted or the demands in new localities increase. Large quarries are opened at times for single projects, such as supplying aggregates for dams or large public works, and when the projects are completed the quarries are abandoned. The introduction during recent years of portable crushing and screening plants has led to migration of production centers from point to point as market demands fluctuate. Even the more stable centers of production may shift in accordance with shifts in population or with regional increases or decreases in industrial development.

Requirements are governed largely by the demands of construction in its broader sense, including buildings, bridges, highways, and public works. The demands for limestone are governed also by metallurgical and other industrial activity.

PROBLEMS

Unlike the problems covered with dimension stone, those of the crushed-stone industry have many elements in common with those of mining in general. Drilling and blasting have primary importance. The types of drills, the use of detachable bits, the number and arrangement of cutting edges, the use of hollow steel, and various other aspects of drilling require constant attention. Churn drilling for open-pit quarrying presents many problems of spacing and burden of drill holes, economical depth, maximum tonnage moved per pound of explosive, and degree of fragmentation. The use of millisecond-delay blasting, TNT fuse, and other blasting aids deserves continued study. Blasting to obtain a minimum of fines is a problem at many crushed-stone plants. Jet piercing may have important applications in quartzites, granites, and other highly siliceous rocks.

Methods of rock transportation have changed greatly during recent years by substitution of truck for rail haulage. The advantages of such substitution and the feasibility of making the necessary changes, where heavy grades are encountered, require study. Another feature that merits attention is placing crushers on the quarry floor and elevating the crushed stone by belt conveyors.

Underground mining has become common in the stone industries. Crushed stone is such a low-priced product that mining cannot be conducted profitably where timbering is required; hence, the problems of stone mining are related to rock strength, pillar spacing, roof scaling, and other items that depend more or less upon the properties of the natural rock, and structures such as joints and bed seams. Where it seems desirable to change from open-pit to underground methods, the problem usually is studied exhaustively both from the standpoint of efficiency of operation and safety of the workers. The problems of underground mining are attaining increasing importance.

Development during recent years of bulldozers and other massive earth-moving equipment has greatly facilitated stripping. Since depth of overburden is a controlling factor in the choice between open-pit and underground mining, a study of stripping methods and costs under modern conditions is a pertinent problem.

Fundamental studies of energy consumption in crushing with various types of equipment have been made by some research organizations. The translation of such data into terms that would help to determine the relative efficiency of gyratory or jaw crushers and rolls or hammer mills has practical interest. Other factors that enter into the choice of crusher are toughness or brittleness of the rock and its tendency to make excessive fines. A review of the techniques of crushing and grinding is a related problem.

The relative merits of the various types of screens and the conditions under which each is most efficient, a review of the equipment and methods used in size classification, and the dust hazards involved are profitable fields for study.

The principal waste materials at crushed-stone plants are fines that sometimes accumulate in excess of demand; however, lately an increasing demand has grown for screenings, particularly in foundation work and concrete-block manufacture. In recent years there has been an increasing demand for small sizes of stone due principally to the wider use of bituminous pavements. Calcareous waste has a great variety of applications, many of which should be explored more fully. Siliceous rocks, such as granites, sandstones, and quartzites, have fewer outlets, but possible uses in the fields of abrasives and fillers could be investigated. A clay overburden may have possibilities as a filler or for making brick or lightweight aggregate. Waste utilization therefore is an ever-present problem.

A matter of growing importance is the difficulty of procuring suitable quarry locations adjacent to some metropolitan areas. To save transportation expense crushing plants should be as near markets as possible, but the growth of suburban areas (and the consequent zoning restrictions) have forced quarry companies to go farther and farther afield. This situation has, in fact, placed a severe restriction on available rock reserves in some areas.

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STRONTIUM

By

Joseph C. Arundale[1]

THE "ROCKETS' RED GLARE" revealed the Star-Spangled Banner still flying above Fort McHenry during its bombardment in 1814. That red glare could have been burning strontium—it was first used about that time to produce red color in flares. Reserves of strontium are large when compared with the drafts that have been made upon them, and there appear to be opportunities to bring this abundant resource into wider use.

Summary

The principal strontium minerals are celestite ($SrSO_4$) and strontianite ($SrCO_3$). Domestic deposits occur in Texas, California, Arizona, Washington, and several other States. Reserves are large compared with current consumption; but the market is small, and the high cost of transportation to the principal consumers discourages competition with strontium minerals from Great Britain and Mexico. Domestic production is very small and erratic. Imports in 1953 amounted to 7,000 short tons valued at about $125,000. Mining usually is by open-pit methods and beneficiation by hand picking or simple gravity concentration. The bulk of the strontium consumed is in the form of its compounds. Uses include fireworks, flares, tracer bullets, fusees, ceramics, medicine, caustic soda purification, desulfurizing steel, beet sugar, and electronic tubes.

The outlook is for supplies to continue to come from foreign sources and for moderate increases in consumption. Problems of the domestic producers include competition from foreign sources and development of new uses.

Active research and other fact-finding activities will be necessary to encourage development of optimum production and use of strontium minerals.

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[1] Assistant chief, Branch of Construction and Chemical Materials, Bureau of Mines.

BACKGROUND

HISTORY

Strontium is widespread in the rocks and waters of the earth's surface, but only two minerals are important commercially. The more important of these (by reason of its more common occurrence) is celestite, the strontium sulfate ($SrSO_4$). Strontianite, the strontium carbonate ($SrCO_3$), is more limited in supply but often occurs associated or mixed with celestite.

IMPORTS [2]

Before 1914 most strontium compounds came from Germany; however, during World War I, when German supplies were not available, the United States established domestic facilities for processing strontium minerals imported from the United Kingdom. The bulk of the strontium minerals consumed in the United States now is celestite imported from United Kingdom and Mexico. Table 1 shows imports of strontium minerals.

DOMESTIC DEPOSITS

Domestic production of strontium minerals has been small and sporadic; however, several deposits are known in the United States. A celestite deposit occurs in the Vulture Mountains of Maricopa County, Ariz., and another a few miles south of Gila Bend. There are celestite deposits a few miles south of Octotillo, San Diego County, Calif.; along the edge of Bristol Dry Lake in San Bernardino County; near the southern end of Death Valley; and in the Cady Mountains north of Ludlow in San Bernardino County. Strontianite occurs in the Mud Hills near Barstow.

In Texas beds, veinlets, and masses of celestite are found in Brown, Nolan, Fisher, Coke, Burnet, Lampasas, and Trinity Counties; in Arkansas, in Howard County; and in Utah, in Emery and Grand Counties. Celestite and strontianite both occur in Skagit and Whatcom Counties, Wash. In Ottawa County, Ohio, certain limestones contain veins and masses of celestite. Deposits of strontium minerals have been reported from Michigan, Missouri, New York, Pennsylvania, Tennessee, and West Virginia.

Although domestic deposits have not been explored thoroughly, reserves in the known deposits have been estimated at several million tons.

FOREIGN DEPOSITS

Celestite and strontianite occur in many localities in England and Scotland; but the important deposits are in Gloucestershire and Somerset, where celestite lenses and irregular masses are associated with gypsum, marl, and dolomitic conglomerates. In Mexico celestite is found in the Provinces of San Luis Potosi and Nuevo Leon, and high-grade lump ore is being exported to the United States from some of them.

In Germany strontianite has been produced near Münster in Westphalia from irregular lenticular veins in marls. Numerous occurrences of vein deposits are reported in Canada. Celestite is associated with sulfur deposits in Sicily, and there are similar deposits in the vicinity of Archangel in U. S. S. R. Large deposits of celestite occur in Spain, and the United States purchased large tonnages from this source as a result of a preclusive buying agreement during World War II.

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[2] Figures on imports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

TABLE 1.—*Strontium minerals [1] imported for consumption in the United States, 1949–53, by countries*

[Short tons]

(U. S. Department of Commerce)

Country	1949		1950		1951		1952		1953	
	Short tons	Value	Short tons	Value	Short tons	Value	Short tons	Value	Short tons	Value
Canada	59	$788	------	--------	38	$382	59	$607	43	$521
Mexico	1, 158	14, 690	1, 975	$23, 910	2, 034	23, 730	1, 297	16, 870	2, 441	30, 248
Spain	3, 263	74, 829	------	--------	------	--------	------	--------	------	--------
United Kingdom	4, 904	86, 378	6, 655	118, 303	11, 972	280, 392	8, 161	168, 849	4, 413	93, 077
Total	9, 384	176, 685	8, 630	142, 213	14, 044	304, 504	9, 517	186, 326	6, 897	123, 846

[1] "Strontianite or mineral strontium carbonate and celestite or mineral strontium sulfate."

PRODUCTION METHODS

Strontium minerals have been produced by simple methods and on a small scale. The Bureau of Mines and a domestic firm have conducted research on a flotation process for recovering celestite from Texas ores. During World War II this ore was processed by grinding and tabling. The concentrate obtained was low grade, and production was discontinued when higher grade foreign material was made available. In England and Mexico the ore is mined by open-pit methods and handpicked. It is reported that some English operations now are using mechanical separation methods.

Users of celestite for chemical purposes generally specify a minimum of 92 percent $SrSO_4$, maximum of 4 percent $CaSO_4$, and maximum of 4 percent $BaSO_4$. Imported material usually is about 95 to 96 percent $SrSO_4$.

Most of the strontium minerals consumed are converted into various chemical compounds. A common procedure for producing these compounds is to digest finely powdered celestite in a hot soda-ash solution, yielding the less soluble strontium carbonate; this is treated with the appropriate acid to make the desired salt. Another method is to burn finely powdered coal with celestite, producing a soluble strontium sulfide. This is leached from the clinker with water and filtered. The solution may be treated with carbon dioxide to produce the carbonate. The nitrate is produced by treating with nitric acid. Strontium metal can be produced by electrolysis of fused strontium chloride and ammonium chloride. It may be produced also by thermal reduction of strontium oxide with metallic aluminum.

PRODUCERS

In the United States the following firms convert strontium minerals into compounds. E. I. du Pont de Nemours Co. at Wilmington, Del., utilizes imported English celestite. Foote Mineral Co., Philadelphia, Pa., uses imported English and Mexican celestite. Barium Products, Ltd., Modesto, Calif., and Pan Chemical Corp., Los Angeles, use small quantities of locally produced celestite.

Strontium metal is produced in the United States by King Laboratories, Inc., Syracuse, N. Y., and by Cooper Metallurgical Associates, Cleveland, Ohio; however, only small quantities are made. Metal Hydrides, Inc., Beverly, Mass., produces small quantities of strontium hydride.

USES

Strontium compounds impart a characteristic brilliant red color to a flame, and this property is utilized in several pyrotechnical applications. Tracer bullets fired from machine guns contain a charge of strontium nitrate and peroxide in the base of the bullet, which are ignited by the propellant and burn brightly during flight. This permits the gunner to judge the accuracy of his aim. Marine distress-signal equipment consists of a pistol and red parachute flares or rockets for large ships. Larger distress rockets are fired to heights of over 1,000 feet, and some release showers of red stars. Similar red flares may be dropped or shot from aircraft or released by submarines. The Armed Forces use various types of red flares for tactical signaling and for illumination at night.

The familiar red color of numerous types of fireworks and pyrotechnical exhibitions is produced by any of several different strontium compounds. Railroads for many years have used red flares and fusees as emergency signals. Many State laws require that trucks carry similar fusees to warn of danger when for any reason they must stop on the highway.

In Germany sugar-beet molasses is treated with strontium hydroxide to recover sugar. The sugar is precipitated as relatively insoluble strontium saccharate, filtered, and treated with carbon dioxide. Strontium carbonate is precipitated and recovered with about 5 percent loss. The sugar in solution is crystallized by evaporation.

Minor applications of strontium compounds are in ceramics, depilatories, medicine, and heat-treating metals.

Strontium metal and some of its alloys are used principally as "getters" for extracting the last traces of gases from electronic tubes. This use is very small—only a few pounds annually. The price of metal is relatively high and ranges from $7 to $35 a pound, depending on quality.

The crude minerals have been utilized in well drilling mud, in purifying caustic soda, as a filler, and in desulfurizing steel, but only small quantities are used in the United States for these purposes at present.

OUTLOOK

For economic reasons most strontium-mineral supplies probably will continue to come from foreign sources; however, as industry and population shift westward, it is conceivable that new plants and areas of consumption will permit development of some of the known domestic deposits.

There are many potential uses for strontium minerals, compounds, metal, and alloys, but an evaluation of these is difficult. Considerable quantities of strontianite have been used in Europe as a flux and desulfurizer in high-grade steels. Domestic steelmakers have experimented with its use in certain grades of basic open-hearth steel as a flux. Strontianite or manufactured strontium carbonate has been suggested as a flux on stainless steel welding rods. The increased use of strontium compounds in glass, enamels, and ceramics is possible. Strontium chloride as a refrigerant and ammonia absorbent in refrigeration is another possibility. Strontium is said to impart valuable properties to nickel, aluminum, and copper alloys.

PROBLEMS

The principal problem of domestic producers or potential producers is competition from foreign sources. The cost of transportation from the principal deposits in the West to consumers in the East is so high, that it discourages competition with foreign sources. The mineral producers and chemical processors share the problem of creating new uses for their products. The price of strontium metal could be reduced substantially if volume applications could be found. The market for strontium minerals in the United States is so small that there is little incentive to develop domestic mines. Their uses are important enough, however, to justify stockpiling or other measures to assure an adequate supply. The supply base is uncertain, particularly in emergencies. Producers, consumers, and Government agencies need reliable technical, statistical, and economic information to guide management and Government agencies concerned with this commodity.

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SULFUR

By

G. W. Josephson[1]

SULFUR permeates and nourishes the entire industrial fabric so thoroughly that an adequate supply is essential. With the experience of the 1950–52 shortage in mind and with the magnitude of native sulfur reserves in doubt, both industry and Government agencies will be paying close attention to this commodity for some time to come.

Summary

Sulfur to supply the industrial requirements of the United States comes from a variety of sources, such as native-sulfur deposits, pyrites, other metallic sulfide ores, natural gas, petroleum, and coal. United States output is adequate for its own needs and provides an export surplus. Imports are small.

United States consumption was restricted by the National Production Authority from June 1951 to November 1952, primarily to achieve equitable distribution of available supplies among consumers. The 1950–52 shortage stimulated development of new sulfur-production capacity, both in the United States and abroad.

The bulk of domestic supply comes from mines at which sulfur is produced by the hot-water-melting (Frasch) process from great deposits of native sulfur in the caprock of salt domes. All non-Frasch sulfur sources are more or less marginal, as they must have some local advantage in order to compete. The reserve position of Frasch sulfur is obscure, but it apparently is adequate for the near future. Reports from the industry indicate, however, that Frasch-sulfur output in the United States will not long be able to keep pace with growth in demand. Over a period of years, therefore, increasing concern with the international trade in sulfur and the problems of conversion to other sulfur raw materials is anticipated.

The industry has many problems, including those concerned with reserves, stockpiles, international trade, taxation, production processes, waste, and markets and the availability of information regarding them.

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[1] Chief, Branch of Construction and Chemical Materials, Bureau of Mines.

BACKGROUND

HISTORY OF SULFUR INDUSTRY

The term "sulfur" in this report will be used in the inclusive chemical sense, and suitable qualifiers will be applied to distinguish the various forms and compounds in which it is produced and used. The word is sometimes used in a narrower sense to denote native sulfur or elemental sulfur, but it is believed that a clearer picture of the commodity can be presented if the application of the term sulfur is not limited to any specific form. Except where otherwise specified, all tonnages mentioned herein will be long tons—the unit of measurement used in the industry.

The sulfur industry is large, annual shipments being valued at over $150,000,000.

Sulfur has a long history that reaches back into antiquity. This history is significant in that it emphasizes the fact that sulfur has been and can be obtained from many raw-material sources.

Native sulfur, pyrites, metallic sulfide ores, petroleum gases, coal, and gypsum all contribute to the total sulfur supply. The primary recovery may be made in the form of elemental sulfur, pyrites, hydrogen sulfide, sulfuric acid, or sulfur dioxide. Adding further complexity, sulfur or its compounds may be recovered from many industrial wastes, such as steel-mill pickle liquor, refinery wastes, smelter gas, and other industrial gases. Some of the sulfur recovered from these latter sources can be properly classified as primary sulfur, but some of it is secondary or mixed with secondary sulfur.

Although numerous possible sources are available, the bulk of the world supply is obtained from only two, native sulfur and pyrites. "Pyrites" is a group term applied to various metallic sulfides, including chalcopyrite and stannite, but generally its usage is limited in the sulfur industry to the iron sulfides pyrite, pyrrhotite, and marcasite. In the past, leadership has seesawed back and forth between native sulfur and pyrites. For a while Sicilian native sulfur mined by underground methods and beneficiated by methods yielding relatively low recovery dominated the market. When the prices of Sicilian sulfur increased to a point that impelled consumers to look for other raw-material sources, pyrites was developed during the middle of the 19th century on a large scale, particularly in Europe.

Then, about the turn of the century, Herman Frasch developed his process for mining sulfur by pumping hot water under pressure through wells into the sulfur horizon of a Gulf coast salt dome to melt the sulfur and bring it to the surface, where, upon cooling, it again solidifies. The product—the so-called "crude sulfur" of commerce—was found to have many advantages in the market. As its purity usually exceeds 99.5 percent, the shipping costs of crude sulfur are low compared with pyrites. Owing to its purity, it can be converted to sulfuric acid in a relatively simple acid plant. An acid plant using pyrites costs about twice as much as one of the same production capacity using crude sulfur. The cost of production of sulfur at Frasch mines was so low that the product could be sold throughout the world at attractive prices. Because of these advantages, Frasch sulfur over a period of years became the preferred sulfur raw material.

In recent years, consumers both here and abroad have based their expansion programs principally on crude sulfur; but it is noteworthy that, even in the United States, some other raw materials have been able to compete with it. In general, the competitive materials are used in the areas where they have local compensatory advantages, such as exceptionally low handling costs, byproduct advantages, nuisance elimination, low raw-material cost, and a good local market.

For example, sulfur in the form of acid is recovered from gases at metallic sulfide smelters. Production costs are low, but the market normally is limited to an area within a few hundred miles of the smelter owing to the high cost of shipping acid. Availability of market, therefore, has been a limiting factor at many smelters because they have been established in areas of sparse population and little industrialization.

A few relatively low cost pyrites mines are operating successfully in the United States in areas where there is a good market for acid. A modest tonnage of pyrites is also imported, principally material recovered as a byproduct of metal-mining operations. Similarly, some byproduct pyrites is recovered and used in the United States itself. The price, f. o. b. mine, of this material has been low to compensate for high transportation and utilization costs compared with those for crude sulfur. The low price of pyrites has made the sulfur market unattractive to many firms that are in a position to produce byproduct pyrites; consequently, large tonnages of pyrites have been and are being discarded.

Compared with Frasch sulfur, pyrite recently has been at a greater disadvantage than it was 15 years ago because production and transportation costs rose faster than sulfur prices. One result of this situation was that importation of Spanish pyrites, which is produced as a primary product, was discontinued.

For many years small quantities of elemental sulfur have been recovered from industrial gases, and recently this output has increased rapidly. Recovery of sulfur from sour (containing hydrogen sulfide) natural gases and refinery gases reduces line corrosion, makes natural gas more marketable as a fuel, reduces atmospheric pollution, and provides a sulfur byproduct.

One of the most abundant sulfur-bearing minerals, gypsum (and anhydrite), has not been used on a large scale as a source of sulfur to date, but in a few localities in Europe conditions have been such that anhydrite could compete.

The present structure of the industry is based fundamentally on the price and availability of Frasch sulfur. This is true in foreign countries, though to a smaller degree, as well as in the United States. During the past 50 years the sulfur requirements of the world have increased greatly as industrial developments have progressed and the uses of chemical fertilizers have increased. During this period the trend has been toward greater dependence on Frasch sulfur.

This trend resulted in a demand that finally caught up to and passed the production capacity of Frasch mines. A number of years elapsed before this fact became evident, because previously the Frasch-mine operators had accumulated very large inventories that were available for shipment. It came as a shock to many consumers when in 1950 a new upsurge in demand, after the invasion of South Korea, brought about a serious enough shortage to require, first, allocation of crude sulfur by the producing companies themselves and, later, quantitative control by the National Production Authority.

Sulfur has been freely available for so many years that it took time for people to become convinced that the shortage was real. Both producers and consumers initiated or intensified studies of their individual situations. To assist and encourage development of new production capacity, the Government offered various incentives, such as rapid tax amortization, purchase contracts, exploration loans, and other forms of loans. The NPA put sulfur under Control Order M–69 on June 2, 1951.

A sulfur committee was established within the International Materials Conference, which was called to consider sulfur and other raw material problems. This committee encouraged sulfur-production development programs and also prepared periodic recommendations for equitable distribution of the available supply.

Rapid progress was made in increasing production capacity. Numerous expansion projects were begun in foreign countries, as well as in the United States. One outstanding feature of the expansion program was the rapid development of plants for producing "recovered sulfur" (elemental sulfur recovered from sour natural and other industrial gases). There was also a swing toward the use of additional pyrite, and sulfur recovery (as acid) was increased at smelters. However, the largest contribution to new output came from Gulf coast domes. The development program included Spindletop, Nash, Damon, and Moss Bluff domes in Texas and Bay Ste. Elaine, Starks, Chacahoula, and Garden Island Bay domes in Louisiana.

At the same time that production capacity was expanding, requirements proved to be lower than anticipated, and the situation eased enough to justify the removal of Control Order M–69 on November 5, 1952. Controls over exports from the United States were continued but were gradually liberalized until in 1954 the commodity was on open-end quota, and export licenses were required for specified countries only.

SELF-SUFFICIENCY

At the present time, the United States is self-sufficient in sulfur and has a surplus of Frasch sulfur available for export. Industry stocks of sulfur have not been built up to the same ratio to consumption that was carried 15 years ago, but they are increasing.

PRODUCTION AND CONSUMPTION

Presentation of a clear statistical picture of the sulfur industry is difficult because the type and quality of statistics available are highly variable and both production and use patterns are exceedingly complex. The United States contributes about 7 percent of the total world production of pyrites and 90 percent of the world production of native sulfur. Of sulfur in all forms, the United States produces almost half of the world total.

Table 1, showing salient statistics of the sulfur industry in the United States, and table 2, containing statistics on production, indicate the size of various branches of the industry. Gypsum statistics are not included. It is

TABLE 1.—*Salient statistics of the sulfur industry in the United States, 1953, in long tons*

	1953	
	Gross weight	Sulfur content
Native sulfur:		
Production:		
From Frasch mines	5, 155, 342	5, 155, 342
From other mines	151, 819	38, 257
Apparent sales	5, 353, 530	5, 239, 968
Imports	1, 229	1, 229
Exports:		
Crude	1, 241, 536	1, 241, 536
Treated	29, 475	29, 475
Apparent consumption	3, 931, 929	3, 931, 929
Producers' stocks at end of year (Frasch sulfur)	3, 022, 486	3, 022, 486
Pyrites:		
Production	922, 647	379, 500
Imports	190, 474	91, 400
Recovery as byproduct:		
Production of byproduct sulfuric acid (basis, percent acid) at Cu, Zn, and Pb plants [1])	775, 069	253, 000
Recovered elemental sulfur:		
Production	344, 020	341, 660
Apparent sales	328, 978	328, 978
Producers' stocks at end of year	107, 000	107, 000
Production of other byproduct sulfur compounds	92, 787	80, 167

[1] Recovery of byproduct sulfuric acid from smelters in 1953 totaled 868,077 short tons.

TABLE 2.—*Production of sulfur and sulfur-containing raw materials by producers in the United States, 1953, in long tons*

	1953	
	Gross weight	Sulfur content
Native sulfur or sulfur ore:		
From Frasch-process mines	5, 155, 342	5, 155, 342
From other mines	151, 819	38, 257
Total native sulfur	5, 307, 161	5, 193, 599
Recovered elemental sulfur:		
Brimstone	342, 297	340, 827
Paste	1, 723	833
Total recovered elemental sulfur	344, 020	341, 660
Pyrites (including coal brasses)	922, 647	379, 500
Byproduct sulfuric acid (basis, 100 percent produced at Cu, Zn, and Pb plants)	775, 069	253, 000
Other byproduct sulfur compounds [1]	92, 787	80, 167
Total equivalent sulfur	------------	6, 247, 926

[1] Hydrogen sulfide and liquid sulfur dioxide. In addition, a quantity of acid sludge is converted to H_2SO_4 but is excluded from the above figures.

apparent from these figures that the United States depends primarily upon Frasch mines for its current supply of sulfur.

Frasch sulfur is produced by only five companies in the United States. The Texas Gulf Sulphur Co. obtains sulfur from Boling dome in Wharton County, Moss Bluff dome in Liberty County and Spindletop dome in Jefferson County, all in Texas. The Freeport Sulphur Co. mines include the Grand Ecaille and Garden Island Bay in Plaquemines Parish and Bay Ste. Elaine in Terrebonne Parish, in Louisiana, and the Hoskins Mound mine in Brazoria County, and Nash dome in Fort Bend and Brazoria Counties, in Texas. Jefferson Lake Sulphur Co. is operating the Clemens dome in Brazoria County and the Long Point dome in Fort Bend County, both in Texas, and Starks dome in Calcasieu Parish, La. The Duval Sulphur & Potash Co. produces sulfur at the Orchard dome in Fort Bend County, Tex. The Standard Sulphur Co. mines sulfur from Damon Mound in Brazoria County, Tex. Freeport also is developing a mine at Chacahoula dome in Lafourche Parish, La.

In addition to the Frasch sulfur, a relatively small quantity of native sulfur is produced annually in the United States from deposits on the surface or at shallow depth; it cannot be mined by the Frasch process.

In 1953 nine companies reported recovery of pyrites. Over 30 companies operated plants for recovering sulfur, hydrogen sulfide, and liquid sulfur dioxide as byproducts in the liquid purification of gas, and 16 recovered sulfuric acid from smelter gases.

Table 3 shows the apparent consumption of the various forms of sulfur in the United States.

Sulfur is used in making an enormous number of products. It is employed in elemental form for some applications and is converted to intermediate compounds for other uses. They are too numerous to be discussed in detail here, but tables 4 and 5 show the magnitude of the major markets served by the sulfur industry. The largest ultimate market for sulfur is in the production of chemical fertilizer, which consumes over one-third of the total.

Sulfur is an important raw material, principally because it can be converted to sulfuric acid—a low-priced, strong acid having innumerable industrial uses (*9*).[2] Over three-fourths of the sulfur used in the United States is converted into sulfuric acid.

Gas to feed a sulfuric acid plant may be a byproduct of smelting, petroleum and natural-gas processing, or other industrial operations, but in most instances the gas is produced by burning elemental sulfur or pyrites. Feed gas must be clean; consequently, auxiliary cleaning equipment is needed in a plant using gas generated in burning pyrites or other sulfide ores. There are two principal methods for manufacturing acid—one is called the chamber process and the other the contact process. Distinguishing features of the chamber process are its use of large lead reaction chambers and gaseous oxides of nitrogen as the catalyst. In the contact process a solid catalyst—usually platinum or vanadium—is used. The chamber process produces weaker acid than

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

does the contact process. The major use of chamber acid is for producing superphosphate. Contact plants produce acid in strengths up to 100 percent H_2SO_4 and also oleums.

TABLE 3.—*Apparent consumption of sulfur in all forms in the United States, 1953, in long tons* [1]

	1953
Native sulfur	3, 931, 929
Recovered sulfur shipments	313, 800
Pyrites:	
Domestic production	379, 500
Imports	91, 000
Total pyrites	470, 500
Smelter acid production	253, 000
Other production [2]	80, 200
Total	5, 049, 429

[1] Crude sulfur or sulfur content.

[2] Hydrogen sulfide and liquid sulfur dioxide. In addition, a quantity of acid sludge is converted to H_2SO_4, but is excluded from the above figures.

TABLE 4.—*Native and recovered sulfur consumed in the United States, 1953, by uses, in long tons* [1]

Use:	
Chemicals:	*1952*
H_2SO_4	2, 925, 000
CS_2	186, 000
Other	92, 000
Pulp and paper	371, 000
Ground crude and refined	198, 000
Miscellaneous	67, 000
Total	3, 839, 000

[1] Source: U. S. Department of Commerce.

TABLE 5.—*Estimates of United States use of sulfuric acid* [1] *(basis, 100 percent), 1951–53, in thousands of short tons*

[Chemical Engineering]

Industry	1951	1952 (revised)	1953 (preliminary)
Fertilizer materials:			
Superphosphate	3, 900	4, 150	4, 050
Ammonium sulfate	1, 500	1, 250	1, 200
Chemicals	3, 800	3, 870	4, 180
Petroleum refining	1, 550	1, 550	1, 720
Paints and pigments	1, 250	1, 250	1, 370
Rayon and film	710	700	760
Iron and steel	1, 080	960	1, 160
Other metallurgical	200	200	220
Industrial explosives	110	110	[2] 420
Textile finishing	40	40	40
Miscellaneous	380	370	410
Total	14, 520	14, 450	15, 530

[1] Recycled acid, including reused concentrated, fortified, and reconstituted acid is estimated at about 2,130,000 short tons in 1951, 2,106,000 tons in 1952, and 2,330,000 tons in 1953.

[2] Includes estimated total acid going into military explosives. About ⅔ goes later into recycled acid.

Additional statistical detail may be found in annual Sulfur and Pyrites chapter of the Bureau of Mines Minerals Yearbook.

PRICES

During the Korean conflict prices of sulfur produced in the United States were frozen by the Government at the figures in effect about the end of 1950. For Frasch sulfur these ranged from $21 to about $27 per long ton, f. o. b. mine. The prices of sulfur produced outside the United States soared, and sales were reported up to and over $200 per long ton.

When the supply situation eased, foreign prices declined rapidly. In 1954 many producers of native sulfur in foreign countries, where costs are so high that their prices must be well above those of Frasch sulfur, were unable to maintain sales; their offerings were rumored to be in the $60- to $80-per-ton range.

After controls were abolished in March 1953 the prices of United States Frasch sulfur increased, and in 1954 quotations generally ranged from $25.50 to $28 per long ton f. o. b. mine and $27 to $33 per long ton f. o. b. vessel.

Elemental sulfur produced at recovery plants processing sour gas commonly is sold at prices comparable with those of Frasch sulfur. However, its pricing is influenced by local market situations, and in some instances it moves at figures higher than the Frasch-sulfur quotations mentioned above. Much of the recovered sulfur is not sold in the open market, as it is consumed by the producing companies.

Pyrites prices vary widely. The average value for all domestic pyrite output approximates $6 per long ton, and producers report f. o. b. mine valuations ranging from less than $1.50 per ton to about $7.

TAXES AND TARIFF

The sulfur industry has the usual taxation problems faced by industry as a whole and in some States is subject to severance taxes. It has one important tax advantage, in that it is accorded a 23-percent depletion allowance for Federal income-tax purposes. The United States does not levy any tariff on sulfur, but export licenses are required in certain instances.

RESERVES AND PRODUCTION METHODS

As previously indicated, consumers generally prefer their sulfur in elemental form. Consequently, those deposits and processes that will yield such a product are the most interesting; and when it is found that the mineral can be produced at an exceedingly low price, as in the case with Frasch sulfur, it naturally becomes very popular. Logically, such a product should be able to take over the market if enough reserves are available. Before 1950 comparatively little consideration was given to the reserve problem; but, when the shortage struck, the pendulum swung to the other extreme. For many years before, the industry had not developed any major deposits, and from this fact people were inclined to conclude that the end was in sight. That this was a premature conclusion is indicated by the number of new Frasch mines that have since been brought into operation.

Well over 200 salt domes are known in the United States; of these, 20 have been brought into production as sulfur mines or are under development. Undoubtedly a number of additional domes contain relatively small but probably ultimately commercial deposits of sulfur. Offshore oil drilling in the Gulf of Mexico is increasing further the number of known sulfur deposits in the United States.

A number of salt domes are known to exist on the Isthmus of Tehuantepec in Mexico, and some minable Frasch sulfur has been found. The Mexican Gulf Sulfur Co. has begun to produce from the San Cristobal dome, as has the Pan American Sulfur Co. at the Jaltipan dome. Texas Gulf Sulfur Co. has been conducting a large-scale exploration program in the same area.

On the other hand, it cannot be assumed that United States reserves are large enough to maintain production at increasing rates for any long period of years. Unfortunately, the reserve position of the industry is not known accurately enough for evaluating clearly the prospects for production at Frasch mines. Reserves are estimated at 50 to 100 million tons at 1951 prices in volume 2 of the Report of the President's Materials Policy Commission (*8*). These figures are highly speculative and probably conservative; but, influenced by geological factors, the report concludes that it is unlikely that Frasch-sulfur reserves can support a withdrawal of 180 million tons over the next 25 years. Reserves of the other forms of sulfur, such as pyrites, other mineral sulfides, anhydrite, and gypsum are ample for all anticipated requirements.

Dome Mines.—A sulfur deposit that can be mined by the Frasch process is an unusual occurrence (*1*). The sulfur occurs in the caprock of salt domes. This caprock consists of anhydrite, limestone, and gypsum. The sulfur content of the caprock varies greatly, and most domes do not contain deposits of commercial size. A typical commercial deposit may have a sulfur-bearing zone about 100 feet thick containing over 20 percent sulfur. To be amenable to Frasch mining, the formation should have some porosity but should be tight enough to permit maintenance of the necessary water temperatures and pressures. Mud is sometimes pumped into the formation to control porosity. The sulfur is mined by pumping water, heated by boilers to about 320° F., through the outer 1 of 3 concentric pipes placed in a well that has been drilled into the formation and cased to the top of the caprock. The water percolates through the formation near the well, heating it to a temperature above the melting point of sulfur (234°–248° F.). Molten sulfur accumulates around the bottom of the pipe and is raised to the surface through the middle pipe by air introduced through the center pipe. The molten sulfur usually is pumped to a "vat" where it is spread on the surface and air-cooled as part of a great monolithic block that is a typical feature of a Frasch mine. For shipment, it usually is broken by blasting and loaded by power shovels into railroad cars. Sulfur can also be shipped in molten form. One firm has put into operation rotary-cooling equipment in which the molten sulfur is solidified in the form of chips suitable for shipment. In conjunction with this method of cooling, a small amount of gas may be injected into the sulfur as it solidifies to improve its grinding characteristics.

At this stage, the sulfur is ready for sale. When sulfur is discolored by hydrocarbons, it may be filtered to brighten its color, but this treatment is not necessary to permit its use for acid and many other applications.

To date a major difficulty at some mines, particularly those in swamp areas, has been to obtain sufficient fresh water for mining operations. For Bay Ste. Elaine dome, Freeport Sulfur Co. designed a plant to use salt water.

Shallow Native-Sulfur Mines.—In addition to Frasch-type deposits, native sulfur is available in the United States in a number of surface or relatively shallow deposits in several Western States.

In the past such deposits usually have been operated on a small scale, the ore being ground and applied to alkaline soils. When beneficiation is attempted, it sometimes takes the form of flotation, which will not yield a product equal to Frasch sulfur in purity. Subsequent melting and filtration will improve the quality approximately to the Frasch level. Autoclaving and solvent methods also have been used, as has distillation. In Italy, Calcaroni and Gill furnaces burn part of the sulfur to recover 50 to 85 percent of the sulfur by melting and fume recovery (*7*).

Sulfur from the Leviathan mine in Alpine County, Calif., which is being used at a nearby copper plant in Nevada, is converted to sulfuric acid without going through a beneficiation stage, as the ore is burned directly.

When large-scale development is attempted, the primary difficulty experienced by most of these operations is the high cost of production, which makes competition with Frasch sulfur difficult at present prices.

Recovered Sulfur.—The recovery of elemental sulfur from sour natural gas and refinery gas has been quite successful. Removal of the hydrogen sulfide from natural and refinery gases provides a sulfur byproduct, simultaneously sweetens the fuel gas, and reduces air pollution. The latter is important in the vicinity of large cities, where smog has become a civic problem. The 1950–52 sulfur shortage gave great impetus to the installation of recovery units, because the producer could thereby obtain additional sulfur exempt from the consumption restrictions of Control Order M–69 and could amortize his investment at an accelerated rate.

Other industrial gases, principally from coal-burning operations, although of large volume in the aggregate, are not very promising sources of sulfur supply. They are usually too lean or too small in quantity for economic recovery; however, in certain large installations, such as coal-carbonization plants, moderate quantities of sulfur can be obtained.

Many processes for recovering sulfur from industrial gases are available. Descriptions of the commercial processes are included in the Sulfur Symposium noted in a bibliographic reference (*5*).

Pyrites.—In 1953 the United States obtained about 380,000 long tons of sulfur equivalent from domestically produced pyrites and imported 91,000 tons more, principally from Canada. As pyrites is the standard alternate for elemental sulfur, it received a great deal of attention when the 1950–52 shortage was at its height.

Undoubtedly the use of pyrites could be greatly increased. Large tonnages of byproduct pyrites are being discarded at metal and coal mines. There are a number of undeveloped deposits of pyritiferous ores in the United States and Canada. For example, a large pyrite deposit at the United Verde mine in Arizona is idle, as are substantial deposits in Maine and Virginia. Numerous pyrite deposits are owned by or leased to major sulfur producers and consumers, and these no doubt could be developed when needed.

The traditional method of using pyrite is to burn it in furnaces, such as lump burners, or multiple-hearth, flash and Fluo Solids roasters, for the production of sulfluric acid or, more rarely, pulp-mill liquors, but pyrite can also be made to yield elemental sulfur. In Europe the Orkla blast-furnace process is used for this purpose, and in Canada another method employing sintering equipment is being developed by Noranda Mines, Ltd.

Recovery of byproduct values from the pyrite cinder is an important aspect in the economics of pyrite utilization. A common practice is to sinter the cinder for use as an iron ore; in some instances other metals, such as copper, may be recovered before sintering.

SMELTER GAS

Substantial quantities of sulfur are recovered at smelters. Gas produced in roasting sulfide ores is dusty; consequently, it is passed through cyclones, Cottrell precipitators, or other cleaning equipment. The clean gas then is fed to an acid plant, where the sulfur is recovered as sulfuric acid suitable for the available market.

Many people have pointed out that smelter gases now being wasted can provide a great deal of additional sulfur. In a sense, this is true, but it must be partly discounted for various physical and economic reasons. In the first place, much of the sulfurous gas produced by a smelter is too dilute to use as a satisfactory feed for an acid plant. In the second place, the transportation cost to distant markets often is prohibitive. In spite of these problems, a considerable increment in capacity has been installed during the past few years.

An interesting variant of sulfur recovery from smelters is being practiced at Copper Cliff, Ontario (*6*). At this location, sulfur dioxide from oxygen flash-smelting units is concentrated and compressed for shipment in liquid form to the Canadian pulp industry.

ANHYDRITE AND GYPSUM

Finally, anhydrite (and gypsum) is a large potential source of sulfur (*1*). A few such plants are now operating in Europe, but none has been constructed in the United States. An unusual association of anhydrite, a fuel such as coal, and markets for both acid and cement in the same area must be found before the process can be operated profitably. It is not now anticipated that anhydrite or gypsum will contribute directly to the sulfur supplies of the United States in the near future; however the use of anhydrite has been sharply increased in Britain.

CONSERVATION

In addition to expansion of the sulfur supply as indicated, much can be done to conserve it. Here, again, questions of economics and modification of standard practice make every proposal an individual problem to be decided on a local basis.

Refinery sludges and steel-mill pickle liquor are wastes from which acid can be recovered in quantity by available processes. Real advances have been made in refineries, but pickle-liquor recovery has made little progress.

Various substitutions and other modifications of consumption are possible, as, for example, substitution of nitrogen for sulfur in making superphosphates. Fusion methods of producing soluble phosphates also receive more attention when the supply of acid is limited. Broader use of the direct application of ground phosphate rock is another possibility.

RESEARCH

In as complex a field as sulfur production and consumption, there is a broad opportunity and need for research, and a great deal has been done in the past. Sulfur producers maintain their own research laboratories and finance research projects at commercial laboratories. An outstanding example of the results of industry research is the variety of processes available for recovering sulfur from gases. Still, as was clearly shown during the 1950–52 shortage, much remains to be done before means will be at hand to draw upon the submarginal sulfur sources without paying a serious financial premium.

OUTLOOK

The sulfur industry anticipates increasing demand and continuing uncertainty regarding the supply trend. Estimates by the defense agencies of primary sulfur requirements (exports included) for 1955 ranged from about 7,000,000 to over 8,300,000 long tons. These may turn out to be high, but there seems to be little reason to doubt that demand will increase materially during the next 10 years. In 1952 the President's Materials Policy Commission (*8*) estimated United States demand for sulfur in 1975 at about 10,000,000 tons.

Since 1950 there has been a very large expansion of Frasch and other sulfur-production capacity in the United States; and, although the prospects in Mexico are not so clear, ultimate output well over 500,000 tons a year in that country would not be surprising. On the other hand, the new Frasch sulfur-production capacity cannot be considered all gain, because declining output is expected from some of the older domes in Texas, particularly the Hoskins Mound mine of Freeport Sulphur Co., Clemens dome of Jefferson Lake Sulphur Co., and Orchard dome of Duval Sulphur & Potash Co. Their exhaustion cannot be predicted with any degree of certainty, because the date will depend on such imponderables as the results of exploration for new reserves along the edges of the deposits and the course in prices and production costs.

Obviously, there is a high order of hazard in both the requirement and production estimates. Even recognizing these deficiencies, it seems logical to believe that supply and demand in the United States will balance fairly well during the next few years, if exports do not become unreasonably high. The longer term prospect is not so encouraging, primarily because of uncertainty regarding Frasch-sulfur reserves.

PROBLEMS

The principal problems of the industry revolve around the questions of reserves of native sulfur that can be mined by the Frasch process and the prices at which Frasch sulfur will be sold. Evidences of decline in the reserve position or increases in the relative price of Frasch sulfur would encourage development of other sources of sulfur now considered submarginal.

The maintenance of supplies adequate for emergency demands is an unusually important commodity problem. Both production capacity and stockpiles contribute to such supplies. Industry stockpiling of sulfur now is an important factor in providing for emergency requirements. The development of additional mines and other sources of sulfur is advantageous.

In the coming 10 years the sulfur industry will be passing through a period in which the reserve position of Frasch sulfur may be clarified considerably, if not completely. If discoveries are not adequate to support Frasch-sulfur output at an accelerating rate, many technical and economic problems will have to be solved to achieve transition to other sources of supply at minimum cost.

Exploration of sulfur-bearing deposits will be needed to throw more light on Frasch and other sulfur reserves. As offshore deposits are found, new exploration and development problems will be faced.

For non-Frasch and conservation projects, research on mining and processing methods that may provide the means of utilizing marginal resources at competitive costs is of major interest. Some of them involve long-range research, such as will be required in investigations of novel methods of utilizing gypsum. The large tonnage of sulfur now being discarded at such operations, as mines, smelters, and steel plants is at once a problem and an opportunity. Both the technical and economic aspects of processes designed to utilize this material deserve attention.

Publications containing details of sulfur resources and requirements and technology are of particular value to potential producers and to consumers facing problems at specific locations.

In addition to those problems cited above, the sulfur industry, in common with industry in general, has many others such as taxation, freight rates, air and stream pollution, and competition with substitutes. Accumulation and dissemination of a wide variety of technical, statistical, and economic information is needed to assist producers, consumers, and Government agencies in solving their sulfur problems.

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TALC, SOAPSTONE, AND PYROPHYLLITE

By

Donald R. Irving [1]

TALC, soapstone, and pyrophyllite, worldwide in occurrence, have been exploited since ancient times in one form or another. Today, these minerals pervade everyday life in such end products as electronic equipment, marking crayons, paper, rubber, pottery, roofing, ceramic tile, asphalt tile, paint, pharmaceuticals, cosmetics, ornaments, insecticides, and many others.

Summary

The earliest recorded uses of soapstone were for carved ornaments, building stone, and cooking utensils. Ground talc first was produced in the United States in New York in 1880. New York consistently has produced more talc than any other State. In recent years North Carolina and California have also been major producers of talc and pyrophyllite as well.

Deposits occur in many States and in most foreign countries; the greatest production usually comes from those areas where industrial development is the greatest. The United States produces about 40 percent of the world total. Strategic block talc is imported from India and Sardinia. High-grade ground talc is imported from India, Italy, and France.

Talc and soapstone deposits normally are formed under conditions of intense dynamic disturbance, through hydrothermal alteration, or by contact metamorphism, from or in association with ultrabasic igneous rocks or sedimentary magnesium carbonate rocks. The whiter, purer talcs are associated with sedimentary carbonate rocks. Pyrophyllite is formed by hydrothermal alteration or replacement of acidic rocks.

Reserves in the United States of all grades of talc other than block steatite appear adequate to meet the foreseeable demand for generations to come.

The physical and chemical properties of talc vary with its mode of origin, and uses often depend on certain properties. Thus, talcs from different deposits cannot always be substituted in industrial applications. Few, if any, minerals surpass talc in multiplicity of uses; the ceramic, paint, rubber, insecticide, roofing, and paper industries consumed 82 percent of the 1953 output of ground talc, soapstone, and pyrophyllite. The balance was used in toilet preparations, pharmaceuticals, foundry facings, refractories, and minor applications.

Demand has expanded rapidly for ground talc, soapstone, and pyrophyllite during the past 20 years with development of new uses in ceramics and insecticides; meanwhile, steady or increased demand has been maintained in formerly major consuming industries. Except for maintaining the supply of block steatite for war emergency use, the industry has no critical or major problems. The opportunity exists for increased production through expanded per capita consumption for known uses and the development of new uses.

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[1] Assistant chief, Branch of Ceramic and Fertilizer Materials ,Bureau of Mines.

BACKGROUND

TERMINOLOGY

Talc, in its broader definition, is an inclusive term covering all gradations from the pure mineral to impure, massive talcose rocks (soapstone) that may contain as little as 50 percent of the mineral talc.

Talc (the mineral) is a hydrous silicate of magnesium, $3MgO{\cdot}4SiO_2{\cdot}H_2O$,[2] theoretically containing 63.5 percent SiO_2, 31.7 percent MgO, and 4.8 percent H_2O. These components may vary within rather wide limits; the magnesia-silica ratio may range from 1:1 to 4:3, and the water content may range from 3 to more than 7 percent. Talc is No. 1 in Mohs' scale of hardness and has a specific gravity of 2.7 to 2.8, a greasy feel, and perfect basal cleavage with pearly luster on cleavage surfaces. It crystallizes in the monoclinic system and usually occurs in soft, foliated or compact, massive aggregates, less often as platy or crystalline aggregates, and occasionally in fibrous or micaceous form. The color ranges from white through gray, yellow, and brown to dark green.

Steatite was originally an alternative mineralogical name, often restricted to massive talc (*12*).[3] It was also used to refer to any massive talcose rock. In recent years *steatite* has been used to designate a grade of talc suitable for making electronic insulators. Specifications for *steatite talc* state the allowable maximum impurities, shrinkage, dielectric properties, and fired color (*11*).

Steatite ceramics are dielectric bodies compounded from ground steatite talc and a suitable flux and binder, formed by dry pressing or extrusion and subsequently fired at relatively high temperatures to form strong finished products. Frequently, these steatite ceramics are referred to only as "steatite."

Block talc is any massive talc or soapstone from which products can be obtained by machining.

Block steatite talc is block talc that meets specifications for electronic insulators. Block steatite talc has been designated as strategic; thus, the two grades of block talc frequently are referred to as *strategic block talc* and *nonstrategic block talc*.

Lava is a term frequently used to designate block talc or even the finished product made from block talc.

French chalk is a soft, massive variety of talc used for marking cloth.

Soapstone in its original meaning was synonymous with steatite. Under present usage *soapstone* includes all massive gray to bluish or greenish talcose rocks, which, with few exceptions, have a soapy feel and can be carved easily with a knife. Virtually all commercial soapstones are highly ferruginous metamorphic rocks composed of 50 to 80 percent of talc intimately mixed with varying proportions of chlorite, amphibole, pyroxene, and mica, as well as pyrite, quartz, calcite, or dolomite. Soapstone therefore is generally a rock rather than a mineral, although the term may be used accurately to refer to massive talc, however pure (*10*). Soapstone grades imperceptibly into talc.

Pyrophyllite is a hydrous aluminum silicate similar to talc in properties and most applications. Its formula is $Al_2O_3{\cdot}4SiO_2{\cdot}H_2O$, with 66.7 percent SiO_2, 28.3 percent Al_2O_3, and 5.0 percent H_2O. It crystallizes in the orthorhombic system and has a hardness of 1 to 2, specific gravity of 2.8 to 2.9, a greasy feel, and basal cleavage. Its color ranges from white to gray or brown green. It usually occurs as compact or foliated masses, less commonly in radial fibrous aggregates. Much pyrophyllite was mined and sold for talc, especially in North Carolina, and doubtless also at other less well known localities, before it was recognized as a separate mineral.

Agalmatolite and *pagodite* are terms applied to massive pyrophyllite from China and Korea, used to produce carved ornaments.

Wonderstone is a term applied to a massive, compact pyrophyllite from South West Africa.

Changes in terminology, similarity in appearance, properties, and uses, and gradational characteristics of talc, soapstone, and pyrophyllite have resulted in considerable confusion. In many instances it is difficult, if not impossible, to determine from the literature just what type of material is meant in the description of occurrences, or in statistical records. Talc, soapstone, and pyrophyllite statistics always have been combined by the Bureau of Mines.

HISTORY

The earliest known uses of soapstone probably depended upon the ease with which it could be cut (*10*). Soapstone was used for Egyptian scarabs and other amulets, for Assyrian cylinder seals and other ancient signets, and for carved ornaments. For centuries the Chinese and Japanese

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[2] In National Research Council Bull. 118, Data on Chemicals for Ceramic Use, June 1949, p. 73, the formula for talc is given as $4MgO{\cdot}5SiO_2{\cdot}H_2O$.

[3] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

have carved art figures out of "steatite" or "agalmatolite." The Cathedral of Trondhjem, Norway, was built about 1200 A.D. of soapstone from Gudbransdal. The stone has hardened considerably since the building was erected; some damage occurred in recent years because of fire and vandalism, but otherwise the stone is in good condition. "Steatite" found in association with serpentine near Cornwall, England, was used in manufacturing old Worcester porcelain.

Soapstone was first used in the United States by the American Indians, who early recognized its heat-retaining qualities and used it for bowls, pots, cooking stoves, and other utensils. This utilization accounts for the term "potstone," which still is applied to soapstone in some localities. The early settlers on this continent made extensive use of heated cut-soapstone bricks as "warmingstones" in carriages and sleighs and even in beds, a practice that survived until recent times. Between 1870 and 1880 the manufacture of tubs, sinks, hearthstones, mantels, griddles, fire brick, and various utensils from soapstone was begun. The first talc-grinding mill was established at Gouverneur, N. Y., about 1880, and the product was used principally for paper filler (*1*).

SIZE AND ORGANIZATION OF INDUSTRY

From 1880 to 1900, inclusive, the output of the talc and soapstone industry of the United States was about 88,000 short tons per year valued at about $1,000,000. These figures included manufactured soapstone items.

In 1951, the peak year, the total production of talc, pyrophyllite, and ground soapstone in the United States was 636,068 short tons valued at $11,322,830. These figures did not include soapstone blocks and slabs.

In 1953 New York was the leading quantity producer of combined talc, pyrophyllite, and ground soapstone; California was second and North Carolina third. New York was the leading talc-producing State; California ranked second and Vermont third. Most of the pyrophyllite was produced in North Carolina; the balance came from California. The pyrophyllite output in 1953 was 20 percent of the total. Soapstone was reported from California, Georgia, Maryland, Nevada, North Carolina, Texas, Virginia, and Washington. Production came from 13 States, with 30 talc, 8 pyrophyllite, and 15 soapstone producers reporting. One company produced talc in California, Nevada, and Montana; 1 company produced pyrophyllite in California and talc in Nevada; and 1 company and its subsidiary produced talc in California and Montana. All other operators produced in only one State.

GEOGRAPHICAL DISTRIBUTION

Talc is a relatively abundant mineral, and talc and soapstone deposits occur in nearly every country in the world. Commercial production has been confined mostly to the countries having the highest industrial development. Pyrophyllite deposits are less widely distributed and usually are smaller than those of talc and soapstone. The United States produced 36 percent of the world output of talc, pyrophyllite, and soapstone in 1953, Japan produced 22 percent, and Europe (excluding Russia) produced 22 percent. Almost all of the Japanese production was pyrophyllite.

TECHNOLOGY

GEOLOGY

Commercial talc deposits are of two classes—those derived from or associated with ultrabasic igneous rocks and those derived from sedimentary magnesium carbonate rocks. Both classes of deposits range in size from a few thousand to several million tons. Although there is considerable variation in each class, talc associated with sedimentary carbonate rock tends to be whiter and purer than talc associated with ultrabasic igneous rocks. High-quality steatite talc is associated only with sedimentary carbonate rocks or, occasionally, with quartzites and granites.

Talc usually is formed on a major scale either under conditions of intensive dynamic disturbance, where the magnesium host rock has suffered severe deformation and squeezing with attendant heat and pressure, or where the host rock has been subjected to contact metamorphism or hydrothermal alteration, usually as a result of igneous activity.

Commercial talc deposits may contain quartz, calcite, dolomite, magnesite, tremolite, serpentine, diopside, anthophyllite, hematite, limonite, and other impurities. In certain instances some of the impurities are essential to commercial use of the talc. For example, the St. Lawrence County, N. Y., commercial "talc" may contain 30 percent or less of mineralogically pure talc. The tremolite, serpentine, and anthophyllite are considered essential "impurities," but accessory minerals, such as hematite, limonite, and pyrite, are deleterious impurities.

Pyrophyllite occurs in irregular, lenticular, or bedded deposits, usually in areas of intense metamorphism. According to Spence (*17*), it is believed to have been formed by hydrothermal alteration of acidic rocks (andesites, rhyolite, tuffs, slates, schists, etc.). Stuckey (*18*) states that the North Carolina deposits were formed through metasomatic replacement of acid tuffs and breccias of both dacitic and rhyolitic composition. Other minerals associated with pyrophyllite are quartz, sericite, pyrite, chloritoid, chlorite, feldspar, iron oxides, epidote, zircon, titanite, rutile, zeolites, and apatite. The North Carolina deposits range from a few feet to 500 feet in width, up to 1,500 feet in length and more than 500 feet in depth.

MINING

Commercial talc, soapstone, and pyrophyllite deposits range from one or several relatively narrow, closely spaced beds or bands to immense, lenticular deposits hundreds of feet thick.

The choice of mining method largely depends on the size, shape, and attitude of the deposit and the topography of the terrain. Mining may be by open-pit or underground methods. Underground entry may be by adit, inclined shaft, or vertical shaft.

When the talc is to be used for cutting into crayons or for block, explosives are kept to a minimum to avoid shattering, and as large blocks as practicable are broken out with bars.

Massive soapstone for dimension stone and other uses usually is quarried. Individual quarry dimensions depend on the width of the vein or dike to be worked, and the condition of the walls. At Schuyler, Va. (*17*), where the largest and most important soapstone quarries in the world are located, quarry depth ranges from 100 to 270 feet. It is reported that 80 to 90 percent of the soapstone removed is waste, partly on account of pyrite (which may become oxidized and discolor the stone), partly on account of hard spots (which make sawing difficult), but chiefly because of small cracks that may cause the slabs to break. The development of chemical-bonding techniques may provide a method of utilizing this waste (*5*).

Pyrophyllite is so soft, particularly in the weathered zone, that it drills and breaks easily. Deposits are worked by open-pit, glory-hole, and underground mining methods. Careful mining and hand selection usually are necessary to minimize impurities.

MILLING

Almost all material receives some treatment before it is sold to consumers, because crude talc, soapstone, and pyrophyllite have no end uses (*7*). A few consumers prefer to buy crude talc and prepare it to their own specifications. A small quantity (less than 0.25 percent of the total sales) is sawed into rectangular prisms or marking crayons each year, and one company in Virginia quarries soapstone and cuts it into slabs or blocks for laboratory equipment, tubs or sinks, furnace linings, and similar uses.

One company in Vermont recovers a flotation product from a mixture of talc, high-iron magnesite (breunnerite), and other impurities. The occurrence of talc deposits with satisfactory properties for almost all industrial applications has discouraged widespread interest in beneficiation, although talc is so amenable to flotation that only a frothing agent is required (*14*, *15*). Ore from some deposits is screened to remove fines and passed over a belt conveyor to permit operators to remove the larger pieces of waste or low-grade ore. At least one company washes the crude ore in a trommel screen to remove fines and facilitate the removal of waste and low-grade ore by hand picking.

Primary crushing is done in jaw or gyratory crushers; secondary crushing is effected in rolls, rotary, or disk-type crushers.

Products ranging in particle size from about 40- to 100-mesh usually are produced in hammer mills, with or without air separation.

Roller mills, in closed circuit with air separators, are the most satisfactory type for finer grinding (100- to 325-mesh) of soft talcs or pyrophyllites. Many roller mills are equipped with combustion chambers which permit simultaneous drying and grinding; some are equipped with magnetic separators.

For more abrasive materials, such as New York talc and North Carolina ceramic-grade pyrophyllite, quartzite- or silex-lined pebble mills are employed, using quartzite pebbles as a grinding medium. These mills are ordinarily in closed circuit with air separators but sometimes are used as batch grinders, especially if products with finer particle sizes are required.

During the past 10 or 15 years fluid-energy grinding mills, popularly called "micronizers," have been developed to make products of finer particle size than any that can be produced by standard mills. Micronizers introduce steam or air under high pressure through nozzles around the periphery of a steel chamber into which the feed drops through a central opening and where the particles are subjected to an intense mutual bombardment. Standard particle sizes from 20 to less than 5 microns are attained. Demand for extra finely ground material for certain uses is increasing.

Crayons are made from selected, medium-hard, massive talc or soapstone, using circular saws similar to those used in wood working. The lumps of crude talc are squared with large saws (18 to 24 inches in diameter). Next, the blocks are sawed into thin slabs equal in thickness to the desired width of the crayons. The thin slabs are cut to final size with 6- to 12-inch-diameter saws, sorted, and packed in small wooden boxes for shipment. For greatest strength, the grain of the talc should run lengthwise of the crayon. Talc is one of the few materials that will make a mark on either hot or cold metal which is visible when the metal is cooled or reheated.

RESERVES

United States reserves of soapstone and all grades of talc except block steatite are adequate to meet any foreseeable future demand. Estimates of developed reserves of steatite-grade talc for grinding range from 200,000 to 400,000 tons; total reserves exceed 1,000,000 tons. The United States has no developed reserves of block steatite talc, although one mine in Montana produced some block steatite talc during World War II. Reserves of ceramic-grade pyrophyllite in North Carolina are small, and in recent years supplies have been allocated to consumers on the basis of past consumption. Lower grade pyrophyllite reserves are ample.

World reserves are large and widely distributed. Japan is almost completely lacking in talc deposits but has estimated reserves of more than 70 million tons of pyrophyllite. In Japan ground pyrophyllite is used satisfactorily for most applications for which talc is used in other countries, so the lack of talc deposits is no hindrance to industrial development.

USES

Few if any minerals surpass talc in multiplicity of industrial uses. Properties that make it so desirable are: Extreme softness and smoothness, good luster, high slip, low moisture, oil and grease absorption, chemical inertness, high fusion point, low electrical and heat conductivity, high dielectric strength, good retention for filler purposes, whiteness, hiding power, and high specific heat. In addition it has the advantage of being relatively abundant, easily mined, and easily prepared for market.

Six industries—ceramics, paint, rubber (both as a dusting powder and filler), insecticides, roofing, and paper—consumed 82 percent of the domestically produced ground talc, pyrophyllite, and soapstone sold in 1953. Other uses include: Toilet preparations; pharmaceuticals; foundry facings; refractories; white-shoe polishes; white-glove cleaners; as a dusting powder, to apply a finish to wire nails; to prevent sticking of bottle, cork, and candy molds or to give a smooth finish to leather goods of all kinds; as a filler in cotton textiles, asphalt, plaster, flooring compounds, linoleum, and plastics; as a component of some cup greases; as an admixture for certain concretes; as an abrasive to polish rice, barley, corn, beans, peanuts, gunpowder grains, and turned wooden articles, like knife handles; and as a bedding material in the steam chests used to cure insulated cable.

Table 1 shows the sales of talc, pyrophyllite, and ground soapstone, by uses, in the United States for 1931 and 1939–53 and changes in the use pattern during the same periods. Ceramics used 1 percent of the total consumption in 1931 and 24 percent in 1953. The insecticide industry before 1941 used only a minor part of the total consumption; in 1941 its consumption was 2 percent but in 1953 it was 16 percent, mostly because of the use of pyrophyllite as a diluent for DDT and other organic insecticides. As the percentage of the total consumption in the ceramic and insecticide industries increased, the percentage of total consumption in the paint and paper industries decreased. The use in paint dropped from 48 percent in 1931 to 20 in 1953, although the total consumption increased from about 80,000 to 118,000 tons. The use in paper dropped from 16 percent in 1931 to 4 in 1953, although the total tonnage remained about the same.

The uses of talc and soapstone, and pyrophyllite are tabulated separately for 1949–53 in tables 2 and 3.

Because of its chemical inertness, massive soapstone is used to make table tops, sinks, drainboards, gutters, fume hoods, shelving, floors, tanks, and miscellaneous special equipment for chemical laboratories. Appreciable quantities are used as furnace linings. In building construction, large quantities of soapstone previously were used for partitions, sills, interior and exterior trim, wainscoting, spandrels, stair treads, and floor tile. Its high dielectric strength, uniform texture, low absorption, flame resistance, high strength, and workability make it desirable for switchboards, panels, barriers, bases, circuit breaker compartments, insulating floor slabs, battery room flooring and shelving, and other related uses in the electrical industry. A small quantity of massive soapstone is used to carve art objects.

Nonstrategic block talc is used for metalworkers' crayons, gas-burner tips, and insulators for electrical appliances. For these uses the talc must be fine grained, homogeneous, and compact, have a low water content, be free from grit, low in iron, and easily machined, and have suitable firing properties.

Steatite-grade talc, mostly ground but including a small quantity of block, is used for manufacturing tube spacers (insulators) for vacuum power tubes.

Talc, soapstone, and pyrophyllite, like many other nonmetallic-mineral commodities, do not

TABLE 1.—*Talc, pyrophyllite, and ground soapstone sold by producers in the United States, by uses, 1931 and 1939–53*

Use	1931		1939		1940		1941		1942		1943		1944		1945	
	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total
Paint	Not available	48	67, 859	27	67, 875	24	120, 319	29	125, 518	32	129, 238	31	110, 767	28	103, 583	26
Ceramics		1	38, 407	15	48, 661	18	78, 990	19	48, 372	13	47, 900	12	31, 932	8	37, 889	10
Rubber		11	31, 078	12	28, 501	10	58, 114	14	40, 487	10	48, 994	12	51, 833	13	63, 758	16
Insecticides		------	(1)	(1)	(1)	------	10, 479	2	15, 810	4	24, 462	6	21, 454	5	37, 012	9
Roofing		11	30, 516	12	34, 347	12	40, 605	10	48, 870	13	47, 500	12	45, 535	11	46, 783	12
Paper		16	30, 177	12	31, 657	11	37, 884	9	30, 440	8	34, 449	8	27, 161	7	28, 138	7
Toilet preparations		3	9, 672	4	8, 818	3	21, 119	5	18, 902	5	17, 682	4	15, 965	4	21, 543	5
Foundry facings		1	3, 986	2	5, 532	2	6, 705	2	7, 822	2	10, 348	3	6, 703	2	7, 072	2
Crayons		------	(1)	(1)	1, 829	1	3, 186	1	1, 474	(1)	1, 446	(2)	1, 358	(2)	336	(2)
Other (specified) [4]		9	12, 918	5	29, 502	10	24, 280	6	27, 631	7	29, 528	7	40, 046	10	23, 506	6
Other (not specified)		------	29, 363	11	24, 653	9	14, 688	3	22, 637	6	21, 321	5	46, 109	12	28, 764	7
Total	163, 752	100	253, 976	100	281, 375	100	416, 369	100	387, 963	100	412, 868	100	398, 863	100	398, 384	100

Use	1946		1947		1948		1949		1950 [3]		1951 [3]		1952 [3]		1953	
	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total
Paint	103, 828	23	112, 101	22	108, 505	21	100, 100	22	145, 000	23	122, 557	19	125, 126	21	118, 383	20
Ceramics	62, 389	14	94, 755	18	107, 907	21	94, 700	20	148, 500	24	170, 521	27	143, 161	24	140, 429	24
Rubber	65, 980	14	71, 840	14	66, 226	13	53, 400	12	75, 900	12	70, 970	11	64, 476	11	57, 711	10
Insecticides	64, 954	14	66, 952	13	72, 740	14	61, 100	13	77, 000	12	90, 418	14	87, 361	15	92, 627	16
Roofing	54, 581	12	63, 545	12	54, 990	11	44, 200	10	55, 400	9	64, 768	10	49, 561	8	55, 358	9
Paper	32, 300	7	31, 772	6	32, 430	6	25, 300	5	29, 600	5	27, 974	5	26, 327	4	25, 045	4
Toilet preparations	17, 111	4	13, 147	3	7, 431	1	8, 400	2	11, 700	2	7, 946	1	8, 811	2	8, 426	1
Foundry facings	7, 045	2	7, 496	1	6, 764	1	5, 300	1	7, 800	1	7, 986	1	7, 279	1	7, 502	1
Crayons	306	(2)	603	(2)	400	(2)	560	(2)	600	(2)	738	(2)	703	(2)	660	(2)
Other (specified) [4]	24, 803	5	36, 120	7	37, 995	7	47, 300	10	48, 000	8	56, 675	9	67, 924	12	78, 715	13
Other (not specified)	23, 769	5	17, 763	4	23, 358	5	21, 536	5	21, 250	4	15, 515	3	12, 418	2	13, 743	2
Total	457, 066	100	516, 094	100	518, 746	100	461, 896	100	620, 750	100	636, 068	100	593, 147	100	598, 599	100

[1] Included in "Other."
[2] Less than 0.5 percent.
[3] Partly estimated.
[4] Includes concrete admixture, pharmaceuticals, plastic cement, tiling, wall plaster, wire insulation, textiles, stove linings, lubricants, cleansing preparations, cork, crayons, bleaching, and insecticides.

SOURCE: Bureau of Mines, Minerals Yearbooks.

TABLE 2.—*Talc and ground soapstone sold by producers in the United States, 1949–53, by uses*

Use	1949		1950		1951		1952		1953	
	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total
Ceramics	77, 439	21	125, 781	25	119, 938	23	120, 404	26	113, 406	24
Paint	97, 212	26	140, 166	28	142, 852	28	117, 046	25	114, 216	24
Rubber	28, 952	8	42, 377	8	53, 781	10	52, 280	11	57, 762	12
Roofing	43, 424	11	54, 204	11	63, 568	12	48, 721	10	53, 858	11
Insecticides	34, 175	9	42, 260	8	46, 334	9	33, 305	7	28, 440	6
Paper	25, 294	7	29, 566	6	27, 884	5	26, 327	6	25, 018	5
Asphalt filler	9, 482	3	6, 931	1	11, 229	2	23, 005	5	21, 305	4
Textiles	13, 909	4	13, 311	3	11, 414	2	12, 029	3	9, 811	2
Toilet preparations	7, 749	2	11, 168	2	7, 916	2	8, 361	2	8, 126	2
Foundry facings	5, 307	1	7, 800	2	7, 986	2	7, 279	1	7, 502	2
Rice polish	645	(1)	2, 845	1	1, 944	1	1, 438	(1)	2, 624	1
Crayons	560	(1)	600	(1)	738	(1)	703	(1)	660	(1)
Other	28, 887	8	25, 932	5	21, 640	4	17, 762	4	34, 334	7
Total	373, 035	100	502, 941	100	517, 224	100	468, 660	100	477, 062	100

[1] Less than 0.5 percent.

TABLE 3.—*Pyrophyllite sold by producers in the United States, 1949–53, by uses*

Use	1949		1950		1951		1952		1953	
	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total	Short tons	Percent of total
Insecticides	26, 897	30	34, 667	29	36, 637	31	35, 081	28	34, 865	28
Rubber	24, 462	28	33, 510	29	27, 086	23	31, 171	25	29, 271	24
Ceramics	17, 226	19	22, 658	19	27, 669	23	26, 115	21	26, 213	22
Refractories	8, 135	9	13, 236	11	15, 020	13	15, 507	12	15, 565	13
Plaster products	6, 582	7	6, 975	6	7, 916	7	10, 570	9	6, 929	6
Paint	2, 888	3	4, 779	4	2, 619	2	4, 722	4	4, 977	4
Roofing	760	1	1, 200	1	1, 200	1	840	1	1, 500	1
Other	1, 911	3	784	1	697	(1)	481	(1)	2, 217	2
Total	88, 861	100	117, 809	100	118, 844	100	124, 487	100	121, 537	100

[1] Less than 0.5 percent.

have constant physical and chemical properties. Each deposit has its own special characteristics. For example, the talc produced in New York contains 40 to more than 60 percent residual or semialtered tremolite and substantial quantities of anthophyllite and serpentine. Its fibrous structure and its whiteness when ground make it a desirable paint pigment, although it does not have the usual talc properties of slip and softness. Talc deposits in California range from those containing steatite-grade talc to others that contain mostly medium-grained tremolite schist, in part altered to serpentine and talc.

Because of the differences in properties, talc from any one deposit cannot be utilized for all purposes, and a variety of domestic and foreign supplies can enter an area where industrial activity is large and diverse, even when the area is near large producing mines and mills. New York is the classic example, as it receives talc from India, Italy, France, Canada, California, Georgia, Maryland, Montana, North Carolina, Texas, Vermont, and Virginia, in addition to that from its own mines. The large number of talcs available does not mean that they cannot be interchanged for many of their uses. It does illustrate the importance of marketing activities, such as customer preference or resistance to change, the aggressive sales policy of certain producers or brokers, personalities, availability, and selling price.

Only the qualities of talc, soapstone, and pyrophyllite that command premium prices can be shipped appreciable distances from the source.

No country exceeds the industrial diversification of the United States; consequently, no other country utilizes talc, soapstone, or pyrophyllite for so many purposes. The uses listed for the United States represent the range of known uses (except, perhaps, for very local and minor applications) in the rest of the world.

BUREAU OF MINES RESEARCH ON STEATITE-GRADE TALC DURING WORLD WAR II

Before World War I most of the steatite-grade talc used in the United States was from foreign countries, including France, Italy, and Manchuria. During World War I California became an important producer and between World War I and World War II was the only domestic source for talc used in steatite ceramics (*16*). When the United States entered World War II, ceramic manufacturers depended almost exclusively on supplies from one California mine.

In order to develop additional supplies of steatite talc suitable for manufacture of steatite ceramics, the Bureau of Mines initiated an investigation of domestic talcs in December 1941 at the request of the Army. Limits on the maximum quantities of Fe_2O_3 and CaO permissible in steatite-grade talcs first were set at 1.0 percent Fe_2O_3 and 1.0 percent CaO, based on specifications set by manufacturers. Individual manufacturers had formulated their own standards, ranging from 0.6 to 1.0 percent CaO and 0.1 to 1.5 percent Fe_2O_3. Despite these chemical limitations, some manufacturers added lime to their bodies, used blends of pure and high-lime talcs, or in other ways disregarded their own standards. The imposed limitations resulted from unsatisfactory experiences in maintaining uniform shipments. The California talc most commonly used contained approximately 1 percent Fe_2O_3, yet a talc with 0.8 percent Fe_2O_3 was rejected because of its high iron oxide content. Another sample containing 2 percent Fe_2O_3 was judged satisfactory by a different company.

The Bureau of Mines studies disclosed that steatite-grade talc must be not only within certain limits chemically but must have the proper physical properties to fit the common manufacturing process. Finally, it must give a fired body that satisfies definite electrical and mechanical requirements. One objection to setting the same limits for all talcs is the commercial practice of blending talcs of different characteristics. Thus, a talc with high shrinkage may be mixed with one of lower shrinkage in proportions that will yield an intermediate-shrinkage product. Either one of these talcs might be unsatisfactory when used alone. Under such circumstances, definite limits are difficult to set.

The Bureau of Mines adopted the following specifications for use in evaluation of steatite-grade talc samples:

Fe_2O_3, maximum limit 1.5 percent.
CaO, maximum limit 1.5 percent.
Al_2O_3, maximum limit 4.0 percent.

The total of all nontalc minerals should not exceed 5 to 10 percent and preferably should be not more than 1 to 2 percent, to be determined by microscopic examination or by flotation. After heat treatment for 1 hour at 2,300° F., the color should be not darker than that of the standard California talc treated under similar conditions. The talc should show no more loss in an abrasion test than the standard California talc. If some of these limits are exceeded slightly the talc may be recommended for blending.

Essentially, these specifications were accepted by industry.

Progress in the Bureau of Mines studies on talc during the early part of World War II was made available to representatives of the Army and Navy through frequent interim reports.

WORLD WAR II CONTROLS

To conserve existing supplies for essential wartime uses, the War Production Board in 1942, through Conservation Order M–239, prohibited the sale of talcs containing less than 1.5 percent Fe_2O_3 and 1.5 percent CaO, except for:

insulators in communications, radio, radar, and underground sound instruments, spark plugs, steatite porcelain products used for electrical and heat insulating purposes (exclusive of refractories), electric light bulbs, filtering of foods, flavoring of extracts, medicines, health supplies, educational, testing, experimentation, and research.

Producers of steatite talc soon discovered that the market, as limited by the order, was insufficient to keep the mines open. Meanwhile, certain other types of talc were becoming scarce. To correct this situation, the order was revised on April 29, 1943, to permit inventories of steatite talc for use in cosmetics, insecticides, and roofing materials equal to a 2-month supply. In more essential products, such as electronic insulators, sparkplugs, and whiteware, a 12-month supply was permitted. All other users were limited to a 3-month supply. The inventory applied to raw materials, semiprocessed materials, finished parts, or subassemblies. Grinding of steatite talc to a finer mesh than is suitable for insulator manufacture was limited to that quantity needed to fill orders calling for delivery within 30 days of the grinding. Each processor of talc was required to report monthly, and each consumer of more than 1 ton of talc a month was required to report quarterly.

On March 28, 1944, M–239 was revised further to permit a 6-month inventory of steatite for use in other than electronic insulators, food processing, and medicinal preparations. The number of reporting consumers was reduced by requiring only those using more than 3 tons a month to file quarterly reports of consumption.

Thus, through the efforts of the Government agencies concerned with raw-materials supply and the domestic talc producers, the condition of scarcity that developed in 1942 disappeared in 1943.

SUBSTITUTES

Because talc and soapstone are abundant and widely distributed, there is no need to develop substitutes for them except for block steatite talc. These commodities and pyrophyllite are subject to competition from each other and from kaolin, fuller's earth, limestone, and other nonmetallic fillers, and from feldspar for ceramics, partly on a price basis and partly on a performance basis.

For example, although pyrophyllite may be more suitable as a diluent for organic insecticides because it is more neutral, talc, soapstone, fuller's earth, kaolin, diatomite, or other diluents may be used in areas where pyrophyllite costs much more or when other properties of the filler are more important. Another example is the substitution of kaolin for New York and Vermont talc as paper filler. Despite almost complete substitution, the production of talc continued to increase because of demand from other applications.

The difficulty of obtaining adequate supplies of block steatite talc during World War II emphasized the desirability of obtaining a domestic substitute material, and research was begun independently by the Federal Bureau of Mines at Norris, Tenn., and the American Lava Corp., Chattanooga, Tenn., to produce a builtup talc block from ground steatite talc and phosphate bonding agents. After the investigation was well advanced in both laboratories, each learned of the other's activities and the work was consolidated and an article prepared for publication (*5*). Subsequently, the Bureau of Mines developed a builtup talc block using magnesium oxychloride as a bonding agent, but preliminary tests indicate that it is inferior in mechanical strength and machining properties to phosphate-bonded talc.

Two other materials that show considerable promise as substitutes for block steatite talc are hot-pressed synthetic mica and phosphate-bonded synthetic mica (*4*, *6*). Table 4 compares the properties of the substitute materials and those of Indian block steatite talc.

The degassing characteristics and life expectancy of insulators made from these substitute materials are being evaluated by power-tube manufacturers under a program sponsored by the Industrial Division, United States Army Signal Corps Supply Agency, Philadelphia, Pa. It is estimated that the evaluation, under proper impetus, can be completed in 2 years.

TABLE 4.—*Properties of various machinable dielectrics*

Property	Natural block talc (India)[1]	Phosphate-bonded talc (dry-pressed)[1]	Synthetic mica	
			Hot-pressed [2]	Phosphate-bonded [3]
Specific gravity (gm./cc.):				
Unfired	2. 5	2. 3–2. 5	----------	2. 1–2. 4
Fired	2. 8	2. 4–2. 6	2. 5–3. 5	2. 0–2. 5
Hardness (Mohs' scale)	[4] 1	[4] 1–2	2–4	2–3
Water absorption (percent)	2–3	3. 5–9	0. 05–10	3–10
Linear firing shrinkage (percent)	1. 5–2. 0	1. 5–2. 5	None	0–4
Flexural strength (p. s. i.):				
Unfired	----------	200–400	10, 000	200–500
Fired	9, 000	4, 500–7, 000	----------	5, 000–9, 000
Compressive strength (p. s. i.):				
Unfired	----------	700–1, 000	35, 000 plus	Not determined
Fired	30, 000	25, 000–40, 000	Not determined	Do.
Dielectric constant (1 mc.): Room temperature	5. 8	5. 5	5. 8	4–7
Power factor (percent at 1 mc.):				
Room temperature	0. 03	0. 06	0. 03–0. 3	0. 03–0. 3
200° C	0. 13	0. 08	0. 05–0. 3	0. 4–0. 6
350° C	0. 7	0. 13	0. 8–1. 0	1. 1–2. 0

[1] Comeforo, J. E., Breedlove, J. G., and Thurnauer, Hans, Phosphate-Bonded Talc: A Superior Block-Talc Substitute: Jour. Am. Ceram. Soc., vol. 37, No. 4, April 1954, pp. 191–195.

[2] Comeforo, J. E., Hatch, R. A., Humphrey, R. A., and Eitel, Wilhelm, Synthetic Mica Investigations, I: A Hot-Pressed Machinable Ceramic Dielectric: Jour. Am. Ceram. Soc., vol. 36, No. 9, September 1953, pp. 286–294.

[3] Comeforo, J. E., Synthetic Mica Investigations, VIII: A Low-Shrinkage, Machinable Ceramic and Phosphate Bonded Synthetic Mica; submitted for pub. to the Am. Ceram. Soc. in 1954.

[4] Unfired.

PRODUCTION, CONSUMPTION, AND FOREIGN TRADE

Table 5 shows the sales, imports, exports, and apparent consumption of talc, pyrophyllite, and ground soapstone in the United States from 1925 to 1953.

Crude talc, ground soapstone, and pyrophyllite produced in the United States, 1952–53, by States, is shown in table 6.

Ground talc, soapstone, and pyrophyllite sold by grinders in the United States, 1952–53, by States, is shown in table 7.

Pyrophyllite produced and sold by producers in the United States, 1944–48 (average) and 1949–53, is shown in table 8.

Table 9 lists world production of talc, pyrophyllite, and soapstone, by countries, 1944–48 (average) and 1949–53, in metric tons.

During World War II supplies of block steatite talc remained in tight supply. The bulk of the material was imported from India, much of it by airplane, under priority and at great expense. Rejections of the material received were high, frequently exceeding 50 percent and occasionally reaching 70 percent. Some material was imported from Sardinia (Italy) after the occupation. A small quantity of block steatite talc was obtained from one deposit in Montana. The War Production Board classified block steatite talc in Group 1 on its Material Substitution and Supply List.

TABLE 5.—*Talc, pyrophyllite, and ground soapstone apparent consumption in the United States, 1925–53*

Year	Sold by producers		Imports for consumption		Exports		Apparent consumption
	Short tons	Value	Short tons	Value	Short tons	Value	Short tons [1]
1925	182,256	$2,011,793	20,993	$450,532	[2] 1,734	$1,882,196	201,515
1926	181,568	2,110,994	23,841	540,082	[2] 1,645	1,726,933	203,764
1927	192,316	2,234,724	25,194	550,382	[2] 1,627	1,621,068	215,883
1928	202,976	2,537,994	27,049	579,915	[2] 1,568	1,605,630	228,457
1929	219,783	2,628,662	31,177	671,629	(3)	1,592,301	(4)
1930	179,385	2,108,338	25,779	529,822	[5] 478	1,484,338	204,686
1931	163,752	1,852,472	23,481	433,682	(6)	1,244,525	(4)
1932	123,221	1,361,633	20,088	359,935	(6)	646,605	(4)
1933	166,023	1,731,882	22,147	391,516	3,956	686,415	184,214
1934	138,505	1,448,685	20,449	426,369	4,903	681,934	154,051
1935	172,716	1,848,055	23,896	492,274	5,814	812,673	190,798
1936	216,191	1,343,171	24,520	456,667	6,670	919,005	234,041
1937	229,999	2,561,753	26,876	472,819	8,878	1,116,098	247,997
1938	212,775	2,302,560	22,127	391,198	7,118	1,102,294	227,784
1939	253,976	2,700,834	26,170	425,221	9,047	1,277,602	271,099
1940	281,375	3,008,320	28,363	487,267	9,402	1,113,522	300,336
1941	416,369	4,701,892	18,637	242,248	10,889	1,445,720	424,117
1942	387,963	4,754,076	8,778	100,476	9,246	870,213	387,495
1943	412,868	5,121,414	6,610	77,055	10,693	992,292	408,785
1944	398,863	5,017,462	8,478	168,983	10,709	1,558,183	396,632
1945	398,384	5,407,235	6,699	101,858	11,141	2,555,936	393,942
1946	457,066	6,445,344	18,449	400,267	16,373	3,912,626	459,142
1947	516,094	7,682,481	17,704	424,923	17,557	4,681,964	516,241
1948	518,746	8,265,363	18,337	518,825	16,327	2,661,132	520,756
1949	461,896	7,523,478	18,816	577,114	15,841	2,076,646	464,871
1950	620,750	10,620,743	23,387	691,678	20,644	1,820,562	623,493
1951	636,068	11,322,830	20,640	694,066	23,009	2,168,815	633,699
1952	593,147	11,347,317	20,302	726,846	23,223	2,002,317	590,226
1953	598,599	11,244,219	22,803	716,709	23,230	1,993,765	598,172

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[1] Compiled from incomplete data. Estimated to be 1,500 to 3,000 short tons high after 1928.
[2] "Talcum and other toilet powders."
[3] "Powders, talcum, face, compact, bath, and other toilet powders." Quantity not recorded.
[4] Not available.
[5] Does not include "Powders, talcum, face, and compact."
[6] Does not include "Powders, talcum (in package), face, and compact." Quantity not recorded.

TABLE 6.—*Crude talc, ground soapstone, and pyrophyllite produced in the United States, 1952–53, by States*

State	1952		1953	
	Short tons	Value [1]	Short tons	Value [2]
California	123,793	(1)	126,422	$1,132,700
Georgia	58,411	(1)	57,891	202,619
Maryland and Virginia	36,963	(1)	37,358	131,744
Nevada	7,580	(1)	10,906	72,971
New York	150,138	(1)	156,299	940,541
North Carolina	116,722	(1)	119,341	578,239
Pennsylvania	---------	---------	[3] 2,463	4,926
Texas	17,495	(1)	6,198	5,580
Vermont	72,533	(1)	80,209	240,627
Washington	(4)	(1)	5,351	28,833
Other States [5]	17,273	(1)	19,068	120,177
Total	600,908	(1)	621,506	3,458,957

[1] Data not available.
[2] Partly estimated.
[3] Sericite schist.
[4] Included with "Other States."
[5] Includes Arkansas (1953), Montana (1952–53), and Washington (1952).

Between World War II and the beginning of Korean hostilities India was again the major source of block steatite talc imported into the United States. Since 1950 both Indian and Sardinian talcs have been accepted for the National Stockpile. Sardinian material has been in greater supply but is not suitable for ultra-high-frequency insulators. Block steatite talc is classified as strategic and critical.

Supplies of Indian and Sardinian block steatite talc appear to be adequate; but, in emergencies, labor and transportation difficulties increase to the point where the cost of suitable material delivered in the United States is exceptionally high and access to foreign sources may be cut off. A reasonable stockpile of block steatite talc can obviate most of the difficulties encountered during World War II. Industry is cooperative in this effort by attempting to maintain a private stockpile.

TABLE 7.—*Ground talc, soapstone, and pyrophyllite sold by grinders in the United States, 1952–53, by States*

State	1952		1953	
	Short tons	Value	Short tons	Value
California	117,717	$2,904,806	106,606	$2,759,314
Georgia	56,181	575,033	57,581	594,900
Maryland and Virginia	36,024	323,803	35,524	320,285
New York	149,103	4,059,116	155,995	3,950,035
North Carolina	111,291	1,564,341	115,794	1,675,308
Pennsylvania	----------	----------	1,283	10,893
Texas	15,601	202,274	9,653	111,009
Vermont	70,623	899,966	74,778	712,303
Washington	6,064	52,963	2,563	35,294
Other States [1]	10,538	251,849	23,102	558,494
Total	573,142	10,834,151	582,879	10,727,835

[1] Includes Nebraska, Oregon, and Utah.

TABLE 8.—*Pyrophyllite produced and sold by producers in the United States, 1944–48 (average) and 1949–53*

Year	Production (short tons)	Sales					
		Crude		Ground		Total	
		Short tons	Value	Short tons	Value	Short tons	Value
1944–48 (average)	91,814	6,799	$45,781	83,492	$895,888	90,291	$941,669
1949	90,920	5,927	31,489	82,934	1,070,838	88,861	1,102,327
1950	116,800	5,690	30,016	112,119	1,504,141	117,809	1,534,157
1951	120,031	4,446	23,741	114,398	1,664,058	118,844	1,687,799
1952	125,496	4,720	29,922	119,767	1,569,471	124,487	1,599,393
1953 [1]	123,457	2,480	15,564	119,057	1,581,826	121,537	1,597,390

[1] Includes some sericite schist.

TABLE 9.—*World production of talc, soapstone, and pyrophyllite, by countries [1] 1944–48 (average) and 1949–53 [2]*

[Metric tons]

(Compiled by Helen L. Hunt)

Country [1]	1944–48 (average)	1949	1950	1951	1952	1953
North America:						
Canada (shipments)	26,585	24,423	29,578	22,540	22,709	24,370
United States	417,112	416,709	559,440	581,009	545,132	563,818
South America:						
Argentina [3]	3,670	(4)	(4)	15,000	16,000	(4)
Brazil	4,271	17,782	12,632	11,304	(4)	(4)
Chile	681	110	142	25	(4)	(4)
Paraguay	----------	----------	----------	----------	----------	90
Peru	----------	----------	----------	131	124	(4)
Uruguay	2,311	660	681	959	679	891
Europe:						
Austria	28,542	56,050	58,681	72,784	50,822	51,335
Finland	182	----------	300	5,000	6,000	3,688
France	58,405	100,055	87,416	103,236	107,701	103,900
Germany, West	15,160	17,700	27,300	35,200	27,589	40,789
Greece	500	1,700	2,500	2,625	1,200	(4)
Italy	46,985	61,462	67,616	75,996	80,336	80,282
Norway	38,249	54,305	64,099	76,479	64,074	(4)
Portugal	[5] 21	3	2	1	6	16
Rumania [3]	633	(4)	(4)	(4)	(4)	(4)
Spain	24,411	38,208	25,131	36,034	27,859	28,447
Sweden	10,259	11,293	13,843	13,332	8,787	(4)
United Kingdom	3,163	2,616	1,727	2,540	2,628	(4)
Asia:						
Afghanistan	----------	100	75	840	800	600
India	36,007	21,535	25,894	32,314	20,373	(4)
Indochina	106	----------	----------	(4)	(4)	(4)
Japan	208,929	262,433	283,566	400,626	318,386	(4)
Korea, Republic of	10,647	2,773	7,485	3,208	3,964	24,479
Taiwan (Formosa)	----------	76	700	2,057	1,093	1,764
Africa:						
Egypt	4,609	5,573	3,731	3,754	4,903	2,198
Kenya	287	590	334	337	235	155
Union of South Africa	3,220	5,386	3,978	5,663	8,674	7,234
Oceania:						
Australia	5,802	8,717	9,851	13,359	7,772	9,963
New Zealand	5	----------	----------	----------	----------	----------
Total (est.) [1]	1,025,000	1,275,000	1,450,000	1,650,000	1,500,000	1,550,000

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[1] In addition to countries listed, talc or pyrophyllite is reported in China and U. S. S. R., but data on production are not available; estimates have been included in total.

[2] This table incorporates a number of revisions of data published in previous Mineral Yearbook Talc chapters.

[3] Estimate.

[4] Data not available; estimate by author included in total.

[5] Average for 1 year only, as 1948 was first year of production.

In 1945 (*13*) there were 27 steatite-ceramic insulator-manufacturing plants compared with 5 before the war. Despite the closer tolerances required and the radically different production technique, a number of plants previously making electrical porcelain, wall tile, and other products successfully converted to steatite-insulator manufacture. These plant conversions, involving relatively little new equipment, made possible the production of 21.6 million dollars worth of steatite ceramic products in 1943 compared with only 1.6 million dollars worth in 1939. This expansion demonstrates the tremendous manufacturing potential available for an emergency.

PRICES

Prices usually are quoted on the short-ton basis, f. o. b. cars at the grinding mill, and vary according to the type of talc, pyrophyllite, or soapstone, fineness of grind, and purity. In general, prices of domestic products are relatively low because of competition. In some instances certain grades may be sold at prices lower than the cost of production in an effort to market a surplus of offgrade or highly competitive products. Prices quoted in the trade journals are nominal, and actual selling prices are the result of direct negotiation between buyer and seller. Also, trade-journal quotations are incomplete; quotations are not listed for all producing areas and obviously cannot be listed for all the quality and particle-size variations available. In one instance quotations are given for New Jersey, where the last reported production was in 1941.

Talc prices quoted by the E&MJ Metal and Mineral Markets in May 1954 were as follows:

Georgia: 98 percent through 200-mesh, gray, $10.50–$11.00; white, $12.50–$15.00, packed in paper bags.
New Jersey: Mineral pulp, ground, $10.50–$12.50, bags extra.
New York: Double air-floated, short fiber, 325-mesh, $18.00–$20.00.
Vermont: 100 percent through 200-mesh, extra white, bulk basis, $12.50; 99½ percent through 200-mesh, medium white, $11.50–$12.50. Packed in paper bags, $1.75 per ton extra.
Virginia: 200-mesh, $10.00–$12.00; 325-mesh, $12.00–$14.00; crude, $5.50.

Talc prices quoted in the Oil, Paint and Drug Reporter in March 1954, were as follows:

California: Ground, $32.00–$38.50.
Vermont: Offcolor, ground, $14.00.
New York: Fibrous, ground, $25.00–$30.00. 99.5 percent, 325-mesh, $27.00. 99.95 percent, micronized, $36.00.
Canada: Ground, f. o. b. mines, $15.25–$35.00.
Per ton, carlots, f. o. b. works, containers included unless otherwise specified.

TARIFF

Talc, steatite or soapstone, and French-chalk imports into the United States are subject to duty under Paragraph 209 of the tariff regulations under four classifications, as follows: (*a*) Crude and unground, (*b*) ground, washed, powdered, or pulverized (except toilet preparations), (*c*) cut or sawed, or in blanks, crayons, cubes, disks, or other forms, (*d*) manufactures (except toilet preparations) of which talc, steatite or soapstone, or French chalk is the component material of chief value, wholly or partly finished, n. s. p. f. (not specifically provided for).

A summary of the tariff history beginning with the Act of 1922 is shown in table 10.

TRANSPORTATION

Except in California, Nevada, and Montana, grinding mills are near the deposits and generally accessible to the major marketing areas. Mills in the California-Nevada area frequently are 100 or more miles from the mines, and the market for much of the higher grade talc from the Western States is on the east coast, 3,000 miles from the mills. Crude talc from Montana is shipped to Ogden, Utah, and Grand Island, Nebr., for grinding.

The development of high-grade talc deposits in Montana east of the Continental Divide has partly compensated for the unfavorable market position of western talcs caused by higher transportation charges.

RESEARCH

Investigations by the Bureau of Mines (*3, 11, 15*) and by industry (*8, 9*) have resulted in progress toward standardization of industrial steatite-talc specifications and advances in beneficiation techniques for talcs to upgrade them to steatite quality.

Industry and the Bureau of Mines have developed builtup talc block by phosphate bonding (*5*) and the Bureau of Mines has developed hot-pressed and phosphate-bonded synthetic-mica products (*4, 6*) that, on the basis of preliminary tests, appear to be satisfactory substitutes for block steatite talc. A magnesium oxychloride bonded talc developed by the Bureau of Mines has not been evaluated but cannot be excluded from consideration as a potential block-steatite-talc substitute.

TABLE 10.—*Changes in United States rates of duty on talc, steatite, soapstone and French chalk* [1]

Item	Tariff rate in—				
	Act of 1922	Act of 1930	1945	1948	1952
A. Crude and unground (cents per pound [2]): [3]					
Talc, steatite or soapstone, and French chalk, crude and unground	¼	¼	¼	[4] ⅛	[4] ⅛
B. Ground, washed, powdered, or pulverized (except toilet preparations) (percent ad valorem):					
Talc, steatite or soapstone, and French chalk, ground, washed, powdered, or pulverized (except toilet preparations):					
French chalk	25	35	35	35	[5] 17½
Other:					
Valued at not more than $14 per ton	25	35	[6] 17½	[7] 10	[8] 8¾
Valued at more than $14 per ton	25	35	35	35	[5] 17½
C. Cut or sawed or in blanks, or other forms (cents per pound): [9]					
Talc, steatite or soapstone, and French chalk, cut or sawed, or in blanks, crayons, cubes, disks, or other forms	1	1	1	1	[5] ½
D. Manufacturers of, n. s. p. f. (except toilet preparations) (percent ad valorem):					
Manufactures (except toilet preparations), of which talc, steatite or soapstone, or French chalk is the component material of chief value, wholly or partly finished, n. s. p. f.:					
If not decorated	35	35	35	[10] 17½	[11] 35
If decorated	45	45	45	[10] 22½	[11] 45

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[1] U. S. Tariff, par. 209.

[2] Rate of ¼ cent per pound is equal to $5 per short ton; rate of ⅛ cent per pound, to $2.50 per short ton.

[3] The ad valorem equivalent of the 1945 rate of duty on talc, etc., crude and unground, is 27.8 percent based on the average import value in 1939. The corresponding ad valorem equivalent of the 1948 rate is 13.9 percent. Imports in 1946 were too small to furnish a significant basis for computing the equivalent of the duty; moreover, they were of much higher average grade than the prewar imports.

[4] General Agreement on Tariffs and Trade (Geneva), effective June 14, 1948.

[5] General Agreement on Tariffs and Trade (Annecy), effective May 30, 1950.

[6] Trade agreement with Canada, effective January 1939; rate previously reduced in first trade agreement with Canada, effective January 1936, to 25 percent if valued at not more than $12.50 per ton.

[7] General Agreement on Tariffs and Trade (Geneva), effective January 1, 1948.

[8] General Agreement on Tariffs and Trade (Torquay), effective June 6, 1951.

[9] The average ad valorem equivalent of the 1945 and 1948 rates of duty is 12.8 percent based on the value of dutiable imports in 1939 and 6.3 percent based on the value of dutiable imports in 1946. If the 1945 and 1948 rates are calculated on the basis of average unit foreign value of all imports including those entered free for Government use the ad valorem equivalent is 13.8 percent. Most of the imports in 1946 were free for Government use whereas all the imports in 1939 were dutiable.

[10] General Agreement on Tariffs and Trade (Geneva), effective May 22, 1948.

[11] Effective December 11, 1950 as the result of the withdrawal of the GATT concession. This concession was initially negotiated with the Republic of China which had ceased to be a contracting party to the GATT.

OUTLOOK

Consumption of talc, pyrophyllite, and soapstone increased more than threefold from 1931 to 1951 and reached an alltime high of 636,068 tons in 1951. During this period the quantity used in paints and rubber increased 56 and 294 percent, respectively. New uses were developed in ceramics and insecticides, and in 1951 these two uses accounted for 41 percent of the consumption. The per capita consumption in the United States increased from 2.9 to 8.0 pounds per year from 1930 to 1950 and may be expected to increase further as the standard of living rises and new uses for talc, pyrophyllite, and soapstone develop. One potential new use for talc is as a filler in plastics.

Reserves of soapstone, all grades of talc except block steatite, and the lower quality grades of pyrophyllite appear adequate to meet demands in the foreseeable future, even if requirements double. Prospects for new discoveries, extension of reserves in producing mines, and development of known deposits not now in production are good.

Ceramic-grade pyrophyllite is in short supply, but other body formulas using talc, feldspar, wollastonite, and other fluxes could be substituted.

The need for steatite block talc is steadily declining. New techniques developed through research have permitted steatite ceramic bodies to be substituted for all but a few minor applications. Industry and the Bureau of Mines have produced synthetic mica block and builtup talc block. Successful completion of service tests now in progress may eliminate the need for block steatite talc.

Talc, pyrophyllite, and soapstone are bulk low-cost commodities, and the use pattern depends upon an interplay between cost and physical properties. Nevertheless, California talc, of ceramic and paint grades, is shipped to the east coast, and New York talc is shipped to Canada and the west coast. Worldwide expansion in production, can be anticipated, mostly in highly industrialized countries.

PROBLEMS

BLOCK STEATITE TALC

Four materials have suitable physical properties for replacing block steatite talc, but they have not been evaluated to determine whether the degassing characteristics and life expectancy are satisfactory for vacuum power tubes.

Under the sponsorship and general supervision of the Industrial Division of the Signal Corps Supply Agency, Department of the Army, industry has begun a voluntary evaluation program. Because it is voluntary, progress depends upon factors that cannot be controlled by the Signal Corps, and it is estimated that it may require 2 to 4 years to complete the work.

The problem of obtaining a proved replacement for block steatite talc to assure independence of foreign supplies during wartime is the only important domestic problem concerning these commodities. Worldwide, during times of normal trade, known deposits can supply enough block steatite talc to meet the demand.

SOAPSTONE

Large quantities of soapstone are discarded during quarrying and the preparation of blocks and slabs, because of structural defects. The application of chemical-bonding techniques to this quarry waste might result in the development of products to replace those from natural sawed blocks and slabs and at the same time reduce the troublesome disposal problem.

TALC AND PYROPHYLLITE

Continued research by the industry is needed to improve the quality of talc and pyrophyllite products and to lower the cost of their preparation, to prevent the loss of markets to other low-cost materials and to counteract the threat of shrinking marketing areas caused by the continuing increase in transportation costs. Research designed to discover new uses for the plentiful grades of talc, pyrophyllite, and soapstone is also needed. Because each deposit has characteristic properties, the establishment of industry-wide standard specifications is difficult, and each deposit presents problems of its own.

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TELLURIUM

By

John D. Sargent[1]

TELLURIUM is produced as a byproduct of copper and lead refining. Consumption of tellurium has been increasing, but production has been larger than consumption.

Summary

Tellurium is recovered metallurgically from anode slime produced in electrolytic refining of copper and lead. Blister copper is the principal industrial source of tellurium and contains 0.001 to 0.05 percent of tellurium.

The principal uses of tellurium are in the manufacture of alloys, such as tellurium lead, stainless steel, tellurium copper, and tellurium bronzes and in the rubber industry as a secondary vulcanizing agent. Minor uses are as an additive and a core wash to induce chill in manufacturing iron castings and as a coloring agent in the manufacture of art glass and ceramics.

Domestic consumption of tellurium has increased from about 1,000 pounds in 1928 to 22,600 pounds in 1935, to 89,000 pounds in 1940, and to about 100,000 pounds in 1951, 1952, and 1953.

Producers' stocks of refined tellurium equal about a 1-year supply, and stocks of tellurium raw materials constitute an even larger backlog of potential production.

The overall tellurium problem is development of an increased demand for the metal that would absorb the annual output produced in connection with the electrolytic refining of copper and lead.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

HISTORY

In 1782 Muller von Reichenstein suspected the presence of a new metal in an auriferous mineral from the Mariahilf mine near Zalethna, in the Fatzburg Mountains of Transylvania. Tellurium was isolated from this mineral by Kitaibel in 1789, and in 1798 the existence of the new element was confirmed by Klaproth, who gave it the name of tellurium from the Latin word *tellus*, the earth (*5*).[2]

OCCURRENCE

Tellurium occurs native in small quantities associated with silver and gold but is usually found combined with these or other metals as tellurides (*5*). It is recovered metallurgically from anode slime obtained in the electrolytic refining of copper and lead.

METALLURGY

In metallurgical recovery tellurium in anode slime can be converted into tellurite and tellurate of sodium. The alkaline solution of the metal is neutralized with sulfuric acid, and precipitate of TeO_2 is formed. This precipitate is dissolved in strong hydrochloric acid, then diluted with water, and SO_2 is passed into the solution. Tellurium is precipitated as a gray powder. Another method of recovering tellurium consists of dissolving TeO_2 in caustic soda solution and plating out metallic tellurium as a steel-gray, brittle cathode deposit (*11*). Tellurium oxide can also be reduced directly by fusing with carbon in a crucible furnace.

Blister copper is the principal industrial source of both selenium and tellurium. United States blister copper contains 0.01 to 0.14 percent of selenium and 0.001 to 0.05 percent of tellurium.

USES

Much research effort has been devoted by Government agencies, educational institutions, industry, and others to developing new uses for tellurium and the substitution of tellurium for selenium (*13*). From time to time new uses have been found, but they have provided either temporary outlets or involve relatively small quantities of the metal. Endeavor to substitute tellurium for selenium has been, with relatively few exceptions, generally unsuccessful. The physiological characteristics of tellurium have operated against its wider use in industry; it imparts an odor resembling that of garlic to the breath of workers, and it can be a depilatory.

The principal uses of tellurium are (1) in the manufacture of alloys, such as tellurium lead, stainless steel, tellurium copper, and various tellurium bronzes and (2) in the rubber industry as a secondary vulcanizing agent. Minor uses are as an additive and core wash to induce chill in the manufacture of iron castings and as a coloring agent in the manufacture of art glass and ceramics. Tellurium is reported to provide better porosity control than selenium in the manufacture of stainless steel castings.

Tellurium imparts hardness, toughness, and increased corrosion resistance to lead. Tellurium lead is valuable for lining tanks and for acid pipes and coils in plants handling sulfuric acid and hot chromic acid solutions.

A tellurium-copper alloy (99.5 percent copper and 0.5 percent tellurium) with machinability approaching that of free-cutting brass and with electrical conductivity of 90 percent of that of pure copper—making it particularly suitable for the mass production of electrical and electronic connectors and other parts—has recently become available. Tellurium-copper may be extensively hot-worked, inasmuch as the addition of tellurium does not adversely affect the hot-working properties of copper. The alloy may also be severely cold-worked, although it is somewhat less ductile at room temperatures than pure copper.

A tellurium bronze—98.3 percent copper, 1.0 percent nickel, 0.2 percent phosphorus, and 0.5 percent tellurium—is also available for the production of forgings and screw-machine parts that require high strength, hardenability, ease of machining, and corrosion resistance.

Metallic tellurium is used in the rubber industry as a secondary vulcanizing agent to

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

impart resistance to heat, steam, and aging and to provide what the industry terms "a fast, tight cure."

Because an excess supply of tellurium is available producers are not disposed, for competitive reasons, to disclose outlets or end uses involved in their sales.

Considerable experimental work involving the use of tellurium is reported in the electronics industry.

PRODUCTION AND CONSUMPTION

Bureau of Mines Report of Investigations 2385, published in 1922, states "The present uses for tellurium are very limited, the demand being supplied by a few hundred pounds a year." Bureau of Mines Report of Investigations 6317, published in 1930, states "Tellurium has only slight industrial application. In 1928 the estimated output was 2,650 pounds, but sales apparently amounted to only 1,060 pounds." By 1935, however, sales of tellurium had reached 22,600 pounds and in 1940 had totaled 89,000 pounds. In 1951, 1952, and 1953 the apparent United States consumption of tellurium was about 100,000 pounds a year, and exports averaged about 50,000 pounds. Approximately 750,000 pounds of tellurium is available in accumulated slimes and other stocks of tellurium raw materials.

Domestic primary tellurium production declined 63 percent from 189,076 pounds in 1952 to 70,446 pounds in 1953 owing largely to the leading producer of former years reporting no output in 1953.

PRICES

Tellurium has not been affected by the price fluctuations of metals and minerals in the past. It sold for $2.00 a pound for several years before 1938 and has remained unchanged at $1.75 a pound ever since.

OUTLOOK

The quantity of tellurium produced would depend directly on the output of refined copper and lead if all copper and lead refineries recovered all byproduct tellurium. The present tellurium recovery rate is about 0.10 pound per ton of electrolytically refined copper. An increased recovery rate probably would be attained if markets could be developed for larger quantities of tellurium. About a quarter of a million pounds of recoverable tellurium is contained in ores processed each year by the nonferrous metals industry in the United States and Canada.

In view of the increase in apparent consumption of tellurium some optimism would seem warranted relative to the outlook for the metal.

PROBLEMS

The overall problem—present and anticipated—is development of an increased demand for tellurium to absorb the annual output of the metal, which is produced in connection with the making of refined copper.

Tellurium has objectionable physiological effects on the workers that operate against its wider use in the steel industry. Nevertheless, it appears to have large-scale acceptance among base-metal-alloy producers and could probably be made acceptable to steel producers.

In 1951 exports jumped to 64,000 pounds from about 9,000 pounds the year before. A study of export outlets for the metal might disclose additional potential expansion in exports of tellurium in the years ahead.

Research in electronics looking toward greater use of tellurium is reported in progress.

A long-range program for tellurium should have the following objectives, in the order listed:

(1) Development of new uses for the metal through research.

(2) When apparent consumption approaches production, metallurgical investigations should be begun to increase recovery of the metal and to reduce losses in smelter and refinery operations.

(3) Investigations should thereafter be initiated to discover new mineralogical sources of the metal and to recover additional supplies of the metal from such sources.

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THALLIUM

By

John D. Sargent [1]

THALLIUM is a rare metal, chemically resembling lead, produced by a single company and utilized in rat poison.

Summary

Thallium, a soft, bluish-white metal resembling lead, is one of the rare and relatively neglected elements that has assumed importance in recent years. It is widely distributed in the earth's crust, and igneous rocks have been estimated to have an average thallium content of 30 grams per ton. However, it is usually present in ore deposits in such small quantities that direct extraction is not commercially feasible. It is being recovered in the United States as a byproduct from cadmium-production operations at the rate of a few thousand pounds a year.

The comparative scarcity of high-grade thallium ores, the limited knowledge of its properties, and the high cost of extraction have been factors retarding its commercial development. As a result, thallium was more or less a laboratory curiosity, with few industrial applications, until World War II, when a number of commercial and military uses were developed. The principal use of thallium has been in the form of the sulfate, which is the active agent in some rodent poisons. An important, but small, market for thallium is in the form of bromo-iodide crystals used for infrared-signal transmission in military and laboratory equipment. As additional sources of supply become available at lower production costs, the demand for thallium undoubtedly will increase.

Domestic production of thallium is adequate to meet present United States requirements.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

HISTORY

The discovery of thallium dates back to 1861, when W. Crookes noted a new element while making a spectroscopic examination of seleniferous deposits in the lead chambers of the sulfuric acid works at Tilkerode in the Hartz Mountains of Germany. Crookes named the new element thallium (from the Greek word *thallos*—a budding twig), because of the characteristic green line it imparted to the spectrum (*2*).[2]

Thallium compounds were not used until about 1896, when they began to be employed in medicine. Later the use of thallium in highly refractive optical glass was proposed, and in 1919 a photosensitive cell using thallium oxysulfide was patented. Beginning in 1925, the use of thallium salts as poisons for rodents and later as insecticides created an increased demand, and the price rose from $12 to $15 per pound (*10*).

Substantial imports of thallium from Europe in 1932 resulted in a price drop to as low as $5 per pound, and requirements continued to be met with imported material until World War II. In 1942–43, when European supplies were not available, the metal was quoted at $10 a pound and went to $12.50 in 1944, when supply was far below demand. Production in Canada and the United States since 1945 has been adequate to satisfy domestic requirements.

OCCURRENCE

Thallium has never been found in nature in a free or uncombined state; and, although the element has been detected in numerous types of rocks and ores, the quantity present is usually so small that its direct extraction is economically impracticable. Four minerals containing 16 to 60 percent thallium are known, but none of these minerals has been found in sufficient quantity to be of commercial importance. These minerals are: Crooksite, $(Cu,Tl,Ag)_2Se$; lorandite, $TlAsS_2$; hutchinsonite $(Pb,Tl)_2(Cu,Ag)As_5S_{10}$; and vrbaite, $TlAs_2SbS_5$ (*10*).

Commercial sources of thallium are the minute quantities occurring in some sulfide ores. The metal is recovered as a byproduct in the smelting of lead and zinc ores in the United States, and from the roasting of pyrite for sulfuric acid production in Europe.

PHYSICAL AND CHEMICAL PROPERTIES

Metallic thallium and lead resemble each other closely in a number of physical properties. Pure thallium has a specific gravity close to that of lead and melts at almost the same temperature. It is slightly harder than sodium but softer than lead and may easily be cut with a knife or scratched with the fingernail. It has a metallic luster when freshly cut but rapidly dulls and tarnishes (*3*). The metal is malleable, has a crystalline structure, and, when bent, gives forth a sound like the "cry" of tin.

Thallium is most closely related to the alkali metals and the elements of group III of the periodic table in its chemical properties; hence, it is generally agreed that thallium should be placed there along with boron, aluminum, and a number of so-called rare elements (*3*).

Thallium has valences of 1 and 3; thus it forms both thallous and thallic compounds. The thallous salts are more stable and have been studied more closely than the thallic compounds.

Oxidation of thallium in the air results in the formation of yellow thallous oxide (Tl_2O), but under more highly oxidizing conditions, the dark brown or violet thallic oxide (Tl_2O_3) is produced (*10*). Thallium is slowly soluble in alcohol, insoluble in liquid ammonia, and combines directly with sulfur, phosphorus, and the halogens.

EXTRACTIVE METALLURGY

The preparation of pure metallic thallium is relatively simple, and numerous methods have been practiced for extracting thallium from ores, flue dust, and other industrial residues. The processes employed are based on the difference in solubility between thallium compounds and on the nature of the other elements present in the raw material. Thallium is being produced in the United States as a byproduct of cadmium operations, in the smelting of lead-zinc ores. In Europe it has been recovered as a byproduct from smelting operations; from the flue dusts and residues of sulfuric acid plants in which thallium-bearing ores are used; and as a byproduct of the lithopone industry.

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

An electrolytic process is one of the methods used by the American Smelting & Refining Co. for recovering thallium at its Globe cadmium refinery in Denver, Colo. After the sulfates of cadmium and thallium are electrolyzed, a mixture of these metals deposited on the cathodes is converted to a hydroxide and dissolved by treatment with hot water and steam. Cadmium is precipitated by addition of sodium carbonate, and the cadmium-free filtrate is treated with hydrogen sulfide to precipitate thallium sulfide. This precipitate is dissolved with sulfuric acid and boiled free of hydrogen sulfide, leaving a solution of thallous sulfate—the principal commercial product. If metallic thallium is desired, the sulfate solution is electrolyzed to yield the pure metal (*10*).

Numerous patents issued during the past decade, covering the extraction of thallium and cadmium from other metals, are minor variations of the above process.

The extraction of thallium from flue dust and sludges resulting from the oxidation of sulfide ores is described as follows: Thallium-bearing residues are dissolved with dilute sulfuric acid, and the solution is filtered and concentrated. Thallium is precipitated as a chloride, which is purified and converted to another compound by fusion with potassium cyanide and sodium carbonate. This compound is put into solution, and hydrogen sulfide is used to precipitate the thallium as a sulfide. This general process is still employed, particularly abroad (*10*).

PRODUCTION

World production statistics for thallium are not available, but requirements for thallium sulfate were estimated at 6,000 and 7,500 pounds in 1946 and 1947, respectively. In 1945 the United States imported 3,000 pounds of thallium from Belgium and France Canada reported its first production of 128 pounds of thallium in 1944 from accumulated thallium-rich residues of the base-metal operations of the Hudson Bay Mining & Smelting Co., Flin-Flon, Manitoba, Canada.

Thallium was extracted from crude white-arsenic Cottrell dusts during 1949 at the silver-lead smelters of American Smelting & Refining Co., Murray, Utah. Operations ceased at the smelter on October 1, 1949. The arsenical gold ores at Mercur, Utah, are rich in thallium and may constitute the largest domestic reserve. The owners of the old cyanide tailings dumps at Mercur and Manning, Utah, estimate the thallium content to exceed 4 million pounds, with over 10 times that quantity in the unbroken ores. The dust from the lead Cottrell precipitator at Murray, Utah, was reported to contain as high as 3 percent thallium.

Thallium sulfate is produced at an annual rate of about 1,000 pounds as a byproduct of base-metal smelting operations in Belgium by the Société des Mines et Fonderies de Zinc de la Vielle-Montagne, Liège, Belgium.

The Imperial Smelting Corp., Ltd., London, produces thallium as a byproduct of base-metal operations. Flue dusts are the chief commercial source, particularly those produced in the manufacture of lithopone from zinc or in the reclamation of scrap alloys. The dusts and volatilized contents are collected in electrostatic precipitators and have been found to have an average thallium content of up to 0.6 percent and in some cases as high as 9 percent.

The Cerro de Pasco Copper Corp. has reported the production of thallium at its Oroya, Peru, smelter.

Thallium is present in the base-metal ores of Mount Isa, Queensland, Australia. It has been reported that 1½ tons of thallium enters the smelter there each month but is not recovered because the demand is insufficient to warrant installation of extractive facilities.

USES

The principal commercial use for thallium is in the form of thallium sulfate, an extremely poisonous substance having neither smell nor taste. This compound is a very efficient poison, particularly for exterminating rodents, insects, and other pests (*10*). Thallium carbonate is also a good fungicidal agent because of its more soluble nature. The sale of thallium rodenticides to the public is generally forbidden because of its extreme toxicity to man and animals, for which there is no effective antidote.

Some new uses for thallium developed during World War II are based on the ability of thallium bromoiodide crystals to transmit infrared radiation of very long wavelength. Such crystals find important application in military equipment designed for detection, signaling, and similar operations where visible radiation must be absent (*3*). Bromoiodide crystals in the form of lenses, plates, and prisms are very important in the optical systems of spectrometers used for fundamental research in molecular structure and for the analysis of organic preparations by infrared absorption methods.

Thallium oxysulfide is more sensitive than selenium to light of long wavelength and low intensity and finds application in photoelectric cells (*10*). The electrical resistance of thallium drops 50 percent upon exposure to 0.25 foot-candle of light. Another important development is mercury-thallium (8.5 percent thallium), an alloy with a much lower freezing point than mercury alone, permitting temperature recording with glass-type thermometers in the range of +20° C. to −60° C.

Thallium also has limited use in high-density liquids, special glasses, selenium rectifiers, silver-bearing alloys, corrosion-resistant and fusible lead alloys, and mold- and insect-proofing and as a phosphor activator. Small quantities of thallium are consumed for military purposes as detectors in radiation tubes, optical crystals, mercury - thallium low - temperature switches, scintillation counters, and activated phosphors for cathode-ray tubes. Thallium salts have been used for protecting textiles, leather, and other products against attack by weather, fungus, and insects. Thallium, in metal or compound form, reportedly has potential use as an additive constituent in antiknock motor fuel for internal-combustion engines.

CURRENT PRICES

Thallium metal and sulfate have been offered for sale since September 1949 at $12.50 and $10.50 per pound, respectively, in 10-pound lots.

RESEARCH

Research by industry has been directed principally toward effective development of improved metallurgical procedures for extracting thallium as a byproduct of lead-zinc-cadmium operations.

Because of increased military requirements and expanded peacetime uses of thallium, the Salt Lake City Station of the Federal Bureau of Mines in 1947–49 conducted a metallurgical investigation of thallium in ores, concentrates, and products of the American Smelting & Refining Co. smelter at Murray, Utah. As a result of these tests, the Bureau recommended effective procedures for higher recovery of thallium. These procedures were subsequently adopted and are currently in use, with certain modifications, at the Globe cadmium refinery in Denver.

OUTLOOK

The supply of thallium exceeded demand during 1950–54, with a resulting accumulation of stocks. Current supply appears to be adequate to meet any foreseeable rise in requirements.

Snyder Mines, Inc., owner of the cyanide-tailing dumps at Mercur and Manning, Utah, carried out pilot-plant studies on its tailings, beginning in 1946, for recovery of thallium by sintering. These studies reportedly produced favorable results.

A Bureau of Mines survey of domestic thallium resources in 1946–47, made at the request of the Corps of Engineers, United States Army, showed the existence of relatively large quantities of thallium in the ores, processing liquors, and certain finished products of industries in the Salt Lake Valley area, Utah. The Bureau developed a method for detecting and determining thallium in various raw materials and products. As a result of the Bureau's survey, the American Smelting & Refining Co. constructed a pilot plant for recovering thallium from Cottrell dust at its Murray, Utah, silver-lead smelter. The same company subsequently constructed a new thallium-recovery plant in 1948 at its Globe cadmium refinery in Denver, Colo., with a monthly production capacity of 2,000 pounds of thallium.

The reserves of low-grade ores and materials which contain thallium, such as those of copper, lead, zinc, cadmium, iron pyrite, and white arsenic, are large. The thallium content of old cyanide tailings dumps, combined with the unbroken arsenical gold ores at Mercur, Utah, could supply enough thallium raw material to satisfy the current domestic annual production capacity for many years.

The major use of thallium metal in the form of thallium sulfate rodenticides is expected to be curtailed in future owing to replacement by other improved rodenticides, which are safer to human beings and less harmful to certain animal species.

Promising fields for expanded use are the electronic, phosphor, and corrosion-resistant and fusible alloy industries, with particular emphasis on military applications. Certain thallium compounds have potential industrial applications in special glasses, photoelectric cells, poisons, and insecticides, biological uses, catalysts, photographic emulsions, infrared light detection, pigments, and signal flares and rockets.

PROBLEMS

Thallium compounds have many potential industrial, scientific, and military applications, but the relatively high price for the element is a deterrent.

Metallic thallium as such has had few commercial applications and hence has been regarded as an intermediate form from which thallium alloys and compounds are produced. Research to develop a more complete and wider utilization of the properties of thallium metal, alloys, and compounds has been inadequate.

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THORIUM

By

John E. Crawford[1]

THORIUM may become an economical source of fissionable uranium-233 in the "breeder" reactor, producing more fissionable material than is burned in the reactor's fuel system. Such an accomplishment would make available new reserves of source material to supply man's nuclear energy requirements.

Summary

Most commercial thorium is obtained from the mineral monazite—a phosphate of rare earths containing less than 1 percent to 15 percent thoria (ThO_2). Because of the stringency of imported monazite supplies—principally from Brazil and India—United States consumers and others, including the Bureau of Mines, became active in developing known domestic sources and endeavoring to find new economic sources in the United States. The combined efforts of Government and industry have been successful in the Northwest and in the southeastern coastal regions.

Metallurgical processes produce a pure commercial thorium nitrate from the monazite concentrates. Thorium oxide is obtained by igniting the nitrate. There is only a very small production of thorium metal in the United States.

The principal nonenergy uses of thorium and its compounds are in manufacturing gas mantles, refractories, and polishing compounds, in producing various chemical and medical supplies, in the electrical industry in manufacturing lamp filaments and vacuum tubes, and recently in magnesium alloys to improve their casting quality.

Thorium may in due course be an outstanding source of energy in the breeder reactor, and the current importance attached to it as a possible source of fissionable material is well warranted.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

PROPERTIES

Thorium was discovered by Berzelius in 1829. While working with a black mineral, now known as thorite, from the island of Lovo, Norway, he came upon a new earth which he called "thoria," after the Norse god Thor.

Metallic thorium is soft, and similar to annealed copper in hardness, or, in very pure form, similar to lead. The commercial metal is produced as a powder, which is fabricated into solid metal by powder-metallurgy practice. Thorium has an atomic weight of 232.12 and an atomic number of 90. Thorium–232 is the principal natural isotope found in nature, but thorium–228 (radiothorium) and other disintegration products occur with it. The specific gravity of thorium is about 11.75. The metal melts at 1,827° C. and boils above 3,500° C.; it crystallizes in face-centered cubes. Metallic thorium is a stable metal at ordinary temperatures, being unaffected by water at 100° C. Oxidation of the metal proceeds quite rapidly in air at 200° C. or more. When thorium is heated in air it burns with great brilliancy, forming particles of the white oxide, thoria (ThO_2). Thorium is radioactive and the source of fissionable uranium–233 when bombarded with neutrons (*4, 7*).[2]

The metal can be readily hot- or cold-worked. It can be fabricated by forging, rolling, swaging, or extruding; machined like mild steel with or without cutting fluids; and spun. Thorium, however, is difficult to weld, brazing yields brittle joints, and it has very poor resistance to the atmosphere, water, and most reagents. Metallic thorium has been produced in the form of rod, sheet, thin-walled tube, fine wire, and foil.

OCCURRENCE

Thorium is present in at least 100 minerals in proportions ranging from traces up to about 70 percent. The element is not found in the native state, but its silicate is one of the most important of thorium minerals—thorite ($ThSiO_4$; Th, 25.2 to 62.7 percent). Most commercial thorium, however, is obtained from the mineral monazite—a phosphate of rare earths containing from less than 1 percent to 15 percent thoria and generally expressed by the chemical formula (Ce, La, Di) PO_4 or (Ce, La, Di)$_2O_3P_2O_5$, which does not indicate the thoria content or the minor constitutents, such as samarium, gadolinium, dysprosium, ytterbium, erbium, lutecium, terbium, holmium, thulium, and europium (*4, 10*).

Monazite occurs in the form of sand derived from the weathering of granites, gneisses, and pegmatites, in which it occurs as a primary mineral. Monazite in the host rock seldom exceeds 0.1 percent. The commercial sands are obtained from beach and dune deposits. Inland placers along rivers and streams have been worked. On the beaches wave and tide action has reconcentrated the stream and placer sands, and in the dune deposits selective wind action has concentrated the heavy sands into rich layers. The heavy sands commonly consist of ilmenite, zircon, rutile, and monazite in widely varying proportions. Quartz is the principal gangue mineral, although the sands in some places contain garnet, hornblende, andalusite, and enstatite (*8*).

The world's largest productive accumulations of monazite lie along the coastlines of the States of Travancore in India and Rio de Janeiro, Espirito Santo, and Bahía in Brazil. Other monazite-sand deposits occur along the west coast of Ceylon and at the mouth of the Nile River in Egypt. Granitic rocks have produced many monazite-bearing gold placers in Korea that have been worked over 50 years for their gold content, but the monazite was not saved. Similar but shallower beach deposits occur along the east Korean coast facing the Sea of Japan. Monazite is associated with cassiterite, fergusonite, euxenite, and gadolinite in Western Australia; sands containing fergusonite are mined for their radioactive content near Copley in the southern part of Western Australia. In the New England region of New South Wales monazite is associated with tin and tungsten, and in the Malay Peninsula the alluvial tin deposits contain varying quantities of monazite.

In 1950 hydrothermal vein deposits containing as much as 70 percent monazite and 5 percent copper were found in Cape Province, Union of South Africa (*10*).

In the United States monazite has been produced in North and South Carolina, Florida, and Idaho. The placer deposits in the Carolinas that have been worked for monazites are derived from gneisses, granites, and pegmatites that outcrop intermittently southeast of the Blue Ridge in a belt 200 miles or more long and 15 to 20 miles wide. In Idaho the monazite

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

placer deposits are found in the central part of the State along the western flank of the Idaho batholith. Present and ancient beach sands along and near the east coast of Florida have been mined for some years for their content of ilmenite, rutile, and zircon, and these sands also contain a small percentage of monazite, which evidently was transported to Florida by ancient streams that drained southeastward from the Carolinas, and by shore currents (*9, 14*).

Available data indicate a thoria content of about 5 to 10 percent for Indian monazite; 6 percent for Brazilian monazite; 8 percent for monazite from Ceylon; 4 to 5 percent for Australian monazite; and 3½ to 9 percent for monazite from the Malay Peninsula. Domestically produced monazite from the Carolinas contains 5 to 7 percent thoria; that from Idaho, 3½ to 5 percent; and that from Florida 4½ to 6 percent (*8, 9, 10, 14*).

METALLURGY

The processing of monazite sands for the recovery of thorium (*7, 11*) involves the following steps:

1. Fine grinding.
2. Solubilization of the thorium and rare-earth phosphate content with hot, concentrated sulfuric acid.
3. Fractional precipitation of the rare-earth and thorium salts.
4. Purification of thorium fractional precipitate with oxalic acid.
5. Digestion of thorium oxalate precipitate with sodium hydroxide and solubilization in hydrochloric acid.
6. Re-treatment of acid solution with sodium hydroxide to produce relatively pure thorium hydroxide.

There are three general processes for the purification of thorium hydroxide; these are (1) the oxalate method, (2) the carbonate method, and (3) the sulfate method.

After purification by any or combinations of the above methods the thorium is converted into hydroxide, then dissolved in nitric acid and evaporated until thorium nitrate crystallizes out. The product is sold as crystalline thorium nitrate, $Th(NO_3)_4 \cdot 4H_2O$. The oxide may be obtained by ignition of the nitrate.

PRODUCTION AND CONSUMPTION

PRODUCTION

World supplies of monazite have been obtained principally from Brazil and India and in minor quantities from the United States of Indonesia, Ceylon, and Australia. In 1946 the Indian Government placed an embargo on exports of monazite, and toward the close of 1950 similar action was taken by the Government of Brazil (*14*).

The United States was a significant producer of monazite over the period 1893–1910, but beginning with 1911, when Indian monazite became available, domestic output of the mineral virtually ceased. Not until 1948 was production resumed; by this time stocks of monazite in the United States had been substantially reduced as a result of the embargo action on the part of India and restricted shipments from Brazil over several years before similar embargo action by that country in late 1950. New sources of the mineral became imperative, if serious dislocation of the rare-earth industries in the United States was to be avoided. The status of thorium, however, was not precarious, since the monazite, processed primarily for the rare-earth content, yielded more thoria than was in demand. As a result, stocks of thoria were available.

Under the stringency of monazite supplies, the principal consumers and others became active in developing known domestic sources and in endeavoring to find new economic sources in the United States. Toward the close of 1948 the Bureau of Mines undertook an investigation and evaluation of the monazite resources of the United States. The combined efforts of industry and Government were successful, the Bureau making a major contribution. A small production was effected in 1948 and 1949, and by the close of 1951 enough domestic reserves of monazite had been determined to assure the self-sufficiency of the United States so far as current requirements for the rare earths and thorium were concerned.

For additional information on monazite, reference should be made to the chapter entitled "Rare Earths."

The principal domestic producers of thorium compounds (mostly nitrates and oxides) from monazite concentrates are as follows:

Lindsay Chemical Co.,
West Chicago, Ill.

Maywood Chemical Works,
Maywood, N. J.

Rare Earths, Inc.,
Paterson, N. J.

There is a relatively small production of thorium metal in the United States. The major producers are Westinghouse Electric Corp. (Lamp Division), Bloomfield, N. J., and Metal Hydrides, Inc., Beverly, Mass.

Monazite-production statistics are classified and therefore cannot be published. The mineral comes under the regulations of the Atomic Energy Act of 1954, inasmuch as it is a source of thorium from which fissionable material can be produced.

CONSUMPTION

Atomic Energy Commission authorizations for purchase of thorium in the United States for nonenergy purposes during 1950–54 are shown in the following table:

Atomic Energy Commission authorizations of thorium compounds for nonenergy purposes in the United States, 1950–54

[Pounds of contained ThO_2]

Industry	1950	1951	1952	1953	1954
Magnesium alloys	--------	--------	--------	3, 600	4, 647
Gas-mantle manufacture	48, 471	31, 132	25, 427	8, 707	9, 765
Refractories and polishing compounds	1, 889	3, 382	1, 157	236	24
Chemical and medical	2, 097	6, 246	11, 064	5, 179	3, 738
Electrical	314	1, 457	277	1, 222	2, 016
Total ThO_2	52, 771	42, 217	37, 925	18, 944	20, 190

Import and consumption data on monazite also come under the Atomic Energy Act of 1954 and are therefore not available.

PRICES

Thorium metal (chief impurities—calcium, about 0.05 percent; iron, about 0.05 percent; thorium oxide, about 1.0 to 1.5 percent) was quoted during 1954, f. o. b. plant, by one of the leading producers as follows:

	Price per gram for less than 200 grams per item	*Price per gram for 200 grams or more per item*
Powder	$0. 45	$0. 35
Unsintered bars	. 50	. 40
Sintered bars	. 65	. 50
Sheet, 0.005 in. and over	. 75	. 60
Sheet, 0.002 in. to 0.0049 in.	. 85	. 70

The principal thorium compounds were quoted at the close of 1954, in 100-pound lots, f. o. b. plant, by one of the leading producers as follows:

Thorium nitrate, mantle grade:	
1954 domestic price	$3.00 per pound
export price	3.35 per pound
Thorium oxide, 97 percent ThO_2; rare earth oxide content, 0.01 percent; particle size, 3 to 50 microns:	
1954 domestic price	$8.25 per pound
Thorium oxide, photographic-lens grade, 99 percent ThO_2; rare earth oxide content, 0.006 percent; particle size, 1 micron:	
1954 domestic price	$9.35 per pound

For 50- to 99-pound lots, add 10 percent to all prices.
For 10- to 49-pound lots, add 20 percent to all prices.
For 1- to 9-pound lots, add 50 percent to all prices.

USES

NONENERGY USES

As indicated in the preceding table, the principal nonenergy use of thorium continues to be in the manufacture of gas mantles, which are sold both domestically and for export. Gas mantles are still used for street lighting in some communities, and the well-known Aladdin kerosine lamps and Coleman gasoline lanterns are widely used in rural districts. Millions of these mantles were used by the Armed Forces in World War II. They are composed of about 99 percent thorium oxide and 1 percent cerium oxide (*4*).

Thoria is used as a catalyst for petroleum cracking and in industrial chemistry may be employed as a catalyst in oxidizing ammonia to nitric acid, carbon monoxide to water gas, and sulfur dioxide to sulfur trioxide and in making many organic reagents. A small quantity of high-purity thoria is used as a constituent of special optical glass. Thorium salts have little toxicity; they are astringent and tonic and can cure certain parasitic skin infections. Soluble thorium salts are used in creams and lotions. Thorium oxide forms a white, dense powder insoluble in water and in dilute acid and finds some use in dental powders and pastes (*8*).

Owing to its high melting point (over 2,800° C.) plus its stability, thoria is extensively used in refractories. It is also employed as an ingredient of polishing compounds (*7*).

In the electrical industries thorium oxide (0.8 to 1.2 percent) is incorporated into tungsten to control grain size of tungsten and to produce filaments having high resistance to mechanical shock. Such filaments are also an important source of primary electron emission. These are used in vacuum tubes because of their electron emissive power, which permits the tube to function; in the colloidal state thorium oxide is mixed with the usual "getter" employed in manufacturing such tubes, facilitating final evacuation and maintenance of vacuum in the tubes. In metallic sheet form thorium is used in special lamps, such as sun lamps, to aid in starting the lamps (*4,7*).

In 1953, as indicated in the table of consumption statistics, thorium was employed in magnesium alloys. The addition of thorium to magnesium alloys imparts outstanding mechanical properties at ambient and elevated temperatures. Alloys containing 2 to 3 percent thorium show good ductility and exceptional stability of properties after heating (*13*).

Radiothorium (thorium–228) is a strong emitter of alpha rays and because of its intense ionizing power can be used in dissipating static electricity. Mesothorium (radium–228, another disintegration product of thorium) is added in small quantities to self-luminous paints employed in instrument dials and for blackout uses (*4*).

ENERGY USES

The use of thorium as a possible source of energy depends on the fact that when thorium–232, the naturally occurring isotope of the element, is subjected to bombardment by neutrons it may be converted into fissionable uranium–233. Reactors possibly can be designed to produce uranium–233 from thorium in a quantity that exceeds the original quantity of uranium–233, uranium–235, or plutonium–239 consumed in supplying the stream of neutrons required for the conversion. This operation of creating an excess of nuclear fuel is known as "breeding" (*5*).

During the latter half of 1951 the Atomic Energy Commission completed construction of an experimental breeder reactor at the National Reactor Testing Station, Arco, Idaho. In July 1953 the AEC announced that, following an extended period of operation of the breeder reactor, analyses of representative samples from the fuel elements of the core and from the natural uranium blanket that surrounds the core indicated that the amount of new fissionable material (plutonium) produced in the reactor is at least equal—taking into account chemical separation losses—to the amount of uranium–235 burned. The significance of the breeding process is that, in theory, all of the world supply of minable uranium and thorium can be made to undergo fission (*1*).

OUTLOOK

Current and near-future domestic supplies of thorium would appear to be adequate to satisfy normal requirements for the metal. It is not known whether foreign countries, such as India and Brazil, might resume exportation of monazite or offer thorium and thorium compounds for export.

Although full-mobilization requirements for thorium for nonenergy uses of the metal probably exceed the quantity produced under current treatment of monazite, such requirements might be met by appropriately increasing the scale of domestic monazite-mining operations.

In its Sixteenth Semiannual Report, the Atomic Energy Commission announced that a homogeneous thorium reactor is in the planning stage. The thorium reactor will produce about 65,000 kilowatts of heat, of which some 16,000 kilowatts will be converted into electricity. The reactor will have a blanket of thorium from which uranium–233 will be produced. After completion of development and design data by the Oak Ridge National Laboratory, construction is expected to begin during the fiscal year 1957 and to be finished in the fiscal year 1959 (*2*).

As to requirements for thorium as a source of fissionable material, the AEC stated in 1955 that its position continues as presented in the press release of October 14, 1948, which reads as follows:

ATOMIC ENERGY COMMISSION DESCRIBES ITS PRESENT INTEREST IN THORIUM

Washington, D. C. October 14.—Prompted by an increasing number of inquiries currently being received from individuals and small companies interested in monazite, the principal ore from which the element thorium is obtained, the Atomic Energy Commission today outlined for public information the nature and extent of its present interest in thorium.

While a strict control of exports and a complete record of domestic movements of thorium-bearing materials, including monazite ore, is maintained by the Commission, pursuant to the Atomic Energy Act of 1946, the usefulness of thorium and its principal source in nature, monazite ore, so far as the Commission's program is concerned, is limited for the present time to research. Accordingly, the only thorium the Commission purchases is for experimental purposes, chiefly in the form of thorium salts, and the Commission has no purchase program for monazite ore as such.

Thorium has long been recognized as a potential source of fissionable material. Consequently, like uranium, it was placed under the control of the Atomic Energy Commission by the Atomic Energy Act of 1946. Solving the many complex research problems in the way of the large-scale utilization of thorium may take a decade or two. The possibility, however, cannot be excluded that the time may be shorter. When the research problems are solved, thorium may become of great significance in the atomic energy program. Until these problems have been solved, however, thorium will not be in large demand for use in nuclear reactors.

Inquiries to the Commission regarding monazite ore have increased since April 1948, when the Commission publicly classified uranium-233 as a fissionable material pursuant to the Atomic Energy Act of 1946, since it is technically possible to derive uranium-233 from thorium in a manner similar to the way in which plutonium is derived from uranium. Current demand for monazite ore arises principally from its established uses outside of atomic energy since it is the only commercial source[3] of the element cerium and other rare earth materials which are essential to many industries.

PROBLEMS

From the standpoint of the utilization of thorium for the production of power the breeding process is an important problem that confronts the Atomic Energy Commission and organizations interested in developing peaceful uses for nuclear energy.

Should thorium find uses in the nuclear energy field or should it find important nonenergy applications, through research and development, there are a number of significant attendant problems.

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[3] Since issuance of the news release the development of large bastnaesite ore bodies in California has provided another commercial source of cerium (see Rare Earths chapter in this volume).

Among the more important are:

(1) A thorough determination of domestic resources of monazite, thorite, and other thorium-bearing minerals.

(2) Determination of the possibility of obtaining thorium-bearing ores and concentrates, finished thorium compounds, or thorium metal from foreign sources.

(3) A study of the necessity for stockpiling thorium ores, thorium compounds, or thorium metal to meet emergency requirements.

(4) Development of refining capacity to produce adequate amounts of thorium compounds, if necessary.

(5) Research and development to find economical ways of producing large amounts of thorium metal for nuclear reactors.

(6) Actual construction of thorium metal-production facilities.

If industrial and/or atomic energy requirements for thorium increase markedly, processing of monazite ores for their thorium content could supply enough byproduct rare earths to compete significantly with the bastnaesite deposits now being worked for their rare-earth content in California.

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TIN

By

Abbott Renick [1] and John B. Umhau [2]

NOTHING is more common in the United States than "tin" cans nor more scarce than tin mines. Furthermore, although this country consumes half of the primary world production, importing concentrate and metal to the extent of $200 million a year, American industry exercises but little direct control over tin mining anywhere in the world.

Summary

Tin is an essential component in the manufacture of tinplate and a prime ingredient of solder, bronze, babbitt, other alloys, collapsible tubes and foil, and numerous chemicals. The principal uses of tin have been established for many years, and there is no major substitute for tin. Nevertheless, more or less effective and practical conservation methods have been developed. The most important step in reducing tin consumption in the United States has been progressive displacement of the hot-dip method of tinplate manufacturing by the electrolytic process during the last decade.

In normal years United States imports of tin represent about 50 percent of the Free-World production. World smelter production of tin in 1953 totaled 180,000 long tons.[3] Consumption of primary tin in the world in 1953 is estimated at 131,500 tons, of which the United States used about 54,000. The principal tin-mining countries are, in order of magnitude, Malaya, Indonesia, Bolivia, Belgian Congo, Thailand, and Nigeria. Southeast Asia supplies almost 60 percent of total Free-World mine production of tin. Tin resources in that and other areas appear adequate to sustain the expected increase in the demand for new tin in the foreseeable future. The principal consuming countries are the United States, the United Kingdom, France, Germany, and Canada. The largest tin smelter in the world is at Texas City, Tex., but the combined production of the smelters at Penang and Singapore, treating the bulk of Malayan concentrate and some from Thailand, establishes these smelters as the chief world source of tin.

Free-World reserves are estimated at 5 million tons of metal, or about 30 times the 1950 rate of production. Numerous occurrences of tin ores in the United States have been exhaustively studied, yet the known deposits are either too small or too low grade to make a significant contribution to national supply.

Major problems facing the tin industry are political instability, the competition of materials that substitute for tin, and the fluctuations in demand arising from Government purchasing (for stockpiles or other purposes) and the restrictions imposed on consumption in times of stress.

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[1] Commodity-industry analyst, Bureau of Mines.

[2] Statistical assistant, Bureau of Mines.

[3] All tonnages stated herein with reference to tin are in long tons following commercial practice.

BACKGROUND

SIZE AND WORLD DISTRIBUTION OF TIN INDUSTRY

The world mine production of tin in 1953 was 176,000 tons valued at almost $400 million. The United Kingdom has by far the most influential position in tin production and trade. In addition to the very small output in the United Kingdom itself, the British Empire includes Malaya, Nigeria, Australia, Union of South Africa, and Canada, which together produced 38 percent of the world's tin in 1953. Tin smelting is even more concentrated in the British Empire than is mine production. Virtually no British concentrates are exported; but large imports are received, principally from Bolivia.

Tin is not mined in substantial quantities in any important industrial country, including the United States, and therefore must be imported by all of them. The total recorded mine production of tin in the United States from 1910 to 1953 is 1,950 tons, a quantity insufficient to meet industrial requirements for 2 weeks. Alaska, where tin was discovered in 1900, has furnished about 90 percent of the United States total. South Dakota, the Appalachians, California, Nevada, New Mexico, and elsewhere supplied the remaining 10 percent. The largest production for any single year was 168 tons in 1937.

Tables 1, 2, and 3 show world mine production, 1940–53, world smelter production, and world consumption in 1953, respectively.

HISTORY OF TIN INDUSTRY

Tin is one of the few metals used by prehistoric man and has been well established in

TABLE 1.—*World mine production of tin, 1940–53* [1]

[Long tons]

	European war only, 1940–41 annual average	Pacific war, 1942–45 annual average	Last war year, 1945	Postwar 7-year average, 1946–52	1953	Percent of world total—1953
Southeast Asia:						
Malaya	[2] 81, 200	[2] 13, 552	3, 152	43, 818	56, 254	32
Indonesia	47, 352	9, 260	1, 050	25, 728	33, 822	19
Thailand	16, 471	4, 686	1, 775	6, 265	10, 126	6
Burma	5, 052	[2] 525	[2] 400	1, 330	[2] 960	1
Indochina	1, 383	534	95	54	264	----------
Subtotal	151, 458	28, 557	6, 472	77, 195	101, 426	58
Other major producers:						
Bolivia [3]	39, 995	39, 956	42, 490	34, 092	34, 639	19
Belgian Congo [4]	14, 001	16, 682	17, 049	13, 457	15, 293	9
Nigeria	12, 037	12, 200	11, 224	8, 948	8, 222	5
Subtotal	66, 033	68, 838	70, 763	56, 497	58, 154	33
Other minor producers:						
China [2]	8, 750	3, 375	2, 000	4, 314	6, 300	3
Australia	3, 705	2, 607	2, 279	1, 908	1, 553	1
United Kingdom	1, 564	1, 291	1, 152	[5] 869	1, 103	1
Portugal	2, 164	1, 810	582	904	[2] 1, 520	1
Other countries	6, 661	5, 261	4, 513	3, 734	5, 734	3
Subtotal	22, 844	14, 344	10, 526	11, 729	16, 210	9
World mine production [6]	240, 500	111, 500	87, 500	145, 500	176, 000	100

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[1] From report by F. Stuart Miller, Tin Production and Resources, and International Tin Study Group, Statistical Bulletin, August 1954.

[2] Estimated.

[3] Exports; production figures not available.

[4] Includes production of Ruanda-Urundi.

[5] Intake by smelters, 1946–47; actual production, 1948–53.

[6] Partly estimated; excludes U. S. S. R.

industry from very early times. Ancient production amounted to a very substantial tonnage, and the development of modern industrial society has required an enormous increase in output. Tin is a constituent of bronze, which has been known from remote antiquity and was employed before the art of working in iron was discovered for purposes offensive and defensive, ornamental, and domestic; in fact, it filled the place held by iron today. Some tin was obtained from the Cassiterides Islands, off the west coast of Europe, which has given its name to the principal ore of tin—cassiterite. This tin trade was ended by the fall of Carthage, so that coins minted after this period contain increasing quantities of lead to replace the tin. In pewter, tin rendered an important although lowly service in former ages when metals were less abundant than now. Tin was used for tinning copper at the beginning of the Christian era, and tinplated iron was first manufactured in the 16th century.

TABLE 2.—*World smelter production of tin, by countries, 1953* [1]

Country:	*Production (long tons)*	*Percentage*
Malaya	62, 410	35
United States	37, 562	21
United Kingdom	28, 860	16
Netherlands	26, 950	15
Belgium	9, 039	5
China [2]	6, 300	4
Belgian Congo	2, 715	1
Others	6, 164	3
Total world	180, 000	100

[1] International Tin Study Group, Statistical Bulletin, August 1954.
[2] Estimated.

Production of tin before 1800, unlike that of most base metals, was relatively large compared with that since 1800. Estimates placed the production of Cornwall from 500 B. C. to 1800 A. D. at over 2½ million tons; and that of Malaya from 1400 to 1800 at approximately 1 million tons. Indonesia, China, and Continental Europe also produced quantities of tin before 1800. World mine production of tin since 1800 has totaled over 10 million tons. Modern society has required an enormous increase in tin production, with a rapid advance since 1875 as an outgrowth of industrialization.

At the beginning of the 19th century tin was used chiefly in bronze for cannons and bells, in pewterware, as a lining for copper cooking utensils, and to some extent as tinplate for roofing purposes. Some outstanding developments that have contributed to the increased demand for tin since 1800 were the adoption of the steam engine and power-driven machinery, which required bronze and soft-metal bearings; the development of tin-plated containers for food and other products needed to supply the rapidly increasing urban population resulting from industrialization; electrical power machinery and internal-combustion engines; automobiles; and airplanes. Many of these uses for tin, although developed before 1900, have expanded rapidly during the 20th century.

TABLE 3.—*World consumption of tin, by major consuming countries (except U. S. S. R.), 1953* [1]

Country:	*Consumption (long tons)*	*Percentage*
Africa:		
Union of South Africa	1, 520	1
North America:		
Canada	3, 950	3
United States	53, 961	41
South America:		
Argentina	1, 400	1
Brazil	1, 580	1
Asia:		
China	1, 900	1
India	2, 507	2
Japan	4, 650	4
Europe:		
Belgium-Luxembourg	1, 164	1
Czechoslovakia	1, 700	1
Denmark	2, 650	2
East Germany	1, 500	1
France	7, 300	6
Italy	2, 800	2
Netherlands	4, 330	3
Poland	1, 800	1
United Kingdom	18, 634	14
West Germany	5, 814	5
Oceania: Australia	2, 300	2
Other countries	10, 040	8
Total world [2]	131, 500	100

[1] International Tin Study Group, Statistical Bulletin, August 1954.
[2] Actual consumption figures for countries covering about 70 percent of world consumption; in other cases, consumption is apparent or estimated.

Tin always has been an international metal. The Phoenicians were attracted to Cornwall for tin over 20 centuries ago. Some of these traders settled there and engaged in mining and smelting of tin from streams; vein mining was developed later. Subsequently, during the Roman occupation, traffic in tin was established by overland transportation through France to Marseilles. The maximum production of Cornish tin was attained between 1871 and 1875; thereafter it declined slowly as production costs increased, until now it is only a small portion of the world total.

Asia has been the outstanding source of the world's tin supply due to development of the tin deposits of the Malay Peninsula and Archipelago. Malaya is the principal tin-producing country of this area and of the entire world and has been productive for over 2,000 years. China has also been an important producer for centuries. Long before the Netherlanders obtained control in Indonesia small quantities of tin had been mined by the natives. South America did not become a tin-producing continent until about 1860, when Bolivia began producing. Very little tin was produced in Africa up to 1910; since then, however, Belgian Congo and Nigeria have become important tin-producing countries. Australia began producing tin about the middle of the 19th century and gradually increased its output to nearly 25 percent of the world total from 1876 to 1880, but during the past 5 years its production amounted to less than 1 percent of the total. North America, the principal world consumer of tin, produces virtually none. Mexico has not been an important source of tin, although small-scale mining has been carried on since the Aztecs. There are no commercial bodies of tin ore in Canada, but since 1941 a few hundred tons a year has been derived from the small cassiterite content of the lead-zinc-silver ore of the Sullivan mine in British Columbia. Tin occurrences have been known in the United States for nearly 125 years, but efforts to mine it since early days of American mining have resulted in failure. Even very high prices could not bring out important quantities.

PROPERTIES

Tin is one of the most useful metals because of its fusibility, malleability, resistance to corrosion, readiness to alloy with many other metals, and attractive, silvery white color. Tin is affected little by air at ordinary temperature, even in the presence of moisture; consequently, its major use has been as tinplate for canning. The layer of tin protects the iron or steel from oxidation. In speaking of fabricated articles "tin" is often incorrectly used when tinplate (thin sheet steel coated with tin) is meant—for example, "a tin can." To distinguish, articles (such as condenser coils) actually made of solid tin are said to be made of "block tin." The metal of commerce is about 99.8 percent pure.

Tin's atomic weight is 118.70 and its atomic number 50; specific gravity, 7.29; melting point, 231.9° C.; boiling point, 2,260° C.; and valence, 2 of 4. The specific heat between 18° and 100° C. is 0.0556 and the latent heat of fusion 14.2 gram-calories per gram. The coefficient of linear expansion between 0° and 100° C. is 0.000023 and the electrical resistivity at 20° C. is 11.5 microhms per centimeter cube.

Tin flows readily under pressure and may easily be rolled into foil or extruded into tubes; although it is not easily drawn into fine wire, it can be rolled into that form. A pound of tin can be rolled into foil having an area of 11,000 to 14,000 square inches. Owing to its low temperature of recrystallization, tin may be severely cold-worked without the necessity for annealing.

ORE MINERALS

Tin is one of the scarce metals, and its ores are not numerous. Cassiterite is the only mineral ("tin ore") of commercial importance as the world's source of tin. Some tin is recovered from stannite (Cu_2FeSnS_4), teallite ($PbSnS_2$), cylindrite ($Pb_3FeSn_4Sb_2S_{14}$), franckeite ($Pb_5FeSn_3Sb_2S_{14}$), and other rare tin-bearing minerals when these are associated with cassiterite. Native tin is occasionally found but never in more than trifling quantities. Cassiterite is tin oxide (SnO_2) and when chemically pure has a metallic content of 78.6 percent tin. Usually the mineral is deep brown or black, but it may be white, green, or red. Its hardness is 6 to 7; and its specific gravity (7), aids greatly in concentration of the ore. The streak is white or nearly so. It is notably friable, which creates a problem in ore dressing owing to its tendency to break down into fine particles that are not readily recovered.

OCCURRENCES

The original home of cassiterite is in granites, pegmatite dikes, or quartz veins. All tin has been obtained from ore bodies that were formed originally in or near the neighborhood of rocks of the granite family (magmatic quartz veins, pegmatites, aplites, rhyolites, and quartz porphyries). In weathering and erosion cassiterite, being heavy and resistant, is concentrated in placers in various sizes but usually as sand with various sizes of grain, from pieces the size of a pea down to the finest flour, which may be more or less bonded by clayish material (usually there is no coarse gravel to handle) and covered with overburden. Cassiterite concentrate obtained from placers is purer than that from veins, for it has undergone a natural process of concentration that has eliminated certain heavy minerals that occur associated with it in veins. The chief tin fields are in southeast Asia, Bolivia, and Africa. The leading producers of southeast Asia are Malaya, Indonesia, Siam, Burma, China, and Indochina. In Africa Belgian Congo and Nigeria lead.

The known deposits in the United States are either too small or too low grade to permit profitable operation or to afford a significant national resource. These scanty domestic occurrences have been exhaustively studied; and numerous attempts have been made, at great cost, to exploit certain of the deposits commercially.

The Bureau of Mines and Geological Survey have been studying the tin resources of the United States for many years in course of their investigations of domestic mineral resources. From 1939 to 1945, as a part of the strategic minerals investigations instituted in connection with the World War II, they gave special attention to the problem of tin; during that period an intensive study was made of all the deposits in the United States that were believed to have even a remote chance of containing tin reserves. Nearly all of the information resulting from these investigations is negative. Accordingly, it emphasizes the fact that the United States depends upon foreign countries for virtually all of its tin supply. Nevertheless, the data derived from the wartime and prewar studies constitute an important and valuable part of the national record, and most of them have been published in one form or another.

During World War II the Geological Survey made some 2,000 spectrographic analyses of material from prospects, mines, mine dumps, and smelter slag dumps. These samples were from virtually every State and Territory. No new sources of tin of economic significance were found.

From 1939 through 1943 the Bureau of Mines examined 85 tin occurrences, and projects were run on 12 deposits, 7 of which indicated some tonnage with enough tin content so that it conceivably could be recovered. The total indicated tin content of these deposits is insignificant in terms of national requirements.

The placer-tin deposits near York, Seward Peninsula, Alaska, have been known for about 50 years and have yielded a small output almost every year. The total exceeds that of all other domestic districts combined. From what is known concerning the geology of the region and the extent of tin-bearing gravels, the district clearly presents the best opportunity in the United States or its Territories to recover the most tin for a given expenditure, although it seems that the maximum yield cannot be more than a small part of domestic needs.

PRINCIPAL DEPOSITS

MALAYA

The principal known tin deposits are as follows (*12*): [4]

The tin-ore deposits of Malaya are associated with the granite and therefore occur generally in a zone adjacent to the main range and subsidiary granite ranges. This zone runs on the western side of Malaya from Perlis in the north to Malacca and North Johore in the south.

Before the invasion in 1941 there were 962 mines working; in 1946 there were 216 mines working. This compares with 740 mines operating in December 1951 and 629 in December 1953.

Nearly 90 percent of the total Malayan production of tin in 1953 was obtained by dredging (50.9 percent) and gravel pumping (38.7 percent). The percentages from other methods of mining were hydraulicking, 2.1 percent; opencast mining, 2.3 percent; underground mining, 4.0 percent; dulang washing, 1.7 percent; and miscellaneous, 0.3 percent.

INDONESIA

Tin in Indonesia is found mainly in the three islands of Bangka, Billiton, and Singkep, between Sumatra and Borneo. In addition, small deposits occur also around Bangkinang on the west coast of Sumatra and on the islands of Karimun and Kundur; there are few possibilities on most of the other islands of the Riouw Archipelago.

Geologically, the Indonesian tin-ore deposits are of the same kind as those in the other tin fields of the Far East. Primary tin deposits occur in quartz veins, granite, and greisen on Bangka (at Pemali, Balei, Bandung, and Sambonggiri) and on Billiton (where the Klappa Kampit lode mine was producing about 2,000 tons of tin a year before World War II). More important, however, are the large eluvial and alluvial deposits. Alluvial deposits are found mainly on the bottom of old river valleys; in several places deposits along the present coastline under the sea are being worked.

Over two-thirds of the production now comes from dredges and under one-third from gravel pumps and hydraulicking. The concentrates, averaging about 72½ percent tin, produced by dredges and mines in Indonesia, are shipped to the smelters.

The exports of Indonesian concentrates (about two-thirds of the total output) went almost entirely to the Malayan smelters in the late 1920's and to the Netherlands in the 1930's for treatment in the Arnhem smelter. Since 1946 the destinations have been the United States (roughly one-third) and the Netherlands (about two-thirds).

BOLIVIA

Bolivia is one of the three principal tin-producing areas of the world and depends for financial stability almost wholly on the tin industry, which is not only the chief contributor to the national budget but also the only significant source of foreign exchange.

The tin deposits occur in a belt about 500 miles long and 60 miles broad, extending from Lake Titicaca in the north to the Argentine border in the south and roughly corresponding

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[4] Italicized numbers in parentheses refer to items in the bibliography at the end of this chapter.

to the mountain ranges of the Cordillera Real of the Andes.

The tin mines are at altitudes ranging from 13,000 to 17,000 feet above sea level. The bulk of the production comes from lode mines, some of which are very deep. The tin ores have a relatively low metal content and are generally complex in composition. Concentration is difficult, recovery is relatively low, and the concentrates obtained are often mixed with impurities (sulfide minerals).

BELGIAN CONGO

The mining of tin in the Congo and Ruanda-Urundi is relatively recent, and these territories have ranked as important producers only since the second half of the 1930's.

Production in the Congo is mainly from eluvial or alluvial deposits, although there is some small-scale lode mining by a number of companies all over the country, especially in Katanga and Ruanda. Mining is usually conducted by opencast methods. There are no dredges. Mechanical excavators (electric or diesel shovels or draglines) or hydraulic mining is used wherever suitable. The concentrates obtained after washing and electromagnetic separation are quite pure (averaging 68 to 76 percent tin) and normally contain no arsenic, lead or antimony.

THAILAND

Thailand is one of the more important tin-producing countries. During the 1930's it became the fourth largest producer and at present ranks as the fifth largest tin-producing country in the world. In 1940—the year of greatest output—over 17,000 tons of tin content was produced. Production declined markedly during World War II but recovered to over 10,000 tons in 1950 and again in 1953. Placer deposits furnish over 90 percent of all the tin produced. About half the tin produced is recovered by dredging, and about one-third comes from gravel pump mines.

NIGERIA

The main deposits of cassiterite in Nigeria lie in the plateau area (Jos and Bauchi districts) about 3,000 feet above sea level, 300 to 400 miles from the coast. Broken down by erosion, the primary deposits became alluvial deposits, lying either in the more recent river beds or in the ancient ones, which are covered by volcanic layers. The geologically modern placers are progressively being depleted. The grade of ground, which in 1939 was still 2.26 pounds of cassiterite per cubic yard, was only 1.01 pound in 1948 and 0.71 pound in 1952–53.

In 1953 tin production was in the hands of 51 companies (mainly registered in the United Kingdom) and 88 private owners (mainly Europeans in Nigeria). Only 5 of the companies had an output of over 300 tons a year; about 40 percent of the total output comes from the Amalgamated Tin Mines of Nigeria.

MINING AND CONCENTRATION

Bucketline dredging is the most important method used in recovering cassiterite at placer deposits. Although complete data for world operations are not available, approximately 150 dredges were recovering tin at the end of 1950. When the dislocations due to war have been overcome, the number of operating dredges probably will increase somewhat; but, because of the ever-increasing size and efficiency of the units, it is unlikely that the number will approach the prewar level of nearly 200. Bucketline dredges consist of a large pontoon on which are mounted a power-driven bucketline for excavating gravel from the floor and edge of the pond in which the dredge floats; a trommel for distintegrating and screening the gravel into a coarse and a fine product; riffled sluices for recovering the heavy minerals, including the cassiterite, from the fine product while rejecting the lighter waste materials; and usually jigs for aiding recovery of the cassiterite. Thus the dredge digs its way forward, passing all the gravel in its path through its recovery apparatus. One type of bucketline dredge peculiar to tin mining is the sea dredge employed for offshore deposits in Indonesia. Suction dredges have also played a part in tin recovery, but their importance is overshadowed by the bucketline type.

In addition to bucketline dredging a number of other mechanized and hand methods for working placers are employed. In gravel pumping the tin ore is pumped from a pit to a washing plant outside. Where the terrain has enough relief to permit it, hydraulicking may be practical, and where patches of higher grade gravel are found that are too small for the economic use of mechanical methods hand methods are used. In the hydraulic method a jet of water is directed against a bank of gravel to wash it into sluices and recover the cassiterite. Because of the very low income standards in some of the areas where placer tin occurs, hand mining is practicable, such as "dulang" washing by women in Malaya. These workers recover the tin mineral in a dulang or pan, which they hold in their hands.

The recovery of tin ore from lode mines does not differ essentially from the mining of other nonferrous metals. Of particular interest are the mines of Cornwall. Mining methods developed there and mining terms used by the Cornish miners have been carried to all parts of the English-speaking world, as well as to other areas.

The cassiterite concentrates made on dredges and in the sluiceboxes of other placer methods ordinarily are of relatively high grade. Before shipment, however, these concentrates are given a further cleaning, hand methods frequently being used to eliminate virtually all of the heavy minerals other than cassiterite. Pure cassiterite contains 78.6 percent tin, and regular shipments of concentrates containing 75 percent tin metal are not uncommon.

The recovery of tin and the preparation of high-grade concentrates from the ores of lode mines are much more difficult. Concentration depends largely upon ore dressing based on the high specific gravity of cassiterite compared with that of most of the associated minerals. Flotation, which has been so important in the concentration of copper, lead, zinc, and many other minerals, has made little headway at tin mills. Hydrometallurgical processes have not paralleled the development of cyanidation as applied to gold and silver ores. Recoveries in general are low, and tremendous increases in efficiency are possible if technological problems of treating the raw product of lode mines can be solved.

SMELTING

In general, pure SnO_2 is easily reduced at a low red heat by carbon or reducing gases, such as hydrogen, carbon monoxide, or hydrocarbons. If pure cassiterite could be obtained the reduction on a commercial scale would be easy and inexpensive. It is advisable to purify the concentrate as much as possible before smelting for thereby higher direct recoveries are obtained. SnO_2 readily enters silicate slag as tin silicate and also with basic slags may form stannates. Furthermore, metallic tin readily forms an alloy with iron which requires oxidation to decompose. Since tin ores usually carry SiO_2 and oxides of iron it is evident that losses from both causes will take place. The slag used in tin smelting are lime-iron silicates and temperatures of 1,100° to 1,200° C. are required to make them fusible. At these temperatures considerable volatilization of tin takes place, and modern plants have fume-catching devices. In both the crudest and the most modern plants it is general practice to smelt first for quick recovery of most of the tin and resmelt the slag for further recovery. Unless the original ore is exceptionally pure, the tin produced in both the first and second smeltings contains impurities. Most of them can be removed by drossing, but electrolytic refining is sometimes practiced.

Tin ore is reduced by carbon either in a blast furnace or a reverberatory furnace. The former is adapted to pure stream tin where little or no slag is to be made and where charcoal can be obtained cheaply and in adequate quantity. The reverberatory furnace is the type used in most modern plants.

REFINING

All tin except that produced from exceptionally pure stream tin is contaminated with more or less iron, arsenic, copper, and other elements. It therefore must generally be refined before it is ready for market.

As the chief impurity is iron (which forms a highly infusible alloy with tin) the usual refining operation consists of heating the bars (obtained in smelting) on an inclined hearth to a temperature only slightly above the melting point of pure tin. Comparatively pure tin melts out first, leaving a shell of the iron-tin alloy. While still melted the metal is "poled", that is, stirred with a green pole, applewood being preferred traditionally. Poling tosses the molten metal, bringing it into contact with the air. Tin oxide and most of the impurities oxidized by the poling operation collect in a tough film on the surface. The metal is allowed to settle at a temperature above its freezing point and then ladled out. The top metal is purest. Each succeeding lower stratum contains a larger proportion of impurities, the lowest often consisting of a solid or semisolid alloy of tin and iron.

Electrolytic refining is based upon the position of tin in the electromotive series of the metals, where lead is the only metal that stands equal or close to it. The tin electrolyte must contain a radical that will form an insoluble compound with lead, such as sulfate, chromate, or fluoride. Arsenic, antimony, bismuth, copper, and the other metals that may be present as impurities in tin are not dissolved in this process and remain in the anode slime. A hydrofluosilicic acid bath is used, which consists essentially of H_2SiF_6 with enough sulfuric acid to prevent lead from depositing by precipitating as lead sulfate. This process has been covered by a series of patents.

ROASTING

Some concentrates are pure enough to make roasting unnecessary. Others contain sulfur, arsenic, or antimony, which makes roasting desirable. The sulfur is eliminated as sulfur dioxide and a considerable part of the arsenic and antimony driven off as volatile oxides. At times salt is added, which assists in removal of some constituents as volatile chlorides and renders others water soluble and subject to removal by leaching. If tungsten is high it may be removed by sintering the ore with soda ash forming sodium tungstate, which may be leached out. Tin sulfide is volatilized in smelting operations, so the removal of sulfur is important.

A chemical method for treating tin ore—the Cavaet process (chloridization and volatilization and electrolysis of tin)—has been developed. In this process the tin ore is ground to 8-mesh and dried and roasted in a rotary kiln at 800° C. with charcoal and calcium chloride. Cassiterite is reduced by the carbon and combines with chlorine to form volatile stannous chloride, which is easily dissolved in water at the lower end of the kiln. When the stannous chloride solution has built up to suitable strength and other harmful chlorides, such as antimony, arsenic, and copper, have been precipitated, the solution is electrolyzed and the metallic tin plated out of the solution. The tin is then stripped from the cathodes and cast into ingots.

TEXAS CITY PLANT

The Texas City, Tex., plant has been valued at over $13 million, and the entire wartime activities of the Reconstruction Finance Corporation were written off as of June 1947. The capacity of the smelter exceeds 40,000 tons of fine tin a year but varies with the tin content of concentrates treated, so that if only alluvial concentrates were smelted the annual furnace capacity would be about 90,000 tons of fine tin.

Ore and concentrates are delivered to the smelter from the Galveston or Texas City dock by rail; the concentrates are shipped in burlap sacks. The weight of the sacks will range from about 65 pounds to 150. The sacks are then slit and the concentrates emptied into a sample mill bin. The ores are blended on a mixing floor into various lots for smelting (salt and ground coke being added) and then roasted in a rotary kiln to reduce the iron to the ferrous state and chloridize the lead and silver. The roasted ore is leached in spherical tanks with acidproof lining in a solution of hydrochloric acid at about 105° C. (220° F.) to remove iron and other undesirable impurities before smelting. When leaching is complete a filter screen is placed over the manhole, the tank is inverted, and air pressure is applied until the filter cake is blown fairly dry. The leached residue is smelted in separate lots in eight reverberatory furnaces. About 85 percent of the tin is recovered as crude bullion, and the remainder is retained in a low-silica slag. The slag from the primary smelting furnace is resmelted in a large furnace to recover the tin as an iron-tin alloy known as "hardhead." The "hardhead" is granulated and returned to the primary furnaces for resmelting.

The crude bullion is drossed and held at a temperature slightly above the melting point to permit formation of crystals of the metallic compound $FeSn_2$. These crystals are removed by filtering the hot metal through a porous tile metal filter, giving a bullion containing only about 0.001 percent impurities. The arsenic, antimony, and bismuth from impure ores are removed as a dross by successive additions of metallic aluminum, molten sulfur, and sodium hydroxide. Metal resulting from smelting alluvial ores and the purest Bolivian ores does not require aluminum drossing. The refined bullion is stored in holding kettles for blending before it is removed and cast into pigs.

BYPRODUCTS AND COPRODUCTS

Byproducts from both placer and vein mines include tungsten, columbite, tantalum, and ilmenite. Byproduct recovery of tin at the Climax molybdenum mines, Colorado, the lithium-spodumene, Kings Mountain, N. C., the Sullivan lead-zinc mine, British Columbia, and other mines is a negligible factor in world production.

SECONDARY SOURCES AND RECOVERY

"Secondary tin" is reclaimed from tinplate and other scrap. The annual yield has equaled about a third of the domestic tin consumption. About one-third of this secondary tin is derived from reworking industrial scrap resulting from fabrication of primary tin metal, and two-thirds is recovered from wornout and discarded articles. Of the total secondary tin about 10 percent is recovered as metallic tin—mostly by detinning plants treating clean tinplate clippings. The remainder has been in the form of alloys (87 percent) and chemicals (3 percent). A good portion of the use of tin is dissipative and would not add to the reservoir of metal in use that could be drawn upon should imports cease. Over 85 percent of the tin used for making tinplate is not recovered, and substantial tonnages of tin consumed for tinning and chemicals are not amenable to recovery.

Many methods for recovering tin from waste tinplate have been devised, and several detinning plants are operated as successful enterprises. Essentially, the process consists of treating the tinplate clippings under controlled conditions of temperature and concentration of reagents in an alkaline bath, to which a suitable oxidizing agent is added. This operation is conducted in open reaction tanks, and the tin is removed in a solution containing sodium stan-

TABLE 4.—*Secondary tin recovered in the United States, 1944–48 (average) and 1949–53*

[Long tons]

	Tin recovered at detinning plants			Tin recovered from all sources			
	As metal	In chemicals	Total	As metal	In alloys and chemicals	Total	
						Long tons	Value
1944–48 (average)	2, 926	348	3, 274	3, 140	24, 640	27, 780	$41, 456, 172
1949	2, 850	410	3, 260	3, 170	19, 060	22, 230	49, 461, 354
1950	3, 300	575	3, 875	3, 615	28, 065	31, 680	67, 809, 158
1951	3, 150	415	3, 565	3, 300	27, 445	30, 745	88, 363, 153
1952	2, 640	310	2, 950	2, 860	25, 940	28, 800	77, 710, 297
1953	2, 650	450	3, 100	2, 850	24, 750	27, 600	59, 185, 860

nate. The liquor passes to a thickener, whence the caustic soda solution of sodium stannate, after removal of the lead, is delivered to a battery of electrolytic cells in which the tin is deposited on steel cathodes. Tin recovered from these cathodes is chemically and metallurgically pure, generally as high as 99.99 percent, and is entirely suitable for any industrial use, including the major one of tinplate making.

Detinning plants also recover small tonnages of tin from used tin containers. Obviously, empty tin containers cannot be transported long distances because of their bulk in relation to weight. The recovery of tin from used containers has not been considered profitable and, with minor exceptions, has been attractive only under emergency conditions. This is due in part to the cost of collecting or purchasing cans, to the cost of cleaning cans, transportation and other factors. In 1953, 11,000 tons of used containers was processed, and only 70 tons of tin was recovered. Table 4 shows secondary tin recovered in the United States.

RESERVES

Producing countries that use dredges control substantial reserves, because the financing of dredging ventures is ordinarily predicated on the acquisition of enough gravel to assure operation of the plant for at least a decade. Those who operate by other placer methods seldom control such substantial reserves. In addition to reserves controlled by operating companies, a number of properties were tested before World War II but have not been brought into production or resumed operation because of war damage.

Lode mines usually have less assured life than those worked by dredging because of the high cost of blocking out ore underground. When allowances are made for indicated and inferred ore, however, the total lode reserves are large. In addition, there are areas, particularly in Malaya, Belgian Congo, and Bolivia, where geologic reconnaissance indicates major resources awaiting exploration and development. Table 5 lists the estimated tin reserves of the world.

The President's Materials Policy Commission Report (the Paley Report) estimates the tin reserves of the Free World as follows:

TABLE 5.—*Estimated Free-World tin-in-ore reserves by continents and countries*

[Long tons]

Continent and Country:	*Contained tin metal*[1] *(thousands)*
Southeast Asia:	
Malaya	1, 500
Indonesia	1, 000
Thailand	800
Burma	300
Subtotal	3, 600
Africa:	
Belgian Congo	500
Nigeria	250
Subtotal	750
South America: Bolivia	500
Other free countries	150
Total	5, 000

[1] As very little authoritative information is available on this subject, the foregoing estimates can represent little more than orders of magnitude.

USES

The most important single use for tin is the making of tinplate. Solder—an alloy of tin and lead (sometimes other metals are used)—is the second most important use of the element. Other alloys of tin include the white-metal group—babbitt, type metal, white metal, and others—and the brass and bronze group. Collapsible tubes and foil have smaller requirements. Various tin-bearing chemicals are commercially important. Consumption of tin by uses is indicated in table 6.

TABLE 6.—*Consumption of tin in United States, 1951–53, by finished products*

[Long tons of contained tin]

Product	1951			1952			1953		
	Primary	Secondary	Total	Primary	Secondary [1]	Total	Primary	Secondary [1]	Total
Tinplate	30, 522	--------	30, 522	[2] 27, 316	--------	[2] 27, 316	[2] 31, 327	--------	[2] 31, 327
Terneplate	84	683	767	85	495	580	333	310	643
Solder	13, 066	6, 744	19, 810	7, 678	10, 245	17, 923	10, 110	10, 063	20, 173
Babbitt	2, 493	3, 360	5, 853	1, 968	2, 637	4, 605	2, 492	2, 191	4, 683
Bronze and brass	4, 838	16, 934	21, 772	3, 612	16, 740	20, 352	3, 777	15, 738	19, 515
Collapsible tubes and foil	832	208	1, 040	604	104	708	917	127	1, 044
Tinning	2, 431	277	2, 708	2, 095	221	2, 316	2, 473	179	2, 652
Pipe and tubing	133	94	227	139	18	157	97	80	177
Type metal	120	1, 694	1, 814	86	1, 602	1, 688	171	1, 619	1, 790
Bar tin	875	54	929	642	36	678	835	71	906
Miscellaneous alloys	[3] 844	183	[3] 1, 027	[3] 485	297	[3] 782	294	279	573
White metal	134	146	280	81	53	134	104	140	244
Chemicals (including tin oxide)	374	826	1, 200	414	596	1, 010	481	828	1, 309
Miscellaneous	138	82	220	118	51	169	548	56	604
Total	56, 884	31, 285	88, 169	45, 323	33, 095	78, 418	53, 959	31, 681	85, 640

[1] Includes 5,180 long tons of tin contained in imported tin-base alloys in 1952 and 3,530 tons in 1953.

[2] Includes small tonnage of secondary pig tin and tin acquired in chemicals.

[3] Includes 592 tons of tin in Copan produced in 1951 and 213 tons in 1952.

TINPLATE

The largest use of tin has been in the manufacture of tinplate. The United States produces and consumes the largest quantity of tinplate in the world. More tin cans are produced in the United States than in any other country. Tinplate and terneplate are major products of the steel industry of the United States and normally represent over 5 percent of the total steel production.

Tinplate is black plate (thin, mild sheet steel) coated with commercially pure tin by the hot-dip process and the electrolytic process. Tin-coated vessels were mentioned by Pliny as existing before A. D. 23. Tinplate manufacture as understood today, however, was founded in the early Middle Ages (between 1240 and 1575) in Germany. The modern tinplate industry was introduced into England in 1670, and by 1730 tinplate was being produced from rolling-mill iron sheets. Bessemer steel for tinplate was introduced in 1880 in Wales. The tinplate industry of the United States came into existence in 1872; it made little progress, however, until the McKinley tariff became law on October 1, 1890. In 1891, the first year of record, tinplate production in the United States was 185 short tons. The first export of domestic plate was made in 1898 (quantity, 10 short tons). Growth of the tinplate industry in the United States was rapid and by 1912 equaled that of Great Britian; since 1914 the United States has been the leading world producer. United States practice began to use open-hearth steel from about 1905.

In the *electrolytic process* of tinplating, two broad classes of electrolytes are employed—those using an acid electrolyte and those using an alkaline electrolyte. Satisfactory tinplate can be made from either type. Each has certain advantages, and the choice of bath is governed by plant considerations and administrative aspects, such as patent rights.

The alkaline method has been known almost 100 years and has been utilized commercially for about 30. The acid bath is comparatively recent. Several successful acid baths are used, among them being the phenosulfonic acid bath, the fluoride-chloride bath, the fluoborate bath, and the sulfate bath. In each of these the electrode reactions are the same. Additions of acid are commonly made to raise the conductivity of the electrolyte and thus lower the plating power consumption. In general, the plating of tin from an alkaline solution had found uses for coating articles of complicated shape where a high throwing power is required, but these merits had no great importance in tinning steel strip. It was recognized only about 20 years ago that an alkaline stannate bath would deposit tin in smooth, compact form. The electrode reactions that take place in the alkaline bath are somewhat complex. The overall effects are well known, however, permitting satisfactory control of the deposition.

A typical *hot-dip tinning* shop or tinhouse normally consists of a long, rectangular building with the tin stacks arranged side by side in a straight line. Hot-dip tinning is performed by a continuous operation in the tinning stack or line. Sheared black-plate sheets, fed manually or automatically into the line, pass through a pickler to the tin-pot assembly (comprising flux, molten tin, and palm oil) to a wet washer, to branners, and then to the piler. The tin pot, which holds 8,000 to 17,000 pounds of commercially pure molten tin at about 620° F., is filled to a point about 4 inches from the top of the entry side. The pot is made of plate steel enclosed in a brick chamber and generally is heated by gas or oil burners or an immersion heater.

In usual practice, before packing for shipment, tinplate is visually inspected by assorting by operators who, from long experience or careful training, can discern any surface defects or slight errors in weight or dimensions. Methods of grading vary, but in common practice the assorter separates the tinplate as follows:

1. Primes or prime plates are those that show no visible defects, have no variation of substance outside specified limits, and are of the right weight and dimensions.
2. Menders are plates that show a defect in the tin coating, which can be rectified by passing them through the tinning unit a second time, converting them into primes.
3. Wasters (sometimes called seconds) have defects permitting use for certain less critical purposes. Sometimes, by arrangement, buyers may accept a certain portion of wasters along with primes, but usually they are marketed separately.
4. Waste-waste comprises plates too defective to be sold in above categories and may include "cobbles." Such plates are detinned if they carry heavy tin coatings.

The base box is a unit of measure peculiar to the tinplate industry; it corresponds to 112 tin plates of any gage 14 by 20 inches in surface dimensions, or a single surface area of 31,360 square inches (217.78 square feet). The area of tinned surface in a base box is 62,720 square inches, since both faces of tinplate are tinned. The unit of commerce is 1 base box of 100 pounds base-weight plate. Practical example: 100 sheets 18 by 25 inches in size equal 45,000 square inches each side or 1.434 base boxes.

"Terneplate" is a term applied to black plate or thin, mild-steel sheets coated on both sides with a lead-tin alloy. The word "terne" is French, signifying dull in color. Terneplate is produced as long terne sheets (or long ternes) and short ternes. Short ternes are also called manufactured ternes and are made the same way as tinplate with tin-mill equipment. Long ternes are made in sheet mills in sizes more comparable to galvanized sheets. Uses of the two classifications overlap.

Terneplate appears to have first been made for roofing on a very small scale in the United States in 1830 (*4*). It was introduced in Great Britain in 1848 and may have been made in France before that time. "Tin" roofs dating back to the early 18th century have been analyzed and found to be coated with this alloy. It was being exported to the United States by Great Britain as early as 1862. In 1891, however, large-scale manufacture began in the United States when 934 short tons were recorded produced (*25*). Peak production of terneplate in the United States reached 569,000 short tons (containing 2,045 long tons of tin) in 1941, with about an equal tonnage of both long and short ternes. In 1953 the total production was 278,200 tons (containing 643 tons of tin), of which about 77 percent was long ternes and 23 percent short ternes.

SOLDER

One of the most important uses for tin has been for making solder. Industrial applications of solder are innumerable. Tin used for solder consisted of 20,200 long tons of tin in 1953 compared with 18,000 tons in 1952. Solder alloys and the art of soldering have been known and practiced since early in the Christian era. According to Pliny the so-called "tertiarium" of the Romans contained twice as much lead as tin and was used as solder, and the "argentarium" of the Romans was an alloy containing equal parts of tin and lead—a common solder today. Wiped joints were made as early as the 15th century.

BABBITT METAL

Babbitt metal is a general term for soft-lead and tin-base bearing metals used cast as liners in bronze or steel backing. In 1953, 4,700 long tons of primary tin was needed for babbitt. The highest quantity recorded as used for making babbitt was 10,800 tons in 1917.

BRONZE

Bronze has been manufactured from very ancient times. The true bronzes contain principally copper and tin, but many alloys of copper and other metals such as brasses are commonly called bronzes. Industrial brasses, such as commercial bronze (90 percent copper and 10 percent zinc) with a color closely resembling bronze, are frequently referred to as bronze. Their tin content varies widely. The equipment for producing bronze is the same as for brass. They are also generally regarded as copper alloys and of great industrial importance. Admiralty, naval brass, leaded naval brass, and manganese bronze have the nominal composition 0.75 to 1 percent tin.

SUBSTITUTES

The important uses of tin have been established by long years of experience. No substantial new outlet has been established for tin, but from war developments there has resulted efficiency in the use of tin—that is, electrolytic and dual-coated tinplates and alloys. There is no direct and perfect substitute for tin in any of its uses, but partial and practical alternates have been developed. Glass and paper have been substituted for tin for packing foods and household necessities, thereby saving much tin that would otherwise be used in tinplate. Frozen foods and home freezers likewise displace tinplate. Leading manufacturers of cans have developed an enameled-steel container in which the only tin used is in a 3-percent solder wiping on the side seam of the can. An organic coating replaces tin to protect the steel and the contents of the can. The new electronic sterilization process for food preservation being developed offers opportunities for packaging with container materials that cannot stand the heat of normal sterilization. The 1950 annual report of the American Can Co. stated:

> The can industry in 1950 used about 45 percent less tin per container than in 1941 * * * It is an entirely forseeable fact that cans will some day be made entirely from materials which are available on the North American continent.

Aluminum, which is durable and economical in cooking utensils, has replaced much tin that would otherwise be used for tinplate ware and for tinning and retinning such articles. Nonmetallic products, as well as copper, zinc, and zinc-coated roofing, have supplanted much tinplate and terneplate used for roofing, siding, etc. For foil, other wrapping material may be used, such as aluminum foil, cellophane, parchment, and wax paper. In collapsible tubes aluminum or lead is applicable in many instances. In solder various metals, including silver, can be employed as substitutes. Roller and ball bearings and different alloys of less scarce metals may be substituted for babbitt bearing metal containing tin. Low-tin alloys have been specified for bronze. There are also substitutes for tin in chemicals. Substitution of rayon and nylon for silk and style changes have decreased the demand for tin chloride for weighting silk. As an enamel opacifier, other metal oxides, including titanium oxide have been found to be more effective and cheaper for most purposes. Most important, however, in reducing tin consumption has been the progressive displacement of the hot-dip method of tinplate manufacture by the electrolytic process during the last decade. The latter process requires considerably less tin per unit of product than hot-dipped. During 1951 the first commercial shipment of "differential" or dual-coated, electrolytic tinplate was made. This provides different weight coating on each side of the same sheet, which results in substantial saving in tin.

> When it appears that consumption might equal or exceed production, new technical methods, plant modernization, utilization of low-grade ores, and better mining methods will make available greater supplies of this semi-precious metal (*13, p. 59*).

According to Edward Steidle (*18*) by 2000 A. D.:

> Tin, a once necessary metal for corrosion protection, will have been replaced by more accessible metals, possibly aluminum and magnesium, or even by plastics.

STATISTICAL CONSIDERATIONS

Comprehensive statistical data are available on tin. The Bureau of Mines has been canvassing the tin industry for many years on an integrated basis for statistics on production, distribution, stock, receipts, shipments, and uses of primary and secondary tin. The statistics are gathered monthly and annually by mail, using questionnaire forms. The information is promptly published monthly and annually in mimeographed form, and in Minerals Yearbooks (*5*). Foreign trade statistics available from Census Bureau records are obtained by the Division of Foreign Activities, which also procures statistics for the world production tables that appear annually in Minerals Yearbooks. The International Tin Study Group publishes monthly and annual figures in great detail on the ramifications of the tin industry throughout the world.

PRODUCTION, CONSUMPTION, AND FOREIGN TRADE[5]

The United States depends upon foreign countries for tin, as only negligible quantities are available from domestic sources. Primary tin is derived from metal imports, and the output of the Government-owned Texas City tin smelter, which operates on imported concentrates. Secondary metal is obtained from scrap tin.

The tin requirements of the United States exceed those of any other country in the world and comprise metal for industrial uses (embracing civilian and military) and the National Stockpile and ores for smelter operations. During the 5-year period 1949–53, 32 percent of the world mine production of tin was required for the industrial needs of the United States. Tinplate, the largest single use, is indispensable in preserving food and feeding the United States Armed Forces. Bearing metals, bronze, and solder are important and essential to domestic industry.

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[5] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines from records of the U. S. Department of Commerce.

Tin, one of the principal imports of the United States, ranked eighth in value among all the commodities received in 1953. Imports of pig tin in 1953 totaled 74,538 tons and imports of tin in concentrates 35,973 tons. The principal source of metal has been Malaya and of tin concentrates Bolivia. Bolivia was the source of 64 percent of the tin in concentrates imported from 1941 through 1953; the concentrates were treated by the Texas City smelter, which began to produce in 1942.

Exports of metallic tin are not large, averaging about 600 long tons annually, destined mostly to Western Hemisphere countries.

Tables 7 and 8 present details ot the foreign trade of the United States in tin.

Figure 1 shows the source of tin imported into the United States, 1947–53.

Source:	*Thousand tons*	*Percent*
Belgian Congo	2. 9	3
Belgium, Netherlands, United Kingdom	19. 6	23
Bolivia	17. 1	19
Indonesia, Malaya, Thailand	47. 4	53
Other	2. 1	2
	89. 1	100

TABLE 7.—*Foreign trade of the United States in tin concentrates and tin, 1946–53*

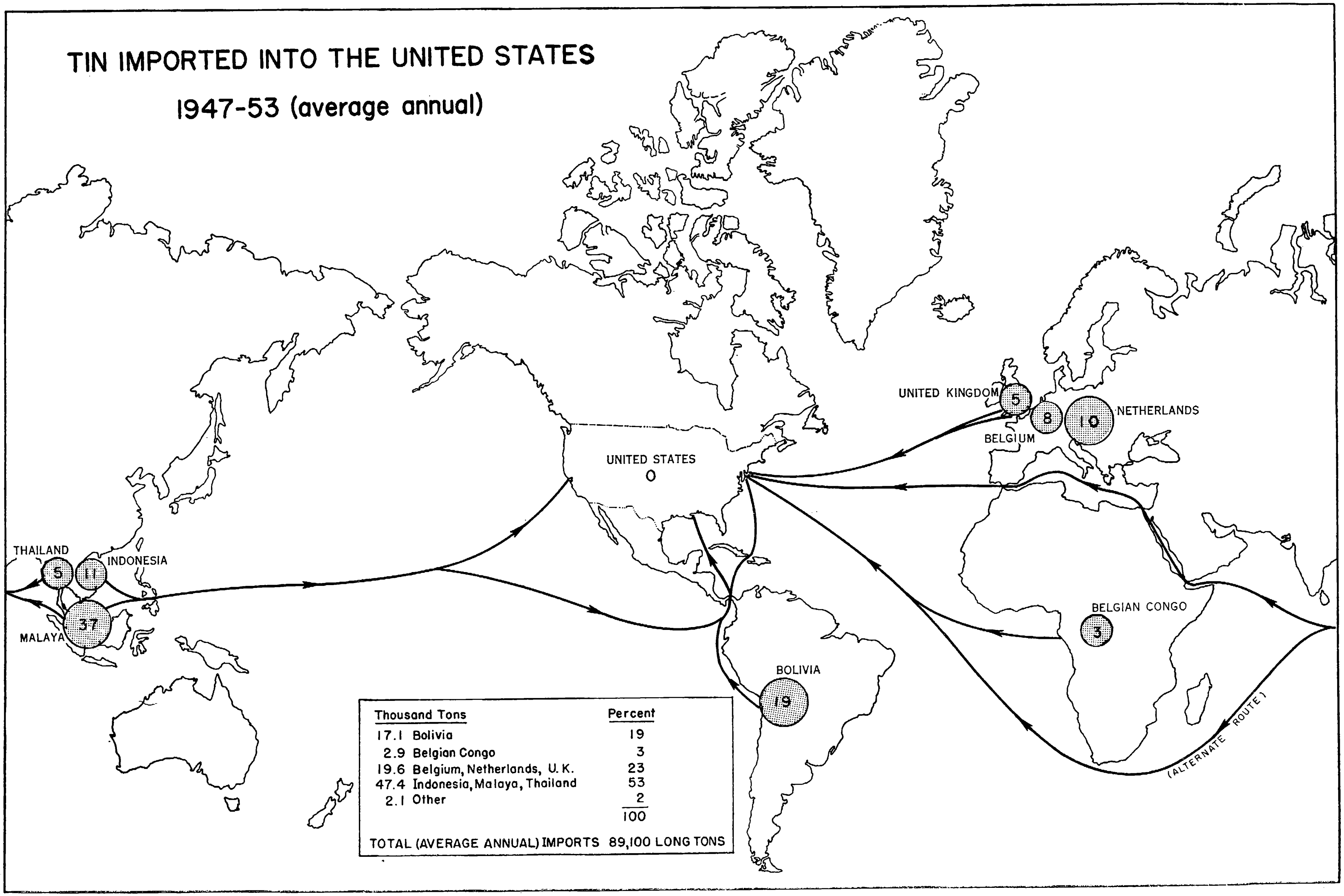
(U. S. Department of Commerce)

	Imports				Exports			
	Concentrates (tin content)		Bars, blocks, pigs, grain, or granulated		Ingots, pigs, bars, etc.			
					Domestic		Foreign	
	Long tons	Value	Long tons	Value	Long tons	Value	Long tons	Value
1946	38, 070	$50, 623, 185	15, 559	$18, 554, 896	859	$1, 153, 936	22	$31, 939
1947	29, 410	43, 220, 686	24, 899	42, 684, 651	415	650, 162	5	9, 887
1948	37, 492	72, 170, 372	49, 196	103, 322, 952	78	163, 428	13	27, 699
1949	38, 311	78, 175, 836	60, 224	133, 707, 223	76	176, 795	78	145, 370
1950	25, 960	47, 163, 305	82, 838	152, 952, 294	287	594, 587	512	990, 000
1951	29, 621	82, 462, 215	28, 255	74, 556, 994	264	762, 662	1, 249	3, 978, 852
1952	26, 491	65, 286, 937	80, 543	215, 603, 146	301	580, 855	79	209, 539
1953	35, 973	82, 713, 269	74, 538	175, 858, 286	128	297, 695	75	141, 901

TABLE 8.—*Tin imported for consumption in the United States, 1935–39 and 1947–53 (average)*

[Long tons]

Country	1935–39 (average)				1947–53 (average)			
	Metal	Concentrates (tin content)	Total	Percent of total imports	Metal	Concentrates (tin content)	Total	Percent of total imports
Malaya	48, 350		48, 350	69	33, 114	7	33, 121	37
Bolivia	37	137	174		86	17, 047	17, 133	19
Indonesia	3, 549		3, 549	5	7	9, 748	9, 755	11
Netherlands	2, 426		2, 426	4	8, 493		8, 493	10
Belgium	494		494	1	6, 682		6, 682	8
Thailand					1, 079	3, 407	4, 486	5
United Kingdom	9, 042		9, 042	13	4, 402	1	4, 403	5
Belgian Congo	239		239		1, 733	1, 145	2, 878	3
Other	5, 504	45	5, 549	8	1, 617	540	2, 157	2
Total	69, 641	182	69, 823	100	57, 213	31, 895	89, 108	100



TIN IMPORTED INTO THE UNITED STATES
1947-53 (average annual)

Thousand Tons		Percent
17.1	Bolivia	19
2.9	Belgian Congo	3
19.6	Belgium, Netherlands, U. K.	23
47.4	Indonesia, Malaya, Thailand	53
2.1	Other	2
		100

TOTAL (AVERAGE ANNUAL) IMPORTS 89,100 LONG TONS

GOVERNMENT SUPERVISION

NATIONAL PRODUCTION AUTHORITY

Following approval of the Defense Production Act on September 8, 1950, the National Production Authority was established on September 11. Inventory Control Regulation 1 (September 18, 1950) was the first to be issued by NPA. It limited to a "practicable minimum working inventory" stocks of various materials in inadequate supply, including primary and secondary tin and all tin and tin-base alloy scrap containing commercially recoverable tin. The announced purpose of the order was to make clear that national interest demanded that there be no accumulation beyond what was needed for immediate production. Under NPA Order M-8 (November 13, 1950) tin was placed under Government control. All uses of tin not expressly set forth in the order were prohibited. Tin reporting was made mandatory, and inventories of alloys or other materials containing tin, excluding ores and concentrates, were limited to 60 days' supply or a practicable minimum working inventory (as defined in NPA Regulation 1) whichever was less. Later limitations on pig-tin inventories were made more exact and specified 120 days' supply for tinplate manufacturers, and 60 days for all others. Effective August 1, 1952, private firms could resume importing tin from suppliers of their own choosing.

On December 18, 1950, an amendment to Order M-8 limited the consumption of pig tin to 100 percent of the monthly average usage during the first half of 1950 and was applicable to January 1951 tonnages. Use was cut to 80 percent in February and March. Defense orders were excepted, and allowance was made for undue or exceptional hardship. During April, May, and June the rate was increased to 95 percent for tinplaters and 90 percent for all other users. Commencing May 1, pig tin was placed under allocation by prohibiting, subject to limited exceptions, any deliveries not covered by allocation authorizations by NPA. Previous limitation rates allowed extra quantities for producing implements of war, but after July 1 all users were subject to the 90-percent limit.

The use of pig tin for making tinplate and terneplate was limited by M-8, which also permitted the use of tinplate and terneplate in accordance with NPA Order M-24. M-24 controlled the use of tinplate and terneplate (for cans M-25 and for closures M-26), the permitted materials in these uses, and the maximum permitted coatings of tin or terne metal. Effective July 1, 1951, tinplate and terneplate became subject to allotments under the Controlled Materials Plan (CMP). The use of tin in the manufacture of collapsible tubes was limited by Order M-27, the percentage varying with products to be packaged in the tubes. The supply of tin reached virtual balance with defense and essential civilian needs in 1952. All controls over uses and inventories of tin were removed in 1953.

Under NPA Delegation 5 (December 18, 1950), the Secretary of the Interior was made claimant for certain production facilities for minerals and metals and given specified powers to control the distribution of metals and minerals through the processes of smelting and refining. The delegation included tin ores and concentrates as related to mines, mills, and primary and secondary smelters and refineries. NPA Notice 1 (December 27, 1950), and Defense Minerals Administration Order MO-1 (December 29, 1950), designating scarce materials whose hoarding was prohibited, included tin, all alloys containing tin, tin chemicals, tin products, tin-base alloys, and tin ores and concentrates.

Tin metal and materials containing tin, including tinplate, required an export license for shipment to any destination abroad except Canada.

Tin was under Government allocation control from December 17, 1941 (M-43), until December 1, 1949, when the control regulations were reduced to a reporting basis until June 30, 1950, when Public Law 153 expired. The experience during World War II with allocation Order M-43 was applied during the defense program of 1950.

From March 12, 1951, to July 31, 1952, the Reconstruction Finance Corporation was the only importer of tin in the country. Resumption of private importation was again permitted by an amendment to NPA Order M-8 (August 1, 1952).

TEXAS CITY SMELTER

Public Law 723, 81st Congress, 2d session, approved August 21, 1950, extended to June 30, 1956, the authority of the RFC to improve, develop, maintain, and operate by lease or otherwise the Government-owned tin smelter at Texas City, Tex., and to finance research in tin smelting and processing.

The operation of the Government-owned Longhorn tin smelter, Texas City, Tex., has been justified as a national defense measure because of the status of the National Stockpile of tin. Also, the importance of Bolivian tin production to Western Hemisphere economic stability and military defense has been recognized. The original decision to build this plant was made in 1940, when the German blockade cut off markets for substantial quantities of

Bolivian tin ore and British smelters were under aerial attack. The smelter has been operating since 1942. The plant was constructed for the account of the Defense Plant Corporation and operated on a management-fee basis for Metals Reserve Company. MRC and DPC were dissolved July 1, 1945, and their functions transferred to RFC. Operation of the smelter was then authorized to June 30, 1947, then for 2 more years to June 30, 1949; then for 2 more years to June 30, 1951 (*22*); and on August 21, 1950, its life was extended for 5 years, to June 30, 1956 (*23*).

As a consequence of the RFC Liquidation Act (*24*), Senate Concurrent Resolution (S. Con. Res. 79) of July 14, 1954, stated among other things that it was the sense of the Congress that the smelter should be continued in operation at least until June 30, 1955. Effective July 1, 1954, operation of the smelter was transferred from the RFC to the Federal Facilities Corporation, a new agency established in the Treasury Department by Executive Order 10,539 of July 22, 1954.

Since its inception, the Texas City smelter has been operated by the Tin Processing Corp. (a Delaware corporation and a subsidiary of N. V. Billiton Maatschappij), an independent contractor, under an operating agreement with RFC or FFC. In conjunction with this arrangement FFC purchases all concentrates, pays all operating costs, and disposes of the resulting tin.

TARIFF

There has been no import duty on tin since 1895. Paragraph 1785 of the Tariff Act of 1930 reads as follows:

> Tin ore or cassiterite, and black oxide of tin: Provided, that there shall be imposed and paid upon cassiterite, or black oxide of tin, a duty of 4 cents per pound, and upon bar, block, pig tin and grain or granulated, a duty of 6 cents per pound when it is made to appear to the satisfaction of the President of the United States that the mines of the United States are producing one thousand five hundred tons of cassiterite and bar, block, and pig tin per year. The President shall make known this fact by proclamation, and thereafter said duties shall go into effect.

Manufactured forms of tin that carry a duty include tinplate and terneplate, foil, powder, chemicals, bottle caps, collapsible tubes, and manufactures not specially provided for.

PRICES

The price of tin is quoted in many markets; but the principal quotations are those in New York in cents per pound, in London in pounds sterling per long ton, and in Singapore in Straits dollars per picul. Both the Commodity Exchange, Inc., New York, and the London Metal Exchange permit trading in tin for immediate and future delivery. Table 9 shows tin prices in New York monthly from 1944 to 1953.

TRANSPORTATION

To move tin ore or concentrates to the smelter and tin bullion from smelter to market, reliable transportation by water or land should be available, as interruption or periodical discontinuance could prove burdensome and expensive. The possibility of interruption of imports of tin because of loss through sinkings has been an important strategic factor from a national defense angle.

Most of the tin coming to the United States has arrived at North Atlantic ports from the Far East shipped through the Suez Canal and the Mediterranean. Bolivian tin ores move from the west coast of South America to the

TABLE 9.—*Monthly average prices of Straits tin in New York, 1944–53*[1]

[Cents per pound]

	1944–45	1946	1947	1948	1949	1950	1951	1952[2]	1953
January	52.00	52.00	70.00	94.00	103.00	75.92	171.72	109.73	121.50
February	52.00	52.00	70.00	94.00	103.00	74.35	182.68	121.50	121.50
March	52.00	52.00	70.00	94.00	103.00	75.74	145.46	121.50	121.40
April	52.00	52.00	80.00	94.00	103.00	76.45	145.83	121.50	101.11
May	52.00	52.00	80.00	94.00	103.00	77.50	139.95	121.50	97.46
June	52.00	52.00	80.00	103.00	103.00	77.70	118.05	121.50	92.95
July	52.00	52.00	80.00	103.00	103.00	89.88	106.00	121.50	81.63
August	52.00	52.00	80.00	103.00	103.00	102.05	103.00	121.15	80.68
September	52.00	52.00	80.00	103.00	102.00	101.29	103.00	121.38	82.31
October	52.00	52.00	80.00	103.00	95.70	113.35	103.00	121.23	80.85
November	52.00	65.00	80.00	103.00	91.11	137.67	103.00	121.27	83.19
December	52.00	70.00	85.33	103.00	79.08	144.77	103.00	121.47	84.61
Average	52.00	54.58	77.94	99.25	99.32	95.56	128.31	120.44	95.77

[1] American Metal Market.

[2] Open-market prices since August 1952.

Gulf of Mexico through the Panama Canal. Ships carrying metallic tin and tin ore from Asia sometimes move through the Panama Canal. According to the Second Annual Reports of the Panama Canal Company and the Canal Zone Government for the fiscal year ended June 30, 1953, during the fiscal year 1953 cargo declarations show that 102,816 long tons (gross weight) of tin ore moved through the Canal—66,053 tons to the United States (of which 56,787 was from South America and 9,266 from Asia) and 36,763 to Europe from South America. It takes 55 to 60 days for a shipment from the Far East to reach the United States and about 2 weeks from the west coast ports of South America to Galveston, Tex. Shipments of small tonnage lots of tin ores or metal from Portugal or the Far East may take much longer.

MARKETING

Tin is usually marketed in pigs, bars, or ingots of varying weights and grades, depending on the individual smelter. First-quality grades or brands (99.8 percent plus) include Straits (Malayan smelters), Longhorn 3 Star (Texas City Smelter), Banka and Billiton (Netherlands at the Arnhem smelter), and English refined in the United Kingdom and at Hoboken in Belgium. Lower grades, which sell more cheaply, are suitable for many uses, such as alloying. London and Singapore are the predominant world exchanges, followed to a smaller degree by New York.

The London Metal Exchange deals in units of 5 long tons of tin metal for both immediate and future deliveries. Cash dealings (for delivery the next day) apply only to metal stored in official warehouses approved by the Metal Exchange, while future deliveries range from succeeding weekly market days up to 3 months. Dealings in London are for both physical tin and for speculative purposes where actual delivery is not intended.

The Singapore tin market operates in a way that seems unusual at first to Americans because it differs from some of our commodity markets in that there is no open bidding on the floor; yet it has worked efficiently and satisfactorily for 50 years and reflects a fair balancing of supply against demand.

Bids are submitted in writing by buyers and opened each morning by the market manager, who is also notified of the supply of tin available for sale by the two Malayan smelters. The bids are matched up with the tin offered in the following manner: Starting with the highest bid, as many bids are accepted as will take up all the tin offered, and all the successful bidders get their tin at the price of the lowest of the high group of bids that matches the quantity of tin available; this is the price at which all the tin offered for the day is sold. Occasionally there are not enough bids to take up all the tin offered. But since the reopening of the market this has happened only 1 day in every month or two. Any tin left over must be absorbed on the following days, and the carryover is known to bidders, so the sales continuously represent a balance between supply and demand.

Most European companies either sell their tin on the day when each shipment reaches the smelter or spread their production for the month equally over the market days of that month. The objective is merely to receive the average price for tin. The Chinese are less systematic and more inclined to gamble on the timing of offering their tin.

There is a comparatively even flow of about 125 to 300 tons per day from the mines to the Malayan smelters with the average around 150 to 200 tons. Except for the odd day every month or so this tin is disposed of on the day that it reaches the market. All sales represent actual tin for 60-day delivery. There is no paper tin in the Singapore market. This method of market operation was evolved to minimize speculation in tin (*14*).

The New York market (which is the sum of all tin transactions accomplished in New York rather than a specific organization or place of business) has a structure different from either the London or Singapore markets and cannot be considered a world market in the true meaning of the word. Its importance is due to the fact that it is the purchasing point for the world's largest consumers of tin. New York prices are quoted on the basis of actual physical sales and offerings of tin for prompt and forward delivery. It also depends on whether the market is moving; then the best offer at the close of trading makes the market. The New York Commodity Exchange, a definite organization, trades in tin futures (for delivery each month and the succeeding 11 months), but the tonnage involved is insignificant to that of physical transactions in New York and cannot compare with future dealings on the London Metal Exchange.

INTERNATIONAL TIN STUDY GROUP

Before the second World War there was in existence an International Tin Committee, representing the most important producing nations, which was concerned with regulating tin production through national quotas enforced by law on individual mines. Afterward extension of this organization for production control was strongly opposed by the United

States, and it was decided to place the problem of tin production in the hands of a study group organized under the United Nations. The first step toward that end was a meeting of representtives of eight major tin producing and consuming countries—the United Kingdom, Belgium, Bolivia, China, France, Netherlands, Thailand, and the United States—in London in October 1946, which agreed that a study group should be established. The International Tin Study Group was organized at a meeting in Brussels in April 1947. The basic reason for the existence of the study group was to effectuate stabilization of the tin industry through international cooperation free from private trade restraints, as outlined by the United States Department of State in its proposal to establish an international trade organization, which, subject to ratification of member governments, came into being at a meeting in Havana in the spring of 1948. (The United States Congress has not approved the Havana Charter.)

A management committee (selected each year) in 1952 was composed of representatives from Belgium, Bolivia, British Colonies, France, Indonesia, United Kingdom, and United States. This committee supervises the work of the group's secretariat, which has been established at 7, Carel van Bylandtlaan, The Hague, Holland.

The membership of the group in 1952 consisted of Australia, Belgium, Bolivia, British Colonies and Dependent Territories, Canada, China, Czechoslovakia, France, Indonesia, India, Italy, Netherlands, Thailand, United Kingdom, and United States.

INTERNATIONAL TIN AGREEMENT OF 1953

The United Nations Commodity Conference on Tin, which was held at Geneva, Switzerland, from November 16 to December 9, 1953, approved, with amendments, the draft of an International Tin Agreement prepared by the International Tin Study Group at a meeting in London in August 1953. The text of this agreement—described as the International Tin Agreement of 1953 (*6*)—was submitted to governments for their consideration and was open for signature from March 1 to June 30, 1954.

The representation at the conference included delegations from 23 countries and observers from 7. The delegations represented, on the basis of the figures for 1952, about 113,000 long tons of tin consumption out of a known world tin consumption of 130,000 tons and about 160,000 tons of tin production out of a known total of 174,000 tons.

The International Tin Agreement of 1953 provides for the formation of an International Tin Council in London, with powers to fix periodically the quantity of tin that might enter world trade through the imposition of export quotas. The agreement aims at stabilizing market conditions for industry through establishment of a buffer stock and through acquisition of 25,000 tons of tin metal to be contributed by producing countries.

The agreement establishes, as an initial basis for operation, a floor price of £640 sterling (roughly equivalent at $2.80 per pound sterling to $0.80 a pound) and a ceiling price of £880 (equivalent to $1.10 a pound). Operation of the buffer stock would be entrusted to a manager and would depend primarily on the relation of the market price to the floor and ceiling prices.

In the agreement drafted at the United Nations Tin Conference in Geneva in 1953 it was required that the signatures and ratification of the Governments of producing countries, holding together at least 900 votes, and of at least 9 consuming countries, holding at least 333 votes, should be obtained before the agreement could come into force.

By June 30, 1954, 6 producing countries—Belgian Congo, Bolivia, Malaya, Nigeria, Indonesia, and Thailand, together holding 1,000 votes—had signed. Fourteen consuming countries—Australia, Belgium, Canada, Denmark, Ecuador, France, India, Italy, Japan, Lebanon, Netherlands, Spain, Turkey, and United Kingdom, together holding 427 votes—had also signed. The United States had already indicated it would not adhere to the agreement.

As soon as possible after complete ratification the United Kingdom will summon the first meeting of the International Tin Council in London to administer the provisions of the agreement and supervise its operation.

TIN-SMELTING POSITION

About 90 percent of the world output of tin concentrates is smelted by 6 smelting companies in 5 countries.

There are only 9 major tin smelters now in operation—2 in Malaya at Penang and Singapore, 2 in Great Britain at Bootle near Liverpool and at Bristol, 1 in the Netherlands at Arnhem, 1 in Belgium at Hoboken, 1 in West Germany at Duisburg-Wanheim am Rhein, 1 in the Belgian Congo at Manono, and 1 in the United States at Texas City, Tex.

Apart from these major smelters, a number of smaller units are concerned in many instances with the treatment of local mine production or with handling low-grade concentrates.

Since 1941, by long-term contract or otherwise, the United States has purchased concentrates mainly from Bolivia, Indonesia, Thailand, and Belgian Congo. In recent years the movement of tin concentrates has been about as follows: Malayan, Burmese, and a major proportion of the Siamese concentrates have gone to the Malayan smelters; Indonesian concentrates have moved about two-thirds to the Netherlands smelter and one-third to the United States; all Nigerian concentrates and about half the Bolivian production (High-grade) have gone to the United Kingdom; the smaller proportion of Thailand's concentrates and about half of the Bolivian concentrates (Low-grade) to the United States; and the major part of Belgian Congo output to the Belgian smelter and the minor part distributed between the local Congo smelter and the United States. South African, Japanese, Chinese, and Australian production has been smelted locally.

OUTLOOK

World production of tin concentrates is accounted for by 30-odd countries, of which only 6 will continue to exercise a decisive influence on the volume of output—Malaya, Indonesia, Bolivia, Thailand, Belgian Congo and Nigeria, which supplied 90 percent of the world production in 1953.

World reserves of tin ore, plus potential reserves to be developed, should maintain tin production at about present levels for at least 20 to 25 years. The possibilities for increasing these reserves are considered good, through continued exploration and improved technology of mining and concentration.

Primary tin is not in short supply. World production exceeds industrial requirements; and adequate supplies will continue to be available to the United States in the foreseeable future, assuming there is no political interference with current sources of supply. Moreover, the United States Government holds strategic and other stocks of tin equal to well over 5 years' requirement of primary tin at the current annual rate of consumption. The demand for tin does not have the bright future that some of the metals like aluminum, magnesium, titanium, and molybdenum may have, but neither has it a dark outlook. It soon may not be regarded as a critical metal because of known alternative materials. Under conditions of no artificial restraint there should be a modest growth in the total use of tin in the next 25 years, but it probably will be well under the average growth in the use of other metals. There is a strong tendency at present in the United States to develop substitutes for tin. This is largely the result of the real shortage in tin supply during World War II, and the apparent shortage of the past few years.

Likewise, as regards conservation of tin, there are many facts to substantiate the surmise that the long-range trend toward reduced tin coatings on the tinplate used by the container industry is far from finished, and there appear to be no technical indications to the contrary.

As a result of extensive research in recent years, some new uses for tin are being developed to replace at least part of the losses in former applications. The most promising of these is the coating of various products by electrodeposition with tin-base alloys.

In the chemical field there are a few new applications of organic chemicals for pharmaceuticals, plastics, and lubricating oils; however, the largest and still growing field for tin salts is for forming electrolytes for electrodeposition of tin. This application is expected to increase substantially.

PROBLEMS

The tin industry in the United States is a consuming industry, and its chief concerns and problems are related largely to the supply and the price of tin. Only passing reference need be made here to certain situations and developments previously described—the political instability in major tin-producing countries and the efforts being made to remedy these conditions; and the determined effort by producing nations under an International Tin Agreement with United Nations sponsorship to achieve stabilization of the supply and price through export control and buffer-stock manipulation. The solutions of these questions lie chiefly in foreign lands and foreign hands and only to a limited extent are amenable to efforts by consumers or Government in this country. It is widely recognized that, in the long run, the welfare of the tin-producing industry depends on its success in assuring the world of an ample supply of tin at a reasonable cost. As things are, this is a staggering problem.

The tin-producing industry regards decreasing demand for tin as a major problem and devotes substantial funds to research and development efforts for new and expanded uses.

Advantage would accrue to both producers and consumers if appreciable gains could be achieved in recovery. Tin ore is outstanding in the difficulties and problems it presents to ore dressers and the recoveries both in lode and placer mining are distressingly low. Any significant improvement would be equivalent to a reduction in production cost or an increase in ore reserves.

Consumers are forced by past experience, by cost considerations, or by their estimate of things to come to give some thought to economies in the use of tin or to substitutes for it. Whether as peacetime or solely as emergency measures these two fields present many technical problems. Tin has no equal at any reasonable cost for many of its uses. There the problem is to use it with the maximum efficiency. What can be done is well illustrated by the tinplate industry. In some instances complete substitution has been achieved on a permanent and apparently satisfactory basis. In others substitution is only tolerable at best and in emergencies. The continuing quest for substitutes invariably encounters difficult technical problems and offers the widest scope for research and development.

A current question in the United States is to determine the possibility and practicability of producing a significant quantity of tin in Alaska; this involves not only the high costs (which are to some degree related to the Arctic weather with long winters, the seasonal shortage of water, and other adverse local factors), but also determination of the indicated size and character of the deposits.

The United States Government has a problem in the management and eventual disposition of the Texas City tin smelter. Since its inception the smelter has been operated by the Government; and costs were higher than those elsewhere, largely because the Texas City smelter accepted all grades of concentrates, some of which were extremely expensive to treat.

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TITANIUM

By

F. J. Cservenyak [1] and A. F. Tumin [2]

THE WHITEST pigments, the "middleweight champion" in the metals field, and glamorous gems are all derived from the "wonder" element, titanium.

Summary

Titanium is estimated to be the fourth most plentiful structural metal in the earth's crust. Although it is widespread in the United States and throughout the world, only a few ores are commercially useful under present technology. Ilmenite—one of the minerals found in titaniferous ores—is the major raw material for the production of titanium dioxide pigment. Before World War II India supplied United States consumers with most of their needs. War disruption of seaborne trade gave impetus to the search for alternate domestic sources, with the result that new domestic deposits were developed, thereby reducing reliance on imports considerably. Canadian operations on the production of high-titania slag from vast deposits of titaniferous hematite offer the United States a supplemental source of raw material for the industry for many years to come.

Rutile, the other commercially important titanium mineral, is used mainly for welding-rod coatings and titanium metal and in smaller quantities for alloys, ceramics, and fiberglass. About four-fifths of the domestic supply in 1953 was imported from Australia, the major rutile-producing country and the only one that shipped rutile to the United States from 1948 to 1953. Although rutile was used in 1954 in producing titanium metal, it is not available in the United States in large enough quantities to support an expanding titanium-metal industry. Any major expansion of titanium-metal production must be premised mainly on the use of ilmenite (including leucoxene) or high-titania slag.

The phenomenal growth of titanium dioxide pigments may be attributed principally to their high hiding power per unit volume. Metallic titanium owes its importance to an unusual combination of properties—lightness, strength, and resistance to corrosion. Commercial production of titanium metal in 1954 was based on modifications of the Kroll process developed by the Federal Bureau of Mines.

Government programs called for increased expansion of titanium dioxide- and metal-production facilities. The metal demand in 1954 was mainly for military purposes where the advantages derived justified the high prices.

Efficient utilization of the widespread domestic titanium ores requires examination, classification, and development of methods for economically

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[1] Chief, Branch of Light Metals, Bureau of Mines.

[2] Commodity-industry analyst, Bureau of Mines.

recovering the titanium values of these ores, including the titanium-iron ores for which commercially feasible processes have been developed.

The continued expansion of titanium-metal production and widespread use of this metal for civilian as well as military purposes depend mainly on reduction in the cost of production and fabrication and growth in knowledge of the properties and uses of titanium and its alloys.

In 1954 titanium ore was mined commercially from rock and sand deposits by open-pit and dredging methods. Mechanical beneficiation methods were employed for concentrating the major titanium minerals—ilmenite and rutile—to meet industry specifications.

Ilmenite was the principal titanium mineral consumed in the titanium-pigment industry. Titanium oxide of pigment quality was produced by the sulfate process, in which titanium dioxide in the ilmenite is dissolved by sulfuric acid and then precipitated by hydrolysis. One firm was producing some titanium dioxide by oxidation of titanium chloride.

In the commercial manufacture of ductile titanium metal by the Kroll process titanium minerals were chlorinated in the presence of carbon to produce titanium tetrachloride. This product was then reduced with magnesium in an inert atmosphere to produce titanium-sponge metal. Titanium sponge was solidified by either arc or induction melting to produce titanium ingots.

BACKGROUND

SOURCES

Titanium is estimated to be the ninth most common element and the fourth most plentiful structural metal in the earth's crust. Spectroscopic or other sensitive qualitative tests show titanium to be distributed widely in low concentration. The pure element (titanium) does not exist in natural form because of its great reactivity with other elements. Minerals rich in titanium are few, and only ilmenite ($FeTiO_3$) and rutile (TiO_2) have commercial importance. Titanium-bearing minerals, such as anatase, leucoxene, brookite, and others, are associated with ilmenite and rutile and usually comprise part of the ilmenite or rutile ore marketed. Ilmenite from some localities has been altered by weathering and contains TiO_2 over the theoretical content. Titanium slag, made from titaniferous iron ores by a smelting process, also has commercial value to the titanium industry.

Ores containing titanium are found extensively throughout the world (*3, 8, 16, 25*).[3] Ilmenite and titaniferous iron ores occur in the United States, India, Norway, Canada, Japan, Egypt, Africa, Ceylon, Malaya, Sweden, Brazil, Mexico, and the U. S. S. R. Commercial sources of rutile were reported in Australia, United States, Brazil, French Cameroon, French Equatorial Africa, and Mexico. Domestic ilmenite rock deposits are found in Virginia and North Carolina. Extensive deposits of titaniferous iron ores occur in New York, Minnesota, Montana, Rhode Island, Wyoming, California, and New Mexico. Other occurrences of iron ores rich in titanium have been reported in Arizona, Colorado, North Carolina, South Carolina, Montana, New Jersey, Oklahoma, and Tennessee. Rutile and ilmenite, with other minerals, are found in beach sands at many places along the Atlantic, Pacific, and Gulf coasts and also in stream sands; they have been worked mainly in Florida and to some extent in Idaho and Oregon. Rutile-rock deposits occur in Virginia and Arkansas. Ilmenite is also present in the black sands of Arkansas bauxite and Florida phosphate.

In general, two types of deposits contain titanium minerals of economic importance, namely, rock and sand. Theoretically pure ilmenite contains 52.67 percent TiO_2, and ilmenite from rock deposits and some sand deposits commonly contain 42 to 50 percent TiO_2. Some sand deposits, however, yield altered ilmenite containing 60 percent or more TiO_2. In rock formations, chiefly anorthosite and related rocks, ilmenite is characteristically associated with iron ores, principally magnetite, but in some places with hematite. Ore-dressing methods may be employed in concentrating ilmenite from those ores in which the titanium and iron minerals are in discrete grains; however, where ilmenite is intimately intergrown with the iron minerals a chemical or pyrometallurgical process is required to recover the titania. Ilmenite in pure state is known to occur in minable rock bodies (nelsonite and schist). The most important occurrences of rutile are in beach sands; however, it is found in nearly all rocks as an accessory mineral and in some rocks is concentrated enough to make them actual or potential ores.

In sand or sedimentary deposits ilmenite occurs with rutile, monazite, zircon, garnet, staurolite, and other minerals of high specific gravity—the so-called "heavy minerals." Titanium minerals found in beach sands come from weathered igneous rocks. Fragments of the rocks are carried to secondary deposits (beach, dune, and placer) by erosion or rainfall where wave and wind action aid in concentrating the heavy minerals.

North American rock deposits containing titanium-iron ores and other titaniferous material were mined in 1954 at Tahawus, N. Y.; Piney River, Va.; and Allard Lake, Quebec, Canada. Before 1954 titanium minerals were mined from rock deposits at Magnet Cove, Ark.; Richland Cove, N. C.; and Roseland, Va.

The Tahawus deposit outcrops on the side of Sanford Hill overlooking Sanford Lake in the Adirondack Mountains and is composed of lenticular masses of closely associated magnetite and ilmenite separated by gabbroic and anorthositic waste zones and inclusions. The length of the deposit is approximately 1,800 feet, with a maximum width of 900. The deposit is about 4 miles north of the southern boundary of a large anorthosite (sodium and calcium aluminum silicate) body (*2*). The ilmenite can be recovered mechanically, and the mining and concentrating operation is known as the MacIntyre development of the National Lead Co.

Titanium minerals in the Virginia deposits

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[3] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

occur near the northwest margin of the Piedmont Plateau in a zone that trends northeasterly across Amherst and Nelson Counties, Va. The dikelike nelsonite ore bodies at Piney River, Va., contain ilmenite with apatite (calcium fluophosphate) and other gangue materials and are reported to be related to a large anorthosite-type intrusion approximately 19 miles long by 2 to 6 miles wide. The titanium deposits of the Roseland district, Va., consist of rutile and ilmenite disseminated in a feldspar rock. The nelsonite deposits are peculiar rocks characterized by the presence of rutile and apatite or by ilmenite, apatite, or both, and only a few large deposits of this type have been reported (*34*).

The titanium ore body at Richland Cove, Caldwell County, N. C., approximately 14 miles from Lenoir, is a dikelike intrusion in gneiss and consists of ilmenite and rutile with a micaceous gangue formed of chlorite, talc, and mica. Only the ilmenite was recovered in past operations (ceasing in October 1952) as the rutile occurs as fine crystals within the mica, which was discarded in the concentrating process. The deposit was reported to be 2,000 feet long, with an average thickness of 20 feet.

Uplift, depression, and erosion developed the configuration of the Arkansas titanium-mineral rock deposits known as Magnet Cove, Christy, and Hardy-Walsh. These deposits consist of a complex mixture of igneous, sedimentary, and metamorphic rocks, many of which contain various quantities of titanium-bearing minerals. Much of the titanium is in the form of rutile or brookite, but many other titanium minerals have been identified. Rutile was recovered commercially from the Magnet Cove deposit from 1932 to 1944 (*7*, *14*, *32*, *33*). About 50 tons of brookite ore was mined from the Magnet Cove deposit in 1953 by the National Lead Co., New York, N. Y., as called for by a contract signed April 1, 1953, between the Government and the National Lead Co. The material mined was to be studied to determine the feasibility of utilizing brookite in producing welding rods and titanium metal. No commercial output of titanium minerals has been reported from the Christy or Hardy-Walsh deposits.

The Bureau of Mines has conducted extensive investigations on processing and utilization of the titanium minerals in Hot Spring County, Ark. This work, described in Bureau reports of investigations, included studies of drilling, mineral dressing, smelting, chlorination, and use of this ore for welding-rod coatings (*12*, *13*, *17*, *22*, *44*).

Other Bureau of Mines laboratory investigations developed mechanical and thermal methods for conservation and recovery of a low-grade ilmenite concentrate and an iron oxide concentrate from "black sands," a waste product of the Reynolds Metals Co. modified Bayer alumina plant at Bauxite, Ark. Utilization of this low-grade ilmenite concentrate would depend on development of smelting techniques for production of high-titania slags from this material (*7*).

The largest known occurrence of ilmenite in the world has been exploited at Allard Lake, Quebec, Canada. Here the ilmenite is intergrown with hematite in ore bodies consisting of a series of narrow dikes, irregular lenses, or sill-like bodies lying within an anorthosite mass covering 134 acres (*19*). The ilmenite cannot be recovered by standard ore-dressing procedures; consequently the ore requires smelting to yield usable products.

Titanium and other heavy minerals occur widely in Florida sands. Weathering of schists, gneisses, and granites in the Piedmont region of Georgia and North and South Carolina released detrital minerals that were ultimately transported into Florida by streams and ocean currents and deposited along their existing shores. Subsequent lowering of the sea level has left some of these old shore deposits at elevations as great as 180 feet above it. All domestic rutile produced in 1954 and large quantities of ilmenite came from these sand deposits in Florida (*38*, *39*, *42*).

Black-sand deposits that contain ilmenite occur in streams draining the west and southwest flanks of the Idaho batholith of central Idaho. Initial recovery of ilmenite in Idaho began in 1951 as a byproduct from monazite operations. A small quantity of this ilmenite was shipped for nontitanium uses in 1953 and the remainder stockpiled awaiting development of local markets.

India was the major foreign supplier of ilmenite and Australia the only supplier of rutile to the United States in 1954. Before World War II India was supplying United States consumers with most of their needs. War disruption of seaborne trade gave impetus to the search for alternate supplies, with the result that the United States in 1953 had the largest single ilmenite-producing mine in the world at Tahawus, N. Y., and that other new domestic properties were developed, considerably reducing reliance on foreign sources.

GOVERNMENT MINERAL ASSISTANCE

Rutile was in short supply in 1952, and increased requirements for defense purposes indicated a serious domestic supply deficit. Various actions subsequently were taken by the Government in an effort to increase rutile supplies. On February 4, 1952, the Government approved the inclusion of the titanium minerals, rutile and brookite, in an exploration program whereby the Government contributed 75 percent of the cost of exploring for these minerals. Rutile and brookite were removed from the exploration program May 15, 1953; however, they were restored March 23, 1954. A withdrawal of 1,000 tons of rutile from the National Stockpile was authorized early in 1952 to partly relieve the immediate shortage. In 1952 the Government set up a program for increasing rutile supplies from domestic and foreign sources. Under this program the National Lead Co. considered expansion of its rutile mines in Florida and Australia and utilization of brookite from the Christy property in Hot Spring County, Ark. Removal of the price ceiling by the Government on January 18, 1952, permitted domestic consumers and dealers to compete with other countries in obtaining increased quantities of Australian rutile. Inventory restrictions placed on rutile consumers by the Government on December 29, 1950, were revoked March 12, 1953. The Bureau of Mines consumption canvass for rutile was increased from an annual to a quarterly basis for 1 year (1952–53) to provide more current information on rutile supplies, consumption, and stocks to assist other Government agencies. No Government assistance was available for exploration or mining of ilmenite as of 1954.

MINING AND BENEFICIATION

Open-pit and dredging mining methods were employed in producing titanium minerals commercially in 1954. Ilmenite produced from rock deposits at Tahawus, N. Y. (National Lead Co.), and Piney River, Va. (American Cyanamid Co.), were mined by open-pit methods. Open-pit methods were also employed in recovering ilmenite and rutile from rock deposits near Roseland, Va. (American Rutile Co.), until 1949; Magnet Cove, Ark. (Magnet Cove Rutile Co.), where rutile was produced up to 1944; and Richland Cove, N. C. (Yadkin Mica & Ilmenite Co.), where output of ilmenite ceased in October 1952. Domestic sand deposits were mined for titanium minerals in 1954 by the Humphreys Gold Corp., Denver, Colo., for E. I. du Pont de Nemours Co., Inc., at Starke, Fla.; National Lead Co. and Rutile Mining Co. of Florida at Jacksonville, Fla.; and the Florida Ore Processing Co. at Melbourne, Fla. Ilmenite in alluvial deposits was recovered in 1954 from monazite operations by Baumhoff-Marshall, Inc., Boise, Idaho (production of Baumhoff-Marshall, Inc., included output of the Idaho-Canadian Dredging Co. and Warren Dredging Corp., both located at Boise, Idaho). A large deposit of titaniferous ore was mined also in 1954 in the Allard Lake region, Quebec, Canada, by the Quebec Iron & Titanium Corp., a subsidiary of the Kennecott Copper Corp., and the New Jersey Zinc Co.; this ilmenite-hematite-bearing ore was smelted by the corporation at Sorel, Canada, producing a high-grade titanium slag and a marketable iron product.

The recovery of titanium minerals from a rock deposit (New York) and sand deposits (Florida) is described as follows:

ROCK DEPOSIT

Open-pit mining, with churn drills for blastholes, electric shovels for loading, and diesel trucks for hauling, was employed. Ore containing ilmenite, magnetite, and gangue as the major constituents assayed about 16 percent TiO_2 and 34 percent Fe. After primary blasting a secondary breakage by the use of a drop-ball crane reduced large pieces of ore for delivery to the crushing unit. The ore up to 4 feet in 1 dimension was crushed in 3 stages—primary, jaw crusher yielding an 8-inch product; secondary, cone crusher having a 1½-inch opening; and tertiary, cone crushers producing a product of minus-7/16-inch square-mesh. The minus-7/16-inch product was fed to rodmills in closed circuit with 28-mesh vibrating screens. The minus-28-mesh rodmill product treated on Crockett magnetic separators produced a magnetite concentrate assaying about 10 percent TiO_2 and 58 percent Fe. This material was sintered and sold to blast-furnace operators. A large part of the TiO_2 in the magnetite product is in solid solution in the magnetite mineral and hence is not recoverable as TiO_2.

The tailing from the magnetite separation, consisting of ilmenite and gangue, was separated into coarse and fine fractions. The coarse fraction was tabled to produce a rough ilmenite concentrate, which was re-treated on Wetherill separators to effect additional gangue rejection, giving a final product assaying 44.5 percent TiO_2. The fine fraction of the Crockett tailing was treated by flotation to produce an additional ilmenite product. The ilmenite concentrate was consumed chiefly in titanium-pigment production. Mechanical beneficiation methods employed on titanium ore mined in Virginia and North Carolina varied from that used in New York owing to differences in the composition of the material (*16*).

SAND DEPOSITS

The successful exploitation of Florida sand deposits, containing about 4 percent heavy minerals as compared to Indian sands containing 75 percent heavy minerals, is possible because cheap mining by dredging can be employed and rapid concentration is obtained by using the Humphreys-spiral concentrator (*9, 10*).

A modified core drill, developed in cooperation with the Bureau of Mines, E. I. du Pont de Nemours Co., Inc., and Humphreys Spiral Corp. of Denver; was used in exploring the sand deposits in Florida. Bureau of Mines drilling investigations disclosed an important heavy-sand concentration at Trail Ridge (*38*, *39*, *42*). Subsequently, this area was mined by Du Pont for titanium minerals and in 1954 was the second largest section producing ilmenite in the United States. Deposits mined in Florida contained an average of about 4 percent heavy minerals, of which about 40 percent is ilmenite, 4 percent leucoxene, 7 percent rutile, 11 percent zircon, and less than 0.5 percent monazite. The proportions vary from one deposit to another. The sands were mined by a floating suction dredge, which delivered the material into regulating feed tanks at a concentrator on shore. Humphreys spirals were used in rougher, cleaner, and recleaner stages (one producer employed a mobile unit known as a Cannon separator) to produce a heavy-mineral concentrate, which was dewatered and dried, then treated electrostatically to produce a bulk titanium-mineral product and a tailing containing zircon, monazite, and staurolite. These minerals are recovered as byproducts of titanium operations and shipped for commercial use. A market for staurolite from the Trail Ridge deposit was found in the cement industry. The bulk titanium-mineral product was separated on induced-roll magnetic machines into (1) a finished ilmenite product assaying around 60 percent TiO_2 and (2) a rutile or high-grade ilmenite product which required some further electrostatic cleaning before shipment. In 1954 rutile and monazite were not concentrated as separate products from the Trail Ridge deposit because of their low concentration.

SMELTING

A general description of the process used on Allard Lake titaniferous ore at the Quebec Iron & Titanium Corp. smelting project at Sorel, Canada, was reported in a patent issued to that firm (Pierce, W. M., and others, Titaniferous Material for Producing Titanium Dioxide: United States Patent 2,476,453, July 19, 1949). The process marked a new approach to the smelting of titaniferous ore. Previously the titanium and iron have been separated by a combination of ore dressing and normal smelting processes. Large quantities of flux were added to produce fluid slags and permit separation from the iron. The process employed by Quebec Iron & Titanium Corp. utilized little or no flux to give a titanium-rich slag of about 70 percent TiO_2, suitable for further processing, and also a marketable iron product. A proportioned charge of ore, coal, and 0 to 10 percent of lime for fluxing is smelted at 1,500° to 1,700° C. in a stationary, box-shape electric furnace. Low-ash coal, ranging from 8 to 14 percent of ore weight, slightly less than stoichiometric requirements to reduce the iron oxide and to carburize the iron, is included in the charge. Ores reported to be suitable for producing titanium-slag concentrate are those with a titanium dioxide content of 30 to 50 percent, iron 30 to 50 percent, and gangue up to 12 percent. Original plans called for the ultimate annual production of 250,000 tons of titanium slag and 175,000 tons of iron from 550,000 tons of ore (*23*).

Data on operations at this plant from 1951 to 1954 are shown as follows, in long tons:

	1951	1952	1953	1954
Ore mined and crushed	339, 224	237, 249	141, 266	275, 870
Ore treated	44, 299	93, 005	297, 199	252, 457
Iron and steel produced	12, 877	28, 948	95, 424	80, 859
Titanium slag produced	17, 259	37, 626	126, 681	109, 786
Titanium slag shipped	7, 179	34, 739	129, 823	106, 511

Relatively large-scale tests were conducted successfully in 1947 by the Bureau of Mines in cooperation with the National Lead Co. on the removal of iron from ilmenite by electric smelting. Approximately 15 tons of slag containing 65 to 67 percent TiO_2 was produced from ore containing 30 to 45 percent TiO_2, coke, and small quantities of calcium and magnesium fluxes, yielding a material suitable for the production of titanium dioxide pigment. The iron content of the ore which, in chemical treatment, is converted to cheap ferrous sulfate, was obtained as a pig iron of better potential market and strategic value. Earlier thermodynamic studies by the Bureau on selective reduction of iron in ilmenite by hydrogen, carbon monoxide, and carbon had indicated that carbon was the only effective reducing agent (*37*, *40*).

Bureau of Mines laboratory investigations, supplemented by pilot-plant tests, in 1950 developed a process for treating titaniferous ores from Tahawus, N. Y., and Woonsocket, R. I., by sintering with carbon and soda ash, grinding, separating the metallic iron magnetically, and treating the nonmagnetic fractions for recovery of titania, soda, and byproducts. Before this work an investigation was conducted on the smelting of vanadium-bearing titaniferous sinter in an experimental blast furnace to recover vanadium, and small-scale tests were made by the Bureau on the selective reduction of iron in titaniferous iron ores (*27*, *28*, *47*).

PROCESSING

MANUFACTURED TITANIUM DIOXIDE

Titanium oxide of pigment quality was manufactured in 1954, principally by attacking finely ground ilmenite with concentrated sulfuric acid. All ferric iron present must be reduced with iron scrap, so that the iron sulfate formed is all in the ferrous state. The titanium was dissolved in water as a basic titanium sulfate, which, after selective crystallization of the ferrous sulfate, was precipitated by hydrolysis, filtered, calcined to the oxide, and ground to pigment size. The original titanium dioxide pigments were of the anatase crystal modification; but in 1941 rutile-type pigments having greater opacity and resistance to chalking and fading were placed on the market. The rutile pigments were also produced from sulfate solutions by using nucleating agents. Chloride and fluoride processes that yield rutile directly have been developed through the pilot-plant stage, and Du Pont was producing titanium dioxide by oxidation of titanium chloride.

Manufacture of titanium dioxide pigment requires a costly plant, special location, plenty of water, large steam facilities, and large supplies of ilmenite and sulfuric acid. Before World War II it was estimated that the plant cost for each ton of daily production would be $100,000; by 1954 the cost had doubled. The location of the plant is highly important because of the need for ample facilities to dispose of waste products, such as iron sulfate, weak acids, and ilmenite residues. The operation also causes considerable air pollution. A location as far as possible from residential areas and with ample facilities to dispose of highly acid residues is very important (*31*).

TITANIUM METAL

Two methods were known in 1954 for producing titanium that was ductile at room temperature. The first, developed by Van Arkel, DeBoer, and Fast, involves decomposition of titanium iodide on a hot surface to yield titanium and iodine. The process (essentially known as one of purification) required crude titanium metal as a raw material. This method did not appear to lend itself readily to commercial production because of the low pressures, high-cost reagents, and low recoveries involved. Very high purity metal was produced, however, and the softest material reported has been obtained by this method.

The second method for producing ductile titanium is the reduction of titanium halides with active metal, such as sodium, calcium, or magnesium. Following laboratory studies by Wilhelm J. Kroll, the Bureau of Mines explored this field and constructed the first semiworks equipment for process evaluation.

Commercial production of ductile titanium metal in 1954 was based on modifications of the Kroll process. Titanium tetrachloride, produced by chlorination of titanium ores, was reduced to the metallic state with magnesium in an inert atmosphere of helium or argon. Most of the magnesium chloride formed during the reaction was drained to prevent slowing of the reaction rate. The remaining magnesium chloride and excess magnesium were removed from the spongelike titanium by leaching or vacuum distillation.

The titanium sponge was consolidated by melting in induction or electric-arc furnaces. An electric-arc furnace capable of casting a 2-ton ingot, utilizing either titanium sponge or scrap, was developed in 1954. The furnace used a vacuum rather than inert atmosphere and a double melting technique. Reportedly, the advantages of the vacuum-melting procedure are listed as follows:

(1) More stable electrical arcs.
(2) Ability to accommodate greater power inputs.
(3) Production of smooth ingots requiring little or no conditioning prior to continuing mill conversion into mill products.
(4) Maximum removal of hydrogen.
(5) Lower costs for ingot production.

In the double melting process, titanium sponge or scrap was pressed to produce a consumable titanium electrode. The pressed electrodes were joined by welding and melted in a water-cooled crucible to produce an ingot. First-stage ingots were joined by welding to form a second-stage electrode that was melted to form a second-stage ingot. Automatic arc control was employed in both stages of melting.

Studies by the Bureau of Mines have developed procedures for consolidating titanium powder into solid, nonporous metal and developed methods of heat treatment, welding, stretch forming, and fabrication. This research has also provided fundamental information on the corrosion properties of ductile titanium metal and some of its alloy systems (*4*, *24*, *26*, *35*).

The principal raw materials required for producing ductile titanium metal by the Kroll process are titanium ore (rutile, titanium slag, or ilmenite), carbon, magnesium, chlorine, electric energy, inert gas (helium or argon), and fuel. Major requirements per pound of titanium sponge are 4 to 5 pounds of ilmenite (or rutile or titanium slag of equivalent titanium content), 5 pounds of chlorine, 1.25 pounds of magnesium, and 0.9 cubic foot of inert gas. If the magnesium and chlorine are regenerated the requirement for magnesium is about 0.2 pound and for chlorine about 1 pound. Power requirements vary within industry and may range from 7 to 19 kw.-hr. per pound of sponge. The latter electrical energy includes that consumed for regenerating magnesium and chlorine.

Plant costs for producing ductile titanium metal, under the 1954 technology and small-scale commercial production, were very high. Plant costs under Government titanium-production contracts ranged from about $15 million for a 3,600-short-ton-per-year facility to $31.5 million for a 7,500-ton-per-year facility. Government contracts are discussed under the section Development of the Titanium Industry.

Titanium powder, 96 to 98+ percent titanium, was produced commercially in 1954 by reduction of titanium dioxide with calcium hydride to yield titanium hydride, which was heated to drive off hydrogen and produce titanium. This metal, as produced in 1954, was not of ductile quality. Titanium powder was also produced by electropurification of titanium scrap.

Many other processes for producing titanium metal were being considered in 1954; these include electrolytic and vapor-phase reduction, use of sodium, mixtures of sodium and magnesium as reducing agents, and improved iodide-decomposition processes. A contract for erecting and operating a pilot plant to study an electrolytic process for producing titanium metal was signed on July 2, 1954, by the Government and Horizons Titanium Corp.

FERROTITANIUM ALLOYS

The ferroalloys were divided arbitrarily into three groups in 1954, according to their carbon content. High-carbon ferrotitanium (Ti 15 to 18 percent, C 6 to 8 percent) was made by reducing ilmenite with coal or coke in a single-electrode arc furnace. The molten alloy, tapped from the furnace onto a cast-iron bed, formed thin ingots, which were subsequently crushed. The medium-carbon alloy (Ti 17 to 21 percent, C 3 to 4.5 percent), produced in similar manner, was subjected to an additional refining operation during which some of the carbon was oxidized by application of additional titanium oxide ore.

Low-carbon ferrotitanium was nominally specified as containing Ti 20 to 25 percent, Al 3.5 percent maximum, Si 4 percent maximum, and C 0.10 percent maximum. The low-carbon alloys were made by aluminothermic reduction of iron and titanium oxide minerals. It was relatively difficult to produce ferrotitanium by this method because of the high stability of some of the lower oxides of titanium. For this reason excess aluminum was required for the reduction, and the alloys contained appreciable quantities of this element; however, the residual aluminum was advantageous in protecting the titanium from oxidation when the alloy was added to molten steel. In 1954 the aluminothermic process seemed to be the only practicable method for making low-carbon, low-silicon titanium alloys. When silicon was used as a reducing agent the resulting alloys usually contained almost as much silicon as titanium and sometimes even more.

WELDING-ROD COATINGS

The ability of titanium and its compounds to stabilize an electric arc and to overcome the tendency to sputter and flicker has been known for many years, and practical application has been made of this property in the manufacture of fluxes and coatings for improved electric welding rods. Titanium dioxide also reduces the viscosity of the slag formed and greatly reduces the surface tension of the metal droplets on the electrodes, thus yielding small droplets that give more satisfactory welds.

In general, rutile or other natural minerals, such as ilmenite, are employed as a source of the oxide because of low cost, but some applications demand an arc stabilizer of a fineness attainable only by using the chemically precipitated oxide.

Some manufacturers have replaced dip coating of welding rods with extruding machines. Many compositions containing rutile, ilmenite, manufactured TiO_2, titanium slag, or combinations of these materials have been recommended as flux coatings of metallic rods for welding electrodes; however, each manufacturer has developed his own particular formula, which was not made available to the industry.

PRODUCTION AND CONSUMPTION

Production of ilmenite and rutile concentrates from domestic ores from 1940 to 1954 is shown in table 1. During this period ilmenite production established a high record in 1951. Preliminary data indicated that the high record was exceeded slightly in 1954. Factors that contributed to lower production of ilmenite in 1952 and 1953 were inactivity of the Yadkin Mica & Ilmenite Co., Finley, N. C., which ceased operations in October 1952, and increased imports of Canadian titanium slag. Of the estimated production of 546,000 short tons of ilmenite in 1954, New York supplied over half and Florida about a third; the remainder came from Virginia and Idaho. A small quantity of ilmenite produced in California and Idaho for nontitanium use was not included in the production and shipments of titanium concentrates.

Rutile production in the United States more than doubled during World War II and showed no significant increases thereafter. All 1954 domestic production of rutile came from Florida.

TABLE 1.—*Production and mine shipments of titanium concentrates from domestic ores in the United States, 1940–54* [1]

[Short tons]

Year	Ilmenite				Rutile			
	Production	Shipments			Production	Shipments		
		Gross weight	TiO_2 content	Value		Gross weight	TiO_2 content	Value
1940	20, 668	20, 702	9, 505	$183, 686	2, 888	2, 657	2, 475	$393, 243
1941	23, 297	21, 526	9, 930	196, 522	3, 130	3, 431	3, 192	493, 782
1942	77, 208	93, 397	41, 328	1, 805, 823	2, 648	2, 649	2, 466	410, 956
1943	203, 551	211, 715	94, 283	3, 738, 970	3, 987	3, 941	3, 639	610, 879
1944	278, 610	280, 791	128, 095	7, 371, 279	6, 922	6, 770	6, 312	1, 088, 112
1945	308, 516	308, 518	141, 852	7, 359, 170	7, 179	6, 837	6, 414	869, 920
1946	282, 447	282, 708	130, 624	4, 878, 917	7, 453	7, 514	7, 046	996, 989
1947	336, 533	336, 061	157, 328	5, 029, 490	8, 562	5, 157	4, 813	533, 548
1948	383, 745	381, 508	177, 447	5, 793, 973	7, 380	9, 907	9, 226	647, 334
1949	402, 334	389, 234	186, 535	6, 212, 348	11, 988	10, 559	9, 414	489, 798
1950	468, 320	452, 370	230, 826	5, 606, 584	(2)	(2)	(2)	(2)
1951	535, 835	510, 840	261, 982	7, 689, 272	(2)	(2)	(2)	(2)
1952	528, 588	522, 515	265, 596	8, 022, 752	(2)	(2)	(2)	(2)
1953	513, 696	512, 176	258, 247	7, 222, 641	6, 825	6, 476	6, 043	702, 791
1954 [3]	546, 500	528, 900	267, 200	8, 500, 000	6, 800	6, 500	6, 100	720, 000

[1] A mixed product containing altered ilmenite, rutile, and leucoxene was reported under rutile in 1949 and thereafter—from 1950–54—under ilmenite.

[2] Bureau of Mines not at liberty to publish.

[3] Estimated figures (based on a 10-month actual figure and an estimate for November and December 1954).

World production figures on titanium concentrates from 1940 to 1953 are listed in table 2. Domestic production of ilmenite in 1953 represented about 47 percent of the total estimated world production. During World War II the United States replaced India as the largest ilmenite producer and had retained this position as of 1953. Australia, the major producer of rutile, supplied about 86 percent of the estimated world production of rutile in 1953. The United States was listed as the second largest producer of rutile concentrates from 1944 to 1953. Canada was the only producer of titanium slag (1950–54).

TABLE 2.—*World production of titanium concentrates, by countries, 1940–53*

[Metric tons]

(Compiled by Berenice B. Mitchell)

Country	1940	1941	1942	1943	1944	1945	1946	1947	1948	1949	1950	1951	1952	1953
Ilmenite:														
Australia [1]	1, 538	3, 779	4, 588	5, 470	7, 287	8, 494	6, 716	10, 475	11, 756	9, 884	12, 417	12, 091	47	(2)
Brazil	[3] 12	[3] 4, 471	---------	---------	[3] 3, 250	[3] 5, 000	---------	---------	[3] 7, 900	650	---------	[4] 24	---------	---------
Canada	4, 114	11, 477	9, 100	62, 992	30, 820	12, 834	1, 275	6, 445	4, 029	490	[5] 3, 177	[5] 19, 235	[5] 38, 276	[5] 132, 940
Egypt	465	2	[6] 691	---------	9	46	146	---------	1, 601	635	260	317	1, 998	765
India	267, 376	131, 111	49, 977	38, 396	102, 412	174, 848	187, 993	265, 143	233, 098	313, 126	216, 076	227, 681	228, 505	[3] 263, 649
Malaya	[3] 2, 596	[3] 44	---------	---------	---------	---------	---------	[3] 13, 291	12, 909	20, 034	25, 315	[3] 44, 191	[3] 22, 046	[3] 26, 996
Norway	51, 700	61, 086	60, 713	66, 191	63, 975	28, 312	52, 574	69, 711	90, 017	99, 013	105, 150	105, 150	118, 270	90, 000
Portugal	899	798	---------	121	---------	301	633	243	155	919	66	169	432	462
Senegal [7]	7, 082	1, 000	4, 840	730	---------	3, 200	4, 191	11, 282	3, 690	8, 338	540	2, 500	4, 622	3, 500
Spain	---------	71	85	178	548	216	128	150	181	376	637	700	1, 279	1, 591
United States	18, 750	21, 135	70, 042	184, 657	252, 749	279, 880	256, 230	305, 296	348, 126	364, 989	424, 851	486, 099	479, 524	466, 015
Total	354, 532	234, 974	200, 036	358, 735	461, 050	513, 000	509, 886	682, 036	713, 500	818, 500	788, 500	898, 100	895, 000	986, 000
Rutile:														
Australia [8]	1, 643	3, 816	5, 503	6, 730	8, 843	9, 901	8, 283	13, 406	15, 348	13, 958	17, 985	35, 534	38, 624	[8] 40, 000
Brazil	[3] 499	[3] 2, 369	[3] 4, 615	[3] 4, 557	[3] 1, 564	[3] 160	[3] 28	[3] 5	---------	---------	(9)	---------	---------	---------
Fr. Cameroon	400	1, 800	2, 400	2, 735	3, 320	1, 440	1, 260	755	576	403	25	106	294	53
India	934	1, 891	2, 295	2, 396	1, 672	620	262	160	129	---------	(9)	(8)	---------	---------
Norway	156	172	77	116	85	76	63	51	---------	16	(9)	(9)	43	(8)
United States	2, 620	2, 839	2, 402	3, 617	6, 279	6, 513	6, 761	7, 767	6, 695	10, 875	(10)	(10)	(10)	6, 192
Total	6, 252	12, 887	17, 292	20, 151	21, 763	18, 710	16, 657	22, 144	22, 700	25, 300	[11] 25, 300	[11] 42, 000	47, 000	[8] 46, 300

[1] Estimated ilmenite content of all ilmenite-bearing concentrates.

[2] Data not available; estimate included in total.

[3] Exports.

[4] Estimate.

[5] Includes titanium slag containing approximately 70 percent TiO_2.

[6] Includes 26 tons of garnet-ilmenite.

[7] Approximately 20 percent of ilmenite concentrates is zircon.

[8] Estimated rutile content of all rutile-bearing concentrates.

[9] Figure withheld in order to avoid disclosure of United States production by differences; see footnote 10.

[10] Figure withheld in order to avoid disclosure of individual company operation.

[11] Includes French Equatorial Africa in 1950 (6 metric tons), 1951 (estimate included in total); Senegal 1951 (figure withheld in order to avoid disclosure of individual company operation in United States by differences).

Annual figures on the domestic consumption of titanium concentrates from 1940 to 1953, inclusive, are given in table 3; a breakdown by uses is given in table 4. Titanium slag, obtained by electric-furnace reduction of ilmenite-hematite ore, was consumed chiefly in 1953 in the production of titanium pigments, with minor quantities used in welding-rod coatings and for experimental purposes. Initial commercial consumption of titanium slag was reported in 1952. Ilmenite consumption increased steadily from 1940 to 1951 and, unlike rutile and many other materials, did not follow the general decreased usage pattern immediately after World War II. About 99 percent of the ilmenite consumed in 1953 went into the manufacture of titanium pigments which, because of their outstanding characteristics, are replacing other white pigments; rutile was used mainly for welding-rod coatings, alloys, and carbide, with smaller quantities consumed for titanium metal and fiberglass. A mixed product containing ilmenite, rutile, and leucoxene, classified as ilmenite and averaging about 66 percent titanium dioxide, was also consumed in the expanding titanium-metal industry in 1953.

TABLE 3.—*Consumption of titanium concentrates* [1] *in the United States, 1940–53*

[Short tons]

Year	Ilmenite		Titanium slag [2]		Rutile	
	Gross weight	Estimated TiO_2 content	Gross weight	Estimated TiO_2 content	Gross weight	Estimated TiO_2 content
1940	[3] 250, 000	[3] 137, 250			[4] 4, 800	[4] 4, 512
1941	275, 106	150, 966			6, 361	5, 986
1942	257, 535	141, 412			10, 616	9, 952
1943	302, 822	142, 868			17, 634	16, 451
1944	360, 941	175, 475			14, 813	13, 837
1945	381, 178	187, 580			9, 791	9, 144
1946	404, 283	202, 663			7, 134	6, 670
1947	479, 524	250, 859			7, 692	7, 083
1948	565, 000	300, 408			10, 230	9, 488
1949	510, 608	268, 000			11, 888	10, 863
1950	679, 244	351, 675			11, 721	10, 869
1951	713, 363	373, 037			17, 227	16, 019
1952	682, 850	351, 553	24, 236	16, 746	18, 317	17, 353
1953	695, 544	358, 840	73, 528	52, 511	20, 170	19, 033

[1] A mixed product containing altered ilmenite, leucoxene, and rutile, used to make pigments and metal, was reported under rutile in 1949 and thereafter, from 1950–53, under ilmenite.

[2] Initial commercial consumption was reported in 1952.

[3] Estimated.

[4] Apparent consumption.

TABLE 4.—*Consumption of titanium concentrates in the United States in 1953, by products*

[Short tons]

Product	Ilmenite		Titanium slag		Rutile	
	Gross weight	Estimated TiO_2 content	Gross weight	Estimated TiO_2 content	Gross weight	Estimated TiO_2 content
Pigments (mfg. TiO_2)[1] [2]	684, 707	353, 354	73, 324	52, 368	(3)	(3)
Welding-rod coatings [2]	990	584			10, 476	9, 812
Alloys and carbide	9, 823	4, 888			4, 000	3, 821
Ceramics	5	3			317	295
Miscellaneous	19	11	[4] 204	[4] 143	[5] 5, 377	[5] 5, 105
Total	695, 544	358, 840	73, 528	52, 511	20, 170	19, 033

[1] Includes a mixed product containing altered ilmenite, leucoxene, and rutile used to make pigments and metal.

[2] "Pigments" include all manufactured titanium dioxide, consumption of which in welding-rod coatings was 1,770 tons in 1951, 2,209 tons in 1952, and 1,986 tons in 1953.

[3] Included with "Miscellaneous" to avoid disclosure of individual company operations.

[4] Includes consumption for welding-rod coatings and research purposes.

[5] Includes consumption for chemicals, metal, and fiberglass.

The titanium-pigment industry, which included 4 companies with 7 plants and 5 mines, was concentrated in the eastern part of the United States. Titanium pigments were produced in 1954 by the following concerns:

Company:	*Plants*	*Mines*
American Cyanamid Co., Pigments Division	Gloucester City, N. J. Piney River, Va.	Piney River, Va.
Glidden Co., Chemicals-Pigments-Metals Division.	Baltimore, Md.	Finley, Caldwell County, N. C.
E. I. du Pont de Nemours Co., Inc.	Baltimore, Md. Edge Moor, Del.	Trail Ridge mine, Starke, Fla.
National Lead Co.	St. Louis, Mo. Sayreville, N. J.	Tahawus, N. Y. Jacksonville, Fla.

A goal of 370,000 short tons of titanium pigments per year by January 1, 1954, was established by the Government on March 10, 1952. Tax-amortization certificates were issued to assist in constructing added pigment facilities.

The American Cyanamid Co. announced plans in July 1953 for constructing a $14 million titanium-pigment plant at Savannah, Ga. Completion of the new plant was scheduled for 1955. In July 1954 Glidden planned construction of a titanium-pigment plant (rated about 18,000 tons a year) at Hawkins Point, Md., and completion by the end of 1955. The cost of this facility was estimated at $10 million. An announcement issued in July 1954 by the National Lead Co. stated that the company will erect new facilities at St. Louis, Mo., for an additional capacity of about 36,000 tons of titanium dioxide a year. Construction that began in mid-1954 was expected to be completed by July 1955.

Welding-rod coatings using rutile, ilmenite, manufactured titanium dioxide, titanium slag, or combinations of these materials were produced by about 25 companies in 1954. About half of these plants were concentrated in Ohio and Pennsylvania, and the remainder were in Illinois, California, Wisconsin, Maryland, Tennessee, Minnesota, Indiana, and Michigan.

Titanium alloys and carbides were produced by eight companies in 1954 in New York, Pennsylvania, New Jersey, Massachusetts, and Connecticut.

Ceramic producers consuming titanium concentrates in 1954 were scattered throughout the United States. In many instances only a few pounds were used, and as the small producers changed locations frequently it was difficult to obtain a precise figure on the number of producers.

Ductile titanium metal was produced commercially in July 1954, by E. I. du Pont de Nemours Co., Inc., Newport, Del.; Titanium Metals Corp. of America, Henderson, Nev.; Dow Chemical Co., Midland, Mich.; and the Federal Bureau of Mines, Boulder City, Nev. A Government contract with Cramet, Inc., Chattanooga, Tenn., signed in July 1953, called for initial production by the fourth quarter of 1954. Commercial production of titanium-sponge metal, based on the process developed by the Bureau of Mines, was begun in 1948 and increased from about 10 short tons to 25 in 1949, 75 tons in 1950, 495 tons in 1951, 1,075 tons in 1952, 2,241 tons in 1953, and 5,370 tons in 1954.

STOCKPILING

Rutile was the only titanium mineral included in the National Stockpile List of Strategic and Critical Materials, in 1954. No other titanium minerals have been purchased by the Government as of 1954 for stockpiling purposes. Rutile was placed in group I of the Strategic and Critical Materials List for Stockpiling on November 20, 1944; this list includes those materials acquired either by purchasing or by transfer of Government-owned surpluses. On July 6, 1950, rutile was transferred to group II of the stockpiling listing, being acquired only through transfer of Government-owned surpluses; as of the end of 1954 it remained under this grouping. Titanium sponge metal was placed on the Government list of strategic and critical materials on July 16, 1954.

FOREIGN TRADE[4]

IMPORTS

Imports of titanium concentrates in the United States from 1940 to 1953 are given in table 5. India remained the dominant source of ilmenite imports into the United States from 1940 to 1953, except in 1942 and 1943. During these 2 years Canada supplied the major portion of ilmenite. Ilmenite shipped into the United States totaled 2,788,221 short tons from 1940 to 1953. Imports dropped from 170,689 tons in 1941 to 10,407 tons in 1942, as shipping lanes from India became imperiled by submarine warfare; however, total ilmenite imports increased in 1943 by 67,686 short tons over the 1942 total, as a result of substantially greater ilmenite shipments from Canada. The largest quantity of ilmenite shipped into the United States was in 1949, when 324,157 short tons was imported. Ilmenite imports from Canada in 1953 included chiefly titanium slag. The Quebec Iron & Titanium Corp. began production of titanium slag in October 1950; however, titanium slag was not shipped into the United States until 1951.

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[4] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

TABLE 5.—*Titanium concentrates*[1] *imported for consumption in the United States, 1940–53, by countries of origin*

[Short tons]

(U. S. Department of Commerce)

Country	1940	1941	1942	1943	1944	1945	1946	1947	1948	1949	1950	1951	1952	1953
Ilmenite:														
Australia[2]	2	364	235	390	79	1,753	----	1,659	(3)	----	112	100	----	54
Brazil	2	3,709	1,287	----	5,511	10,508	2	1	8,708	----	----	1	----	----
Canada	----	5,725	4,540	65,437	32,580	6,987	1,250	7,122	4,519	540	1,357	[4] 3,776	[4] 38,451	[4] 139,585
India	221,637	156,079	1,165	8,960	62,066	179,693	218,623	262,503	184,309	289,739	187,834	185,145	145,562	147,005
Norway	----	----	----	----	----	9,895	21,077	30,026	41,248	33,155	27,155	----	----	----
Other[5]	----	969	----	----	4,648	----	----	----	3,335	723	1	56	----	----
Total "as reported"	221,641	166,846	7,227	74,787	104,884	208,836	240,952	301,311	242,119	324,157	216,459	189,078	184,013	286,644
Australia: In zirconium ore[2]	2,250	3,843	3,180	3,306	4,064	[6] 1,236	1,388	----	----	----	----	----	----	----
Grand total	223,891	170,689	10,407	78,093	108,948	210,072	242,340	301,311	242,119	324,157	216,459	189,078	184,013	286,644
Value of "as reported" dollars	750,590	629,940	60,490	380,161	596,034	1,217,339	1,440,112	1,791,020	1,758,848	2,479,071	1,198,545	1,323,438	2,478,077	5,463,526
Rutile:														
Australia[2]	22	666	1,311	2,802	1,896	3,070	4,377	7,460	8,771	3,085	3,427	11,023	19,394	16,098
Brazil	128	2,448	4,966	4,920	1,669	234	31	----	----	----	----	----	----	----
French Cameroon	----	----	[7] 146	[7] 1,095	----	----	----	3	----	----	----	----	----	----
Other[8]	6	----	----	818	134	----	----	113	(3)	----	----	----	----	----
Total "as reported"	156	3,114	6,423	9,635	3,699	3,304	4,408	7,576	8,771	3,085	3,427	11,023	19,394	16,098
Australia:[2]														
In zirconium ore	2,008	3,117	4,102	4,703	6,320	7,298	1,456	----	----	1,096	1,133	210	156	84
In ilmenite ore	----	----	----	----	----	----	----	5,061	----	----	----	----	----	----
Grand total	2,164	6,231	10,525	14,338	10,019	10,602	5,864	12,637	8,771	4,181	4,560	11,233	19,550	16,182
Value of "as reported" dollars	14,849	306,176	623,917	823,624	272,283	98,170	213,795	468,810	588,713	179,746	149,733	491,383	1,728,803	1,791,494

[1] Classified as "ore" by the U. S. Department of Commerce.

[2] Most of the imports of titanium from Australia in 1940–47 were in mixed zircon-rutile-ilmenite concentrates which were included as ilmenite, rutile, and zirconium ore in U. S. Department of Commerce figures. The quantities reported by the U. S. Department of Commerce have been adjusted to reflect percentage content of each item based on reports to the Bureau of Mines from importers.

[3] Less than 0.5 ton.

[4] Includes titanium slag containing approximately 70 percent TiO_2.

[5] Includes Portugal 1941 (969); Ceylon 1944 (4,648), 1949 (2); Malaya 1948 (3,335), 1950 (56); Egypt 1949 (721); France 1950 (1).

[6] Includes 309 tons not recovered from mixed concentrates.

[7] Includes quantities reported by the U. S. Department of Commerce as originating in French Equatorial Africa, from which no rutile production has been recorded.

[8] Includes Portugal 1940 (6); India 1943 (818), 1944 (134), 1947 (113); Norway 1948 (less than 0.5 ton).

Rutile has been imported mainly from Australia and Brazil and also from Australia in the form of mixed concentrates containing zircon and ilmenite. Rutile imports into the United States from 1940 to 1953 totaled 136,917 short tons; the smallest quantity was imported in 1940 and the largest in 1952, totaling 2,164 and 19,550 tons, respectively. Australia, the major rutile-producing country, was the only one that shipped rutile to the United States from 1948 to 1953.

EXPORTS

Titanium products exported from the United States, as shown in table 6, for 1940 to 1953 were as follows, in short tons: Concentrates,

TABLE 6.—*Titanium products exported from the United States, 1940–53, by classes*

(U. S. Department of Commerce)

Years	Concentrates		Metal and alloys in crude form and scrap		Metal, primary forms		Ferroalloys		Dioxide and pigments		Tetrachloride and other compounds	
	Short tons	Value	Short tons	Value	Short tons	Value	Short tons	Value	Short tons	Value	Short tons	Value
1940	(1)	(1)	----	----	----	----	(1)	(1)	4,962	$867,535	(1)	(1)
1941	[2] 432	[2] $70,613	----	----	----	----	[2,3] 150	[2,3] $35,687	7,821	1,609,071	[4] 17	[4] $5,792
1942	618	117,886	----	----	----	----	[3] 422	[3] 60,938	8,870	1,707,433	24	13,666
1943	576	103,947	----	----	----	----	[3] 760	[3] 117,402	9,765	1,830,344	728	442,591
1944	291	51,828	----	----	----	----	[3] 793	[3] 127,145	10,925	1,851,457	375	215,696
1945	609	121,951	----	----	----	----	744	122,887	12,824	2,315,552	75	46,718
1946	1,385	200,866	----	----	----	----	550	63,723	16,314	3,092,607	(5)	(5)
1947	1,266	192,703	----	----	----	----	509	80,590	21,171	5,183,936	(5)	(5)
1948	1,454	187,225	----	----	----	----	480	82,874	26,824	7,126,956	(5)	(5)
1949	1,505	143,412	----	----	----	----	179	40,918	29,621	8,140,991	(5)	(5)
1950	600	57,753	----	----	----	----	171	42,741	32,660	8,799,758	(5)	(5)
1951	646	63,050	----	----	----	----	175	107,718	39,242	13,274,143	(5)	(5)
1952	870	110,737	762	$31,134	3	$38,979	325	88,664	35,664	10,691,698	(5)	(5)
1953	3,233	128,046	1,137	26,148	31	796,301	185	48,722	48,722	11,715,798	(5)	(5)

[1] Data not available.

[2] Data cover last 6 months only.

[3] Includes metal and nonferrous alloys.

[4] Data cover last 4 months only.

[5] Beginning Jan. 1, 1946, not separately classified.

13,485; metal and alloys in crude form and scrap, 1,899; metal primary forms, n. e. c., 33; ferroalloys, 5,443; dioxide and pigments, 296,426; and tetrachloride and other compounds, 1,219. The low figure for tetrachloride and other compounds is due to the fact that no further information was available on these commodities in 1940 and that after January 1, 1946, they were not separately classified. Two new export classifications—titanium metals and alloys in crude form and scrap and titanium metal in primary form—were initiated by the Government in 1952 owing to the increased production of titanium metal. Exports of titanium concentrates consisted chiefly of rutile. Because of the short supply of rutile in the United States the Government placed rutile under control, and a closed quota was established (except for Canada) on all rutile exports beginning January 28, 1952; however, rutile was removed from the control list September 9, 1952.

TARIFF

Commodities included under Tariff Paragraph 89, as established in the Tariff Act of 1930, are "Titanium potassium oxalate, and all compounds and mixtures containing titanium." These were dutiable under the act at 30 percent ad valorem. The tariff on all items included under this paragraph was reduced to 15 percent ad valorem in negotiation with Benelux, at the General Agreement on Tariffs and Trade in Geneva, October 30, 1947. The reduction became effective January 1, 1948.

Ferrotitanium is one of the commodities included under Tariff Paragraph 302–m, dutiable under the act at 25 percent ad valorem. Under two bilateral trade agreements with Canada the tariff on these items was reduced to 15 percent. The reduction became effective January 1, 1936, for ferrotitanium. At the General Agreement on Tariffs and Trade in Geneva the rate was reduced to 12½ percent ad valorem in negotations with Canada. The reduction became effective January 1, 1948.

Titanium metal and alloys with aluminum, chromium, cobalt, copper, manganese, nickel, or silicon are included under Tariff Paragraph 302–n, dutiable under the act at 25 percent ad valorem. At the General Agreement on Tariffs and Trade in Torquay, the following rates were established.

1. Alloys of two or more of the following metals: Titanium, barium, boron, niobium (columbium), strontium, tantalum, thorium, vanadium, zirconium, calcium and uranium: (*a*) Containing uranium but not containing niobium (columbium) or tantalum, 25 percent ad valorem; (*b*) other, 12½ percent ad valorem.
2. Alloys of one or more of the above-named metals with one or more of the metals aluminum, chromium, cobalt, copper, manganese, nickel or silicon: (*a*) Containing uranium, 25 percent ad valorem; (*b*) other, 20 percent ad valorem.
3. Titanium metal, 20 percent ad valorem.

Negotiations in this instance were with Canada. The new rates became effective June 6, 1951. Since World War II all imports of titanium alloys listed above were from Canada.

Titanium ores and concentrates, crude, or not advanced in value or condition, that were duty-free under the Tariff Act of 1922 are also duty-free under the Tariff Act of 1930 (Tariff Paragraph 1719) as "minerals, crude, or not advanced in value or condition by refining or grinding, or by other process of manufacture, n. s. p. f." The duty-free status of titanium ores and concentrates (except ilmenite and ilmenite sand) was bound in the GATT at Geneva, in negotiations with Brazil. The effective date of binding was January 1, 1948. Basic titanium slag, ground or unground, is also on the free list (Tariff Paragraph 1685).

USES

About 99 percent of all the ilmenite consumed in 1953 and in several preceding years has gone into the manufacture of titanium dioxide for pigments. About 75 percent of the titanium pigments was used in the paint, enamel, and lacquer industries, and the remainder went into paper, rubber, floor coverings, coated fabrics, textiles, printing ink, and other miscellaneous commodities. The other 1 percent of ilmenite was used in making titanium alloys, carbide, and welding-rod coatings.

Outstanding properties of titanium dioxide pigments that recommend them for their many varied applications are: High opacity and covering power, chemical inertness, and low specific gravity.

The major use for rutile has been for welding-rod coatings. In 1953, 52 percent of the rutile consumed went into welding-rod coatings and 20 percent was used for alloys and carbide, 2 percent for ceramics, and 26 percent for miscellaneous uses such as fiberglass and titanium metal.

Titanium is a low-density, silver-white metal between silver and stainless steel in color. Metallic titanium owes its importance to an unusual combination of properties—lightness, strength, and resistance to corrosion. The density of titanium is 0.16 pound per cubic inch; it is 60 percent heavier than aluminum but only 56 percent as heavy as alloy steel. Titanium alloys are much stronger than aluminum alloys, having tensile strength and hardness approaching that of many alloy steels. The strength : weight ratio at ordinary temperatures exceeds that of either aluminum or stainless steel. Titanium alloys also have unusual resistance to fatigue and great impact strength. Titanium metal is more resistant to corrosion than aluminum and has shown better corrosion resistance to sea water and marine atmospheres than stainless steel; it has been specified in many applications (particularly those involving resistance to chloride on attack) where stainless steels are rapidly attacked.

The chief disadvantages are high cost, difficulties of fabrication, and excessive reactivity at high temperatures. Although the melting point of titanium—3,038° F.—is extremely high, it absorbs oxygen and nitrogen and becomes brittle with longtime exposure above 1,000° F.

Although titanium metal has been suggested for many applications, its major use in 1954 was chiefly for military purposes. The present and potential applications of titanium metal and its alloys follow:

AIRCRAFT AND JET ENGINES

The largest military market for titanium metal was in the aircraft industry for airframes and power plants. Titanium was destined for use by airframe producers in firewalls, skin, landing-gear components, hydraulic tubing, shrouds, oil and fuel tanks, nacelle structures, fasteners, and engine supports. The major use of titanium in aircraft power plants was in compressor disks and blades; compressor housings, retaining rings, and fasteners were also fabricated. Replacement of other metals by titanium can be found in specific airplanes, such as the Douglas X–3 supersonic airplane, where titanium was utilized for aluminum parts that were exposed to engine heat or heated aerodynamically. Commercially pure titanium was used in North American's jet airplanes for shroud assemblies, cable shrouds, ammunition tracks, and flap-rubbing strips; titanium-alloy material was used for fuselage frames and bulkheads.

Producers of civilian aircraft were also interested in the use of titanium in many of the same applications where the metal can be advantageously employed in military craft. The wider use of jet-powered, high-speed, commercial transport planes will provide a ready market for titanium alloys. Douglas Aircraft was the first to use titanium in a commercial airliner. In its DC–7 titanium was substituted for stainless steel in the engine nacelles and firewalls.

ORDNANCE

The use of titanium in ordnance applications that called for light weight, good strength, and corrosion-resistant metal was investigated by the Army. Titanium was employed in place of steel in mortar base plates and in the manufacture of flash suppressors for artillery use. Titanium alloy's resistance to ballistic penetration was considered greater than that of conventional armorplate of equal weight. Titanium replaced brass in certain primer cups because of its greater resistance to chemical action. The fast-burning quality of titanium powder was considered for use in rockets and in detonating fuse or explosives. Lowering the cost of titanium metal would lead to possible additional uses by the Army, such as bridges, cylinders for compressed gases, airstrip landing mats, field-gun mount trails, and antiaircraft-gun outriggers.

MARINE USES

Titanium metal, with its outstanding resistance to sea water and marine environments, together with its light weight and fatigue resistance, was studied by the Navy for use in salt-water valves for valve seats and disk trims, tubing for condenser systems and heat exchangers, wet exhaust valves for submarine diesel engines, and turbine blades for low-temperature steam turbines. Additional naval uses of titanium, based on the lower cost of the metal, may be found in the manufacture of salt-water pump shafts, small outboard propeller shafts, snorkel tubes, and hull material for small craft, such as PT boats.

CHEMICAL EQUIPMENT

Because of its excellent corrosion resistance, titanium metal appears to have promising applications in the chemical industry, such as handling hypochlorite solutions (commercial bleaching compounds); chlorine dioxide, particularly in the paper-pulp industry; metallic chlorides, particularly ferric; and cupric and organic acids.

OTHER USES

The outstanding characteristics of titanium suggest numerous other uses. These include surgical instruments, orthopedic appliances, portable machine tools, sporting equipment (such as lightweight noncorroding golf clubs, tennis rackets, and fishing rods), food handling and processing equipment, valve parts, X-ray-tube targets, and textile machinery.

Alloys of titanium with iron, particularly ferrocarbon-titanium, were used in steelmaking, iron castings, and various nonferrous alloys. Titanium, formerly used mainly as a deoxidizer, has assumed considerable importance in the manufacture of stainless steels as a carbon- and nitrogen-stabilizing element to inhibit intergranular corrosion. Small quantities of titanium were used to obtain improved copper alloys, aluminum, and aluminum alloys. Titanium was employed as an important constituent of alloys used for making permanent magnets. It forms a series of extremely hard, metallike compounds with carbon, silicon, nitrogen, and boron, which were used in a number of fused or sintered alloys and in abrasives for use in tips for cutting tools and dies and in abrasive stones and wheels; of these the carbide was the most important commercially.

In addition to the above applications the element titanium, with its unique properties, can be also found in the following forms:

(1) Ceramic titanates were finding an increasingly important place in the electronic industry. Barium titanate and solid solutions of barium and strontium titanates have exceptionally high dielectric constants and show remarkable piezoelectric effects. The biggest field of application was in capacitors for television sets. They were also used for phonograph pickups, microphones, and high-frequency sound generators. Titanate ceramic radiators for production of ultrasonic energy may find new uses in application of ultrasonic radiation to chemical processes.

(2) Titanium tetrachloride was used during World War I for producing smoke clouds for concealing and spotting purposes. This chemical also was used for skywriting for advertising purposes.

(3) The use of titanium dioxide as a major opacifying agent in porcelain enamels found increased use for stoves, refrigerators, signs, reflectors, architectural products, sanitary ware, and hollow ware. The chief advantages of titania enamels are their superior opacity or hiding power, extreme hardness, and excellent acid resistance.

(4) Titanium compounds have been used experimentally as fire-retardant agents for fabrics. A titanyl chloride-antimony trichloride complex, found to be effective in providing flame retardance, has been applied to many types of cellulose fabrics.

(5) The alkyl titanates derived from the action of titanium tetrachloride on alcohols were reported to be very effective waterproofing agents. These compounds can impart a water-repellant finish to such diverse materials as paper, cotton, wool, rayons, nylon, silk, felt, and wood. Potential industrial applications depend on commercial availability of these compounds at reasonable cost. Titanium esters, acylates, and chelates were reportedly produced in 1954 in pilot quantities. These titanium organic compounds were stated to be suitable for surface-active agents, adhesives, polymers, and metal protection.

(6) Clear, transparent rutile gems were manufactured in 1954. Large boules were produced by fusing purified titanium dioxide in an electric furnace. Gems cut from these boules exhibit brilliance superior to that of diamonds. Rutile gems are only seven-tenths as hard as diamonds but have an index refraction of 2.7 compared to 2.41 for diamonds, which means a higher degree of internal reflection.

PRICES

The quoted market prices for ilmenite and rutile from 1940 to 1954 are listed in table 7. After World War II the price of ilmenite and rutile decreased, undoubtedly due to a large extent to the increased domestic ilmenite supplies and decreased demand for rutile. Ceiling prices for ilmenite and rutile were frozen under the General Ceiling Price Regulation of January 26, 1951. The increased world demand for Australian rutile raised the price so high that United States concerns were not able to negotiate for rutile at these market prices with the ceiling price frozen at a much lower level. Accordingly, Amendment 13 to General Overriding Regulation 13 was issued by the Government on January 18, 1952. This amendment exempted from price control all sales of imported and domestic rutile ores and concentrates and the allied services of mining and processing such materials. The removal of rutile from price control permitted domestic consumers and dealers to compete with other countries in obtaining this material and allowed domestic producers to continue production by meeting the higher prices requested by the operating companies. The ceiling price for ilmenite was not removed, as its price was considered to be

TABLE 7.—*Market price of ilmenite and rutile, 1939–54* [1]

Year	Ilmenite		Rutile	
	TiO_2 content, percent	Price per ton	TiO_2 content, percent	Price per pound
1939	45 to 55	$10 to $12	94	$0.10
			88 to 90	.0275
1940	45 to 55	10 to 12	94	.10
	50 to 60	18 to 20	88 to 90	.0275
			88 to 90	.0375
1941	50 to 60	18 to 20	94	$0.08 to .10
			88 to 90	.0375
			88 to 90	.0475
1942	60	28 to 30	94	.08 to .10
1943	60	28 to 30	94	.08 to .10
1944	60	28 to 30	94	.08 to .10
1945	60	28 to 30	94	.08 to .10
1946	60	28 to 30	94	.08 to .10
	57 to 60	24 to 26		
	57 to 60	22 to 24		
	57 to 60	19 to 20		
1947	57 to 60	19 to 20	94	.08 to .10
	57 to 60	17 to 19		
	57 to 60	19 to 20		
1948	57 to 60	19 to 20	94	.08 to .10
	56 to 59	18 to 19	94	.06 to .08
	56 to 59	18 to 20		
1949	56 to 59	18 to 20	94	.06 to .08
	56 to 59	16 to 18	94	.04 to .06
	56 to 59	15 to 17	94	.04 to .05
	56 to 59	14 to 16		
1950	56 to 59	14 to 16	94	.04 to .05
			94	.035 to .045
1951	56 to 59	14 to 16	94	.035 to .045
	56 to 59	15 to 17	94	.0525 to .0625
	56 to 59	16 to 18		
1952	56 to 59	16 to 18	94	.0525 to .0625
	56 to 59	16 to 20	94	.035 to .045
	56 to 59	18 to 20	94	.04 to .05
			94	.05 to .07
			94	.06 to .08
			94	.07 to .08
			94	.07 to .085
1953	59.5	18 to 20	94	.07 to .085
			94	.06 to .065
			94	.055 to .06
			94	.05 to .06
1954	59.5	18 to 20	94	.05 to .06
			94	.055 to .06
			94	.05¾ to .06¼
			94	.06 to .06½
			94	.06½ to .06¾
			94	.06¾ to .07
			94	.07 to .07½

[1] Prices are quoted from the E & MJ Metal and Mineral Markets.

The price of ilmenite is quoted as nominal, per gross (2,240 pounds) ton, f. o. b. Atlantic seaboard.

at the optimum level for continuing current domestic production and importation, as well as expansion programs.

Ductile titanium sponge metal (titanium 99.3 percent plus) was quoted by E. I. du Pont de Nemours Co., Inc., Newport, Del., effective December 1, 1954, at $4.50 per pound for Grade A–1 (0.30 percent iron, maximum) and $4 per pound for Grade A–2 (0.50 percent iron, maximum), f. o. b. shipping point. Titanium sponge sold for $4.72 per pound for Grade A and $4.46 per pound for Grade B for the period April 1, 1954, to November 30, 1954. Before April 1, 1954, Du Pont's price quotations on titanium sponge were $7.50 per pound in quantities of less than 100 pounds and $5 per pound in quantities of 100 pounds or more, f. o. b. shipping point. Titanium Metals Corp. of America, New York, N. Y., and Rem-Cru Titanium, Inc., Midland, Pa., announced on February 1, 1954, and February 19, 1954, respectively, a price reduction of 8 to 14 percent on titanium-mill product extras. Prices for titanium-mill products, as quoted in the American Metal Market, June 1954, were as follows: Base prices per pound in lots of 10,000 pounds or more, in commercially pure and alloy grades, f. o. b. mill: Forging billets $9, bars and rods $9, wire $10, plate $12, sheet and strip $15. Titanium-mill products were manufactured in 1954 by Titanium Metals Corp. of America, New York, N. Y.; Rem-Cru Titanium, Inc., Midland, Pa.; Mallory-Sharon Titanium Corp., Niles, Ohio; and Republic Steel Corp., Cleveland, Ohio.

TITANIUM AS A SUBSTITUTE

Although titanium is a metal new to industry, large-scale production of its oxide has provided an excellent raw-material base for development of the metal. Titanium-metal development comes at a fortunate time to ease the burden on the earth's dwindling resources. It is one of the "growing-metals group," consisting of titanium, aluminum, and magnesium. The relatively large quantity of titanium potentially available as compared to the small quantity of such metals as copper, lead, and zinc is striking.

It is light, strong, and resistant to corrosion, properties of paramount import to designers and users of modern equipment because of the ever-increasing demand for lightweight, sturdy, high-speed machines. It is not expected that titanium will displace iron or structural steel as a universal material for low-cost construction or that it can compete in all applications with aluminum or magnesium, but this new metal will undoubtedly find many important engineering uses that are only partly filled by established construction materials.

Titanium metal and its alloys can be expected to partly displace stainless steel and such other corrosion-resistant alloys as brass and Monel.

Titanium dioxide pigments similar to titanium metal have the advantages of ample domestic ore supply and outstanding characteristics. Titanium pigments were used increasingly in 1954 in place of other white pigments.

In cemented carbides, titanium carbide may be substituted either partly or entirely, for tungsten carbide.

DEVELOPMENT OF TITANIUM-METAL INDUSTRY

One outstanding metallurgical development of the Bureau of Mines led to establishment of the ductile titanium-metal industry. In 1938 the Bureau began investigating the technology of titanium and conducted a critical survey and small-scale trial of virtually all suggested methods for producing titanium metal. The magnesium reduction of titanium tetrachloride was selected as most suitable for quick development for tonnage production and evaluation of semifabricated titanium products. After the Bureau's demonstration that ductile titanium could be produced on a pilot-plant scale (*15*, *45*, *46*) and following distribution of free samples for property and use studies, the interest of industry was aroused.

The world's first small-scale commercial production of ductile titanium metal was begun at Newport, Del., by the E. I. du Pont de Nemours Co., Inc., Pigments Department, July 1948; limited commercial availability of titanium was announced by Du Pont September 1948. Du Pont's exploratory research was underway in 1944, and a formal research program was established in 1946. In 1948 Du Pont and its subsidiaries took an active interest in titanium. Du Pont's pilot units were expanded through successive stages until by 1952 the company was producing 2½ short tons of titanium sponge a day. Government aid, in the form of a contract, encouraged Du Pont's expansion program for production of 13,500 tons of titanium sponge metal over a 5-year period, in addition to its own private production facilities of 4,500 tons over this period. Du Pont's rated production capacity of 10 tons a day was reached in 1954. Negotiations were underway between Du Pont and the Government early in 1954 toward the erection of a plant at New Johnsonville, Tenn., to produce 7,000 to 8,000 tons of titanium a year.

The National Lead Co. was also quick to see the potentialities of titanium metal. Pilot plants were constructed in 1949 and 1951 at Sayreville, N. J., and Niagara Falls, N. Y., and

National Lead also operated the Bureau of Mines pilot plant from November 1950 to November 1951 to familiarize its personnel with the new techniques developed by the Bureau. The National Lead Co. and Allegheny Ludlum Steel Corp. organized the Titanium Metals Corp. of America early in 1950 to produce and market titanium metal, alloys, and related products. Allegheny Ludlum Steel Corp. established its position in titanium in 1949, when it had a semicommercial titanium-melting facility in operation. TMCA signed a letter of intent in August 1951 for the construction and operation of a plant to produce 3,600 short tons of titanium sponge a year for 5 years. The site of the new operations was the former Basic Magnesium, Inc., plant at Henderson, Nev., built by the Government during World War II.

New titanium organizations were formed during the years 1950 to mid-1954. Titanium Metals Corp of America, E. I. du Pont de Nemours Co., Inc., Cramet, Inc., and Dow Chemical Co., signed Government contracts for the production of titanium sponge metal by modifications of a magnesium reduction process. The Electro Metallurgical Co., Division of the Union Carbide & Carbon Corp., also signed a Government contract for the production of titanium metal by a sodium-reduction method. Other Government activities on titanium, as of 1954, included a contract with the Horizons Titanium Corp., a new organization formed by Horizons, Inc., and Ferro Corp., for constructing and operating a pilot plant to study a new process for producing titanium metal and leasing of the Government magnesium plant at Manteca, Calif., to Western Pyromet Co. The company was to conduct research on a new method for producing titanium metal. Rem-Cru Titanium, Inc., was formed by Remington Arms Co., Inc., and Crucible Steel Co. to make titanium and titanium-alloy products. National Research Corp. and Monsanto Chemical joined to develop an improved process for the production of titanium. P. R. Mallory and Sharon Steel Corp. formed the Mallory-Sharon Titanium Corp. for developing, producing, and marketing titanium and titanium alloys. The Kennecott Copper Corp. constructed a pilot plant (capacity, 200 pounds per day) at Battelle Memorial Institute to develop and evaluate several methods of producing titanium metal. Kennecott Copper Corp. also announced that a new Research and Development Department for the Chase Brass & Copper Co., an affiliate of the Kennecott Copper Corp., was established at Waterbury, Conn., to conduct research on the production, treatment, and uses of copper and titanium alloys. The Glidden Co. and Bohn Aluminum & Brass Corp. combined their research facilities to develop methods for producing pure titanium and titanium alloys. Other companies in the chemical and basic metallurgical industries that have shown an interest in titanium-metal production were Kaiser Aluminum & Chemical Corp., Chicago Development Corp., West Coast Iron & Titanium, Wigton-Abbott Corp., Harvey Machine Co., Inc., Olin Industries, Inc., Derby & Co., K-W Laboratories, Columbia-Southern Chemical Corp., Wyandotte Chemical Co., Cooper Metallurgical Associates, National Distillers Products Corp., New Jersey Zinc Co., Eagle Picher Co., Metal Hydrides, Inc., and United International Research, Inc.

In 1954 titanium sponge metal was produced commercially by E. I. du Pont de Nemours Co., Inc., Newport, Del.; Titanium Metals Corp. of America, Henderson, Nev.; Dow Chemical Co., Midland, Mich.; and the Federal Bureau of Mines, Boulder City, Nev. The Bureau of Mines output of titanium sponge, under contract with a Government defense agency, supplemented but did not compete with private production. Titanium and titanium-alloy mill products were produced and sold by four companies in 1954: Titanium Metals Corp. of America; Mallory-Sharon Titanium Corp.; Republic Steel Corp.; and Rem-Cru Titanium, Inc. The latter three purchased sponge from Du Pont. Allegheny Ludlum processed products for the Titanium Metals Corp. of America, which it owns jointly with the National Lead Co.

The basic process for producing titanium sponge-metal commercially as of 1954 (titanium tetrachloride reduction with magnesium in an inert atmosphere) was patented by Dr. William (Wilhelm) J. Kroll in Germany in 1938 and in the United States in 1940. The United States patent (No. 2,205,854) was vested in the Alien Property Custodian in 1943, under the authority of the Trading With the Enemy Act. In addition, the contract rights of Siemens & Halske, A.-G., a corporation organized under the laws of Germany, as exclusive licensee under the patent, were vested by the Attorney General in 1948—he having succeeded to the powers of the Alien Property Custodian. After the vestings, the patent and contract rights therein were the subject of a suit brought by Dr. Kroll against the Attorney General. The judgment, which was entered on April 3, 1951, required the Attorney General to convey the legal title to the patent to Dr. Kroll but provided that the Attorney General was entitled to retain an exclusive license in and to the patent, a right to sublicense others under the patent, and a right to share the royalties from sublicensees equally with Dr. Kroll. The royalties were to be at rates agreed to by the

Attorney General and Dr. Kroll, or, in the absence of such an agreement, at rates set by an order of the court. The parties were not able to reach such an agreement, and the court, by an order entered on December 1, 1953, established the following per annum rates at which the Attorney General is authorized to grant sublicenses:

(1) 5 percent of the gross sales price of the first 50,000 pounds of titanium sold.

(2) 3 percent of the gross sales price of the next 50,000 pounds of titanium sold.

(3) 1 percent of the gross sales price of all titanium sold in excess of 100,000 pounds.

Sublicenses at these rates may be granted retroactive to April 3, 1951. Such licenses may be obtained from the Office of Alien Property, United States Department of Justice, Washington 25, D. C. The patent will expire on June 25, 1957.

In January 1951 the Government initiated a titanium-expansion program based on an evaluation of military requirements. This program, recommending construction and operation of two 10-ton-per-day plants with an aggregate annual rated capacity of 7,200 tons, served as the basis for the first expansion of titanium production and resulted in the construction and initial operation of a 10-ton-per-day plant by Titanium Metals Corp. of America at Henderson, Nev., and expansion by Du Pont to increase its production at Newport, Del., from 2½ to 10 tons per day.

Expansion goals for titanium sponge-metal production were issued and periodically revised by the Government agencies. Changes in the production goal for titanium sponge are shown in table 8.

TABLE 8.—*Expansion goals for titanium sponge*

Date:	*Issuing agency*	*Goal (short tons)*
March 1951	Defense Production Administration.	7,200 (no goal date specified).
June 20, 1952	do	10,000 by 1955.
Oct. 13, 1952	do	22,000 by 1955.
Aug. 6, 1953	Office of Defense Mobilization.	25,000 in 1956.

At the end of 1954, the Government negotiated in good faith with all potential domestic producers prepared to enter the titanium-sponge production field, irrespective of the 25,000-ton figure. Such proposals were considered for specific approval in the light of facts and conditions prevailing at the time. It was contemplated that, by proceeding in this fashion, the 25,000-ton figure may be exceeded, but no ceiling was placed officially on the additional facilities to which the Government was prepared to lend financial assistance.

The titanium-metal industry has grown from the infant stage more rapidly than other metals, which historically required decades for establishment. At the high cost of production in 1954 virtually all titanium produced was intended for defense purposes. Because of the nature of this new strategic-metal industry—that is, (1) high plant cost, (2) high operating cost, (3) limited civilian applications, (4) possibility of plant and process obsolescence—the Government provided assistance to producers along the following lines to increase production: (1) Loans to construct plants, (2) procurement contracts, (3) accelerated tax amortization, (4) purchases in times of excess supply for a temporary Government working inventory, (5) loans for pilot plants, and (6) demonstration to contractors of all steps in production at the Bureau of Mines pilot plant, Boulder City, Nev.

The Government contracted with industrial concerns on an individual basis for additional plant capacity in amounts commensurate with the technical experience and financial ability of each contractor. Proposals received by the Government were reviewed and judged by a technical committee of the National Research Council, acting on a confidential basis.

The planned annual production capacity of titanium facilities, as of September 15, 1954, totaled 22,500 short tons of titanium sponge metal. Of this total Government contracts called for an annual titanium output of 21,600 tons by 1957. The capacity output of titanium metal, independent of Government contracts, was 900 tons a year in 1954. Contracts signed by the Government under the titanium-metal expansion program are summarized as follows:

Titanium Metals Corp. of America signed a letter agreement on August 1, 1951, for construction and operation of a plant at the Government's former Basic Magnesium, Inc., plant, Henderson, Nev., to produce 3,600 short tons of titanium sponge metal a year for 5 years. An advance of $15 million was made to TMCA by the Government. This advance, plus interest at 4 percent a year on the unpaid balance, was repayable either in dollars or in titanium metal. Commencing with the date when the plant was capable of producing approximately 1,800 tons of titanium a year and continuing thereafter until the termination or completion of the contract (termination date, August 1, 1961), TMCA was required to sell to the Government, or for its account, all or any part of 100 tons of titanium in sponge or ingot form during each 3-month period. Major components of the plant were leased from the State of Nevada. Power was allocated from Hoover and Davis Dams, up to a maximum of 151 million kw.-hr., by the Colorado River Commission from the quota allotted to the State of Nevada. The Henderson plant, the first fully integrated titanium plant with facilities for chloridation of titanium ore and recovery of magnesium and chlorine by electrolysis of magnesium chloride, began commercial production of titanium sponge in October 1951, with a target of 3,600 tons of titanium sponge by 1953. This capacity goal was later extended to 1954.

E. I. du Pont de Nemours Co., Inc., concluded the second major agreement in the Government-industry program to increase titanium sponge-metal production on July 24, 1952. The Government agreed to advance the company up to $14.7 million to expand its titanium-producing facilities at Newport and Edge Moor, Del. The loan was to be repaid with interest (4 percent) as salable titanium sponge was produced. The contract called for the production of 13,500 short tons of titanium sponge in addition to the company's private output of 4,500 tons over a 5-year period. Beginning with the date when the additional facilities were capable of producing approximately 1,350 tons of titanium annually and continuing thereafter until termination or completion of the contract (termination date, June 30, 1962), the Administrator (Government) had the right to require the company to sell to the Government, or for its account, all or any part of the first 75 tons of such titanium sponge metal produced from the additional facilities by the contractor thereafter during any calendar quarter.

The Federal Bureau of Mines was requested to enter into an agreement (signed on April 30, 1953) with another Government agency for production of titanium sponge at the Bureau pilot plant, Boulder City, Nev. The contract was negotiated owing to the shortage and the anticipated future insufficiency of titanium metal for national defense. Under the contract the Bureau of Mines was to endeavor to produce, within a period of 18 months following the effective date of the agreement, a minimum of 180 short tons and a maximum of 250 tons of usable ductile titanium at a minimum rate of 1,000 pounds and an optimum rate of 1,400 pounds daily. Funds were to be advanced to the Bureau based on production costs, not to exceed a maximum of $2,250,000. Bureau of Mines production of acceptable titanium sponge under the contract totaled 246 tons. The Bureau's production (which ceased September 7, 1954, about 2 months before the contract's termination date, October 31, 1954) was to supplement but not compete with private production.

The agreement permitted minor capital improvements and equipment alterations designed to decrease production cost, enabled the Bureau to obtain a more accurate cost evaluation of titanium-production processes, and facilitated certain phases of the research program which required continuous plant operations.

Cramet, Inc. (a wholly owned subsidiary of Crane Co.), signed a contract with the Government on July 31, 1953, for the construction and operation of a plant with a capacity of 6,000 short tons of titanium sponge annually. The new facility, which is at Chattanooga, Tenn., was scheduled to begin partial production by early 1955. The Government agreed to advance the company up to $24,950,000 for construction and equipment costs to be repaid with interest (5 percent a year) on the unpaid balance from the date of the advance. The Government had an option to buy up to 7,500 tons of titanium sponge during the term of the contract at $5 per pound for any sponge bought during the first year of production and $4 per pound or market price, whichever was greater, thereafter. On the other hand, the company could require that the Government buy up to 6,000 tons of sponge during the life of the contract at $5 per pound during the first year of production (not to exceed 1,000 tons) and $4 per pound thereafter. The contract was to expire 7½ years from the date of execution.

Horizons Titanium Corp., entered into a contract with the Government on July 1, 1954, that provided for the design, installation, and operation of a pilot plant at Stamford, Conn., to demonstrate the commercial feasibility of the production of titanium by an electrolytic process. The duration of the undertaking was 1 year, and the maximum Government expenditure was $564,300. The contract provided for repayment to the Government if the process was considered feasible by it and if commercial production was undertaken by the contractor or its licensee or assignee within 10 years from the date of the contractor's final report of results under the agreement.

The Dow Chemical Co. contract, signed July 8, 1954, provided for the expansion of the company facilities at Midland, Mich., from 600 pounds per day to 1½ short tons per day by January 1, 1956, and 5 tons per day by July 1956. The company agreed to expand its productive facilities at its own expense provided that it was assured a market for titanium sponge. Under the agreement the Government could purchase a maximum of 2 million pounds of titanium sponge produced by the company before July 1956 if the company was unable to find other markets or if the Government exercised its option to buy. The titanium sponge was to be purchased by the Government at market price for the first 6,030 pounds in any month and the balance at $5 per pound or the lowest figure at which the company offered the metal to any other customer. The company agreed to maintain the rated capacity of 5 tons per day for a period of 5 years and in case of a plant shutdown to reactivate the facility fully within 120 days. The contract was to be terminated July 1, 1961.

Electro Metallurgical Co. (*Division of the Union Carbide & Carbon Corp.*) signed a Government contract on September 10, 1954, that called for construction of a $31.5 million plant at Ashtabula, Ohio, to produce 7,500 short tons of titanium annually. The agreement stated that the company would use its own funds for the design, construction, and equipment of the facility. The company had the right to deliver to the Government whatever production it was unable to sell in the commercial market, up to a maximum of 7,500 tons annually. The Government had an option to buy 4,500 tons annually at the market price prevailing at time of delivery. The life of the agreement was 5 years from the time the plant began to produce titanium in commercial quantities. Initial production was to begin in 1956. The new plant was reported to be the first commercial facility to employ a method other than the Kroll process. The company pioneered a sodium-reduction method on pilot and prototype stages.

Western Pyromet Co. received from the Government on August 31, 1954, the right to use a portion of the Government-owned magnesium plant at Manteca, Calif., for experimental and pilot operations on titanium production. The company was to operate 2 furnaces, with 32 retorts, at the Manteca plant for 9 to 12 months, to study a titanium-reduction process reported to be a modification of the Kroll process. The Government was to receive $2,000 a month during the research project.

A recapitulation of Government contracts on titanium-metal production can be found in table 9.

Tentative American Society for Testing Materials specifications in 1954 for high-purity titanium produced by the thermal decomposition of titanium iodide vapor on a titanium surface (ASTM B266–52T) and titanium semifabricated products, such as strip and sheet (ASTM B265–52T), are listed in table 10. Tentative specifications for titanium ingot (ASTM B264–52T) are identical to those listed for titanium semifabricated products.

TABLE 9.—*Summary of Government titanium production contracts in force as of Sept. 15, 1954*

	Contractors				
	Titanium Metals Corp. of America	E. I. du Pont de Nemours Co., Inc.	Cramet, Inc.	Dow Chemical Co.	Electro Metallurgical Co.
Date signed	Aug. 1, 1951	July 24, 1952	July 31, 1953	July 8, 1954	Sept. 14, 1954.
Plant location	Henderson, Nev	Newport, Del	Chattanooga, Tenn	Midland, Mich	Ashtabula, Ohio.
Planned annual capacity, under contract short tons	3,600	2,700	6,000	1,800	7,500.
Government loan	$15,000,000	$14,700,000	$24,950,000		
Tax amortization rate percent	90	90	90		90.
Titanium-sponge specifications, percent:					
Titanium (min.)	99.3	99.3	99.5	99.3	
Iron (max.)	0.25	0.25	0.15	0.15	
Nitrogen (max.)	0.03	0.03	0.03	0.03	
Chlorine (max.)		0.15	0.12	0.15	
Magnesium (max.)		0.10	0.07	0.07	
Carbon (max.)		0.05	0.05	0.07	
Total impurities (max.)	0.70	0.70	0.50	0.70	
Brinell hardness No	225	203	170	160	
Repayment of Government loan, poundage payment—per pound	$0.47	$0.61	$0.52		
Contract termination date	Aug. 1, 1961	June 30, 1962	Jan. 31, 1961	July 1, 1961	June 30, 1963.

A temporary revolving-stockpile program was established in August 1951 by the Federal Government to maintain capacity operation of titanium-sponge-manufacturing facilities during the development of military applications, likewise to assure an increased supply of titanium and the utilization of the titanium sponge in the manner most advantageous to the national defense. Under this program the Government maintained a revolving fund of $5 million for purchase and resale of not over 1 million pounds of titanium sponge at a price not to exceed $5 a pound. The revolving stockpile, in which titanium sponge was purchased for resale only, was in no way connected with the National Stockpile of essential materials. The authorization, originally intended to expire March 15, 1952, was extended to December 31, 1952, and later to December 31, 1953. By December 31, 1952, 303 tons of titanium sponge had been supplied to the revolving stockpile by E. I. du Pont de Nemours Co., Inc., Newport, Del. Due to the increasing military demand for titanium metal this material was resold to titanium fabricators, and by June 22, 1953, the stockpile was depleted. Therefore, because of the shortage of titanium sponge in 1953 no material was offered for Government purchase

Table 10.—*Tentative ASTM specifications on titanium chemical composition*

[Percent]

	Iodide titanium [1]	Titanium semifabricated products [2]			
		Grade 1	Grade 2	Grade 3	Grade 4
Titanium (min.)	99. 9	99. 3	99. 2	99. 0	98. 0
Iron (max.)	. 02	. 12	. 25	. 25	. 25
Nitrogen (max.)	. 01	. 08	. 10	. 15	. 15
Carbon (max.)	. 03	. 05	. 20	. 20	. 80
Silicon (max.)	. 02				
Aluminum (max.)	. 03				
Manganese (max.)	. 04				
Tungsten (max.)		. 08	. 02	. 02	. 02
Oxygen (max.)	. 01	. 15	. 20	. 25	. 30
Brinell hardness No. (max.—3,000 kg. load)	65–85	180	190	225	250

[1] Copper, chromium, lead, molybdenum, magnesium, nickel, vanadium, tungsten, tin, and other metals not specifically mentioned shall not exceed 0.01 percent each.

[2] Chromium, magnesium, aluminum, manganese, nickel, copper, lead, molybdenum, vanadium, and other elements not specifically mentioned shall total not over 0.25 percent and no 1 element not specifically mentioned shall exceed 0.10 percent.

throughout the year. Titanium-sponge production in 1954 temporarily exceeded consumption, and the Government increased its "working inventory" to assure continued production; therefore, amendments were added to the revolving-fund contract extending it to June 30, 1954, and later to December 31, 1954. The latest amendment allowed the Government to purchase up to 8 million pounds of titanium sponge under the revolving-fund program. Initial Government specifications and actual average quality of titanium sponge submitted by Du Pont as of December 31, 1952, were as follows:

	Government specifications (percent)	*Actual average quality purchased (percent)*
Titanium	[1] 99. 3	99. 7
Iron	[2] . 25	. 16
Nitrogen	[2] . 03	. 024
Chlorine	[2] . 15	. 10
Magnesium	[2] . 10	. 04
Brinell hardness No	[2] 203	168

[1] Minimum.
[2] Maximum.

Titanium sponge offered under the revolving-fund program in 1954 had to meet the following Government specifications:

	Specifications for Du Pont titanium sponge (percent)	*Specifications for TMCA titanium sponge (percent)*
Titanium (minimum)	99. 3	99. 3
Iron (maximum)	. 30	. 25
Nitrogen (maximum)	. 03	. 03
Chlorine (maximum)	. 15	. 15
Magnesium (maximum)	. 10	. 50
Carbon (maximum)	. 05	. 07
Brinell hardness No. (maximum)	160	170

On February 10, 1954, the Government established an interim expansion goal of 37,500 short tons for titanium-melting facilities capacity to be reached by 1956. The industry had, at that time, a melting capacity of approximately 7,680 tons a year, and 4 smelters were planning expansion of 4,200 tons a year, which will result in domestic output of at least 11,880 tons of ingot by 1956. On August 17, 1954, an interim expansion goal for titanium-processing facilities was announced by the Government, calling for industry annual capacity by the end of 1956 to process 37,500 tons of titanium ingot into titanium mill products.

Titanium products—namely, sponge, tubes, extrusions, titanium-bearing alloys and titanium-base alloys—were designated as scarce materials by the Government on February 18, 1953. These products were removed from this status by amendment on June 18, 1953, but were reinstated on July 16, 1953. The Government established controls on November 18, 1953, on the general distribution of titanium sponge and metal (ingot and mill products) in the civilian market. However, a Government order dated May 19, 1954, authorized producers of titanium products to release 10 percent of the monthly output for civilian use. This action was taken to permit the industry to expand titanium output and development along normal lines.

Secondary or scrap titanium could not be considered in 1954 as a firm source of supply for commercially pure titanium in significant quantities. Because of the large accumulation of scrap by the melter and ingotmakers, they were in no position in 1954 to purchase from their customers until they have reduced their inventories to a working level.

The characteristics of titanium have inspired widespread interest in its engineering development and exploitation. The Government, industry, and research organizations have spent millions of dollars to solve the many uncertainties and difficulties that have arisen in the extractive and physical metallurgy of titanium. Intensive development through research has produced useful titanium alloys with tensile strengths up to 200,000 pounds per square inch, or over twice that of unalloyed annealed titanium. To assist in overcoming the various obstacles in the titanium-metal industry and improve the quality of titanium metal, the Government contracted many research projects to industry and allowed industry personnel to visit and view its titanium operations at Boulder City, Nev., and College Park, Md. An important Government development was a program on basic research in extractive metallurgy on titanium. The general objective of the program was to provide essential fundamental data to accelerate the development of existing titanium-recovery processes and lead to the devising of new ones. Proposed projects covered:

(1) Properties of pure compounds.
(2) Properties of solutions.
(3) Properties of electrolytes.
(4) Heterogeneous equilibria.
(5) Chemical reactions.
(6) Surface phenomena.
(7) Crystal growth.
(8) Analysis and identification.

Joint Government-industry committees were established by the Government to cope with the many problems in titanium production and fabrication. The Titanium Producers and Fabricators Industry Advisory Committee, established on July 17, 1952, was concerned with the basic problem of increasing titanium production for national defense. A Government Titanium Advisory Committee was organized February 17, 1954, to facilitate coordination of Federal policies and programs with respect to the supply of titanium and to serve as a focal point for collective and disseminative technical information to industry.

The Department of Defense announced on November 18, 1954, that the Battelle Memorial Institute, Columbus, Ohio, was designated as a clearing house and consulting service to defense industries for information on the production and application of titanium. This move was not made to concentrate titanium-metallurgy research in one organization but only to coordinate it for dissemination to defense industries.

STATUS OF JAPANESE INDUSTRY

Production of titanium metal in Japan was begun on an experimental basis in 1952, and by 1954 four companies were producing it commercially. Production increased from 9 short tons in 1952 to 77 tons in 1953, and the Japanese estimated 1,000 tons of titanium sponge in 1954. A goal of about 8,000 tons was set for the fiscal year 1957 (April 1, 1957, to March 31, 1958). From the start of operation in 1952 to May 1954 the Japanese had produced 218 tons of titanium and exported 81 tons to the United States and 32 tons to the United Kingdom. Japanese sponge offerings to the United States through dealers were reported to be at a price (including tariff) competitive with that of domestic sponge in 1954.

TITANIUM INDUSTRY IN OTHER COUNTRIES

Great Britain was believed to be the only other country, outside of those behind the Iron Curtain, producing ductile titanium metal on a commercial scale as of 1954. A plant at Widnes with a production capacity of 1,500 long tons of titanium in 1955 was under construction in 1954. Construction and operation of the Widnes plant were wholly financed by the Imperial Chemical Industries, Ltd. Government assistance was in the form of a procurement contract for a portion of the company output.

Research on titanium was underway in Canada and other countries. A new electrolytic process was studied on a pilot-plant scale by the Shawinigan Water & Power Co. in 1954. The company reported that it believed its process will produce titanium metal at prices substantially lower than have been possible under 1954 technology. Experimental work on the production of ductile titanium metal was also conducted by the Dominion Magnesium, Ltd. The company produced titanium material ranging from 96 to 98 percent titanium.

OUTLOOK

A potential supply of titaniferous minerals is available for a considerable metal industry, in addition to the flourishing titanium-pigment industry. Recent technical improvements and industrial activity in smelting titaniferous iron ores promise to alleviate anxiety about supplies in North America for many years to come. Rutile cannot be considered a major domestic source material for production of titanium metal, as it does not occur in the United States in sufficient quantities to sustain an expanded titanium-metal industry. Any major expansion of titanium production must be premised on the use of ilmenite or high-titania slag. Titanium metal for some time will be used mainly for military purposes, where the advantages derived can justify the high price. The demand for titanium metal cannot be projected very accurately, inasmuch as technological developments are so uncertain. Requirements are contingent on many factors, such as development of end uses, availability of alloys with required properties, utilization of scrap, and a decrease in price. The planned annual production capacity of titanium metal facilities, as of October 1, 1954, totaled 22,500 tons. This capacity was expected in 1957.

Estimated future requirements are greater than the production now contemplated; however, the requirements are not firm, as they are based in some instances on an assumption of unlimited availability of the necessary quality of metal for applications requiring little or no development in design. Engineering of titanium into the design of some articles and the tooling of industry may not be developed to the point that indicated quantities could be used during the earlier dates, even if the metal were actually available. On the other hand, if low-cost production processes are developed and, as a result, the price of titanium sheet and other semifabricated products (adjusted for weight and performance) declines to a level competitive with stainless steel, the demand would change considerably. The availability of low-cost titanium metal would stimulate civilian as well as military use, and fabrication and end-use development would be greatly expanded.

Continued expansion of the titanium-pigment industry is anticipated. Plans in 1954 called for an increase in annual output by 1956 of about 80,000 short tons of titanium pigments.

PROBLEMS

The current and continuing problems of the titanium industry are as follows:

CLASSIFICATION AND EFFECTIVE UTILIZATION OF ORE RESERVES

There is inadequate knowledge of the location, quantity, and grade of the widely distributed domestic titanium minerals, methods for economically recovering the titanium, and, in some cases, byproduct values. Processes for producing metal from high-titanium slag or ilmenite would allow maintenance of an expanding titanium industry. Utilization of high-titania slag low in iron for titanium-pigment production would decrease the present sulfuric acid losses in treatment of ilmenite.

REDUCTION IN COST OF TITANIUM METAL

The continued expansion of titanium-metal production and its widespread use for civilian as well as military purposes depend on reduction in the cost of producing uniform high-purity titanium metal and products to a fraction of their 1954 level.

DEVELOPMENT OF INFORMATION ON PROPERTIES AND USES OF TITANIUM METAL

Although a substantial tonnage of titanium sponge (5,370 short tons) was produced commercially in 1954, only a few of the numerous potential applications have been adopted in the military and industrial fields. Many facts must be known before titanium can be utilized with maximum efficiency, even though titanium and titanium alloys have been investigated sufficiently to assure their importance as engineering materials. Although some progress has been made, additional problems remain along the following lines:

(1) Improvement of the ductility of high-strength alloys.

(2) Development of alloys to extend the safe temperature limit beyond 800° F.

(3) Development of more uniform mechanical properties and improved heat-treating practices.

(4) Studies of the embrittling effect of hydrogen and other impurities.

(5) Investigation of welding methods, especially with respect to high-strength alloys.

(6) Surface coating of titanium for wear and abrasion resistance.

(7) Evaluation of suitable methods and materials for containing molten titanium.

(8) Studies of creep phenomena in titanium, especially at room temperature, and determination of additional unnotched fatigue-strength data for all wrought forms of titanium.

(9) Further development and application of extruded aluminum.

(10) Improvement of melting and remelting procedures.

(11) Research on procedures for more rapid evaluation of new alloys with respect to corrosion fatigue and inherent electromotive forces associated with dissimilar metals and strain concentration gradients.

(12) Studies on the electrodeposition of titanium, including fundamental investigations involving the synthesis of titanium compounds.

UTILIZATION OF TITANIUM SCRAP

The reclamation of scrap generated in the fabrication and end uses of titanium is of great concern to the Government and industry. Plant practices for processing scrap were under development, but successful application was limited. Although some of this scrap was utilized, a large portion was stockpiled awaiting development of melting procedures in which the impurities in the scrap would not be increased to an objectionable degree.

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TUNGSTEN

By

Robert W. Geehan[1]

TUNGSTEN is a metal with unique properties that lead to its use to cut and form other metals; it appears in such products as dies, drills, lathe tools, and armor-piercing shells. The defection of China, historically the chief source of tungsten, and the fighting in Korea, which cut off another important source, led to a shortage that became acute early in 1951. Because of the need for the metal, particularly for machine tools and projectiles, this shortage caused a crisis in the defense effort, and a plan of action to increase and conserve the supply was launched. Results of this program are now apparent; domestic production and imports have both reached alltime highs.

Summary

Tungsten is a steel-gray, heavy metal used in alloy steels for high-temperature applications, in tungsten carbide for cutting tools and armor-piercing shells, and as pure metal in lighting and electronics. Concentrates of minerals containing tungsten are produced in many countries; historically China, United States, Burma, Bolivia, Portugal, and Korea have been the leading producers and United States, United Kingdom, France, Germany, Sweden, and U. S. S. R. the important consumers. The concentrates are converted to ferrotungsten for addition to steel, charged direct to steel, and converted to tungsten metal and chemicals. Tungsten carbide is largely produced from the metal but some is produced from concentrates.

Most domestic ores mined contain less than 1 percent WO_3 (tungsten trioxide); these are concentrated at or near the mines to a product containing 50 to 70 percent WO_3 by ore dressing or hydrometallurgy. Impurities present in concentrates are important in determining the usability and type of use; arsenic, phosphorus, sulfur, and molybdenum are the most important of these.

In many instances molybdenum can be substituted for tungsten in steels, but no satisfactory substitute has been developed in carbides or pure-metal uses.

The United States has depended on imports for much of its tungsten; and, to insure a supply for emergency periods, the Government is stockpiling concentrates and has provided incentives to stimulate domestic and foreign production. The largest known ore reserves are in China; reserves developed in other areas are unsatisfactory from the standpoint of long-term production. Restrictions on end use were in effect during the Korean emergency; most important was the provision restricting the quantity of high-tungsten tool steel as related to total tool steel.

The long-term trend for total use indicates greatly increased consumption; however, it is expected that there will be violent fluctuations in both consumption and price, partly because of the unpredictable military component in requirements.

Nearly all problems important to the tungsten industry are related to wide swings in demand and price. Producers, well aware that only a few tungsten mines were worked during past periods of slack demand, may not conduct aggressive development and exploration. Consumers may design tungsten out of their products because of anticipated shortages and high prices during emergency periods.

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[1] Assistant chief, Division of Minerals, Bureau of Mines.

BACKGROUND

HISTORY

The chemistry of the element tungsten was outlined in the period 1781–86, and as early as 1855 patents were issued for alloy steels containing tungsten. However, the first tool steel of the modern high-temperature type was introduced in 1898; general adoption of this product led to redesigning of machines and to great savings in production costs for countless items. During the early years of the 20th century, tungsten filaments were introduced and have been used in electric lamps since that time; later they came into general use in electronic equipment. In 1927 it was estimated [2] that costs more than $2 billion over the actual expenditures for lighting would have been required if carbon-filament lamps had been used to produce the same illumination as the tungsten-filament lamps generated in that year. In the period from 1927 to the present, another tungsten product, tungsten carbide, has been developed; the effect on industry resembles the period when tungsten tool steel was introduced. Machines and processes have been redesigned, and tremendous savings in production costs again have resulted.

The need for this metal was emphasized during World Wars I and II, when it was used to speed production of armament items; however, the present astronomical military requirements for tungsten are, to a large degree, based on introduction of a tungsten carbide armor-piercing shell by the Germans in World War II.

Following both wars there was a very sharp decline in the demand for tungsten, and many of the world's mines were forced to close until wartime inventories were consumed. The marked changes in demand and price have led to an unstable situation in producing areas and in general have resulted in continuation of primitive mining and milling in many areas of the world.

GEOGRAPHIC DISTRIBUTION

Workable tungsten deposits have been found in many areas of the world; indeed, a noteworthy concentration of the better deposits appears in a belt bordering the Pacific Ocean. Exceptions are one very productive mine in North Carolina, the large producing area in Europe on the Iberian Peninsula, and scattered deposits in Africa, Europe, and Asia.

Percentages of total production to date, by geographic areas, are about as follows: Asia 54, North America 14, South America 14, Europe 13, Oceania 4, and Africa 1. Similar data, by nations, indicate that China with 28 percent, United States with 13 percent, Burma with 12 percent, Portugal with 7 percent, Bolivia with 9 percent, and Korea with 6 percent have produced three-quarters of the world's tungsten.

Comparable data on consumption of tungsten concentrates are not available; however, the following is believed to be a fairly accurate estimate:

World War II period.—United States 80 percent, Europe 15 percent, all others 5 percent. (Disturbed by large shipments of products from United States to Europe.)

Postwar, Pre-Korea period.—United States 30 percent, Europe 60 percent, all others 10 percent.

Post-Korea to Present.—United States 50 percent, Europe 40 percent, all others 10 percent. (No estimate included for Russia; however, consumption there must be high if reports of industrial expansion are true.)

GEOLOGY

The only important tungsten minerals are:

Mineral	Composition	Formula	Tungsten, percent
Scheelite	Calcium tungstate	$CaWO_4$	63.9.
Powellite	Calcium tungstomolybdate.	$Ca(MoW)O_4$	Not fixed.
Cuproscheelite	Calcium-copper tungstate.	$(CaCu)WO_4$	Not fixed.
Ferberite	Iron tungstate	$FeWO_4$	60.6.
Wolframite	Iron-manganese tungstate.	$(FeMn)WO_4$	60.6 to 60.7.
Hübnerite	Manganese tungstate	$MnWO_4$	60.7.

In the trade, ferberite may contain up to 20 percent of $MnWO_4$, hübnerite may contain up to 20 percent of $FeWO_4$, and wolframite from 80–20 percent $FeWO_4$–$MnWO_4$ to 80–20 percent $MnWO_4$–$FeWO_4$. There is also a tendency to class all three as wolframite and as "black ore."

Difference in the uses for which concentrates of the various mineral types are suitable is discussed in some detail in a later section.

Geologically, tungsten deposits have been classed into the following types: (1) Segregation, (2) pegmatite dike, (3) veins, (4) replacement, (5) contact metamorphic, and (6) placer. The last 4 types are far more important than the first 2, and there are gradations between replacement and contact metamorphic deposits. Thus, the "hard-rock" deposits are essentially all in

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[2] Fuller, T. S., Less Common Elements in the Electrical Industry: Min. and Met., vol. 9, No. 262, October 1928, pp. 451–453.

TABLE 1.—*Salient statistics of tungsten ores and concentrates in the United States, 1939–54*

[Pounds of contained tungsten]

Year	Production	Shipments from mines	Imports for consumption	Consumption	Industry stocks at end of year		
					Producers	Consumers and dealers	Total
1939	3, 429, 047	4, 080, 024	1, 485, 157	(1)	283, 803	2, 667, 738	2, 951, 541
1940	4, 872, 806	5, 062, 199	5, 610, 882	9, 955, 000	93, 618	2, 803, 249	2, 896, 867
1941	6, 420, 303	6, 249, 945	11, 522, 190	16, 699, 000	263, 626	2, 404, 876	2, 668, 502
1942	8, 977, 575	8, 882, 403	14, 326, 470	17, 389, 000	355, 864	3, 416, 438	3, 772, 302
1943	11, 472, 985	11, 368, 295	19, 445, 017	19, 313, 000	458, 586	2, 459, 246	2, 917, 832
1944	9, 764, 647	9, 786, 537	18, 396, 277	19, 165, 000	435, 634	1, 510, 419	1, 946, 053
1945	5, 388, 639	5, 266, 818	4, 773, 861	14, 146, 000	557, 042	3, 784, 429	4, 341, 471
1946	4, 671, 042	4, 942, 282	6, 869, 438	6, 458, 000	285, 865	3, 694, 256	3, 980, 121
1947	3, 026, 470	2, 944, 622	6, 018, 005	7, 812, 000	368, 316	3, 343, 392	3, 711, 708
1948	4, 033, 389	3, 838, 287	7, 548, 101	8, 853, 000	563, 418	5, 284, 901	5, 848, 319
1949	2, 896, 084	2, 631, 506	6, 274, 102	4, 958, 000	827, 045	4, 229, 444	5, 056, 489
1950	3, 965, 040	4, 587, 687	16, 147, 313	6, 597, 000	216, 468	5, 121, 206	5, 337, 674
1951	5, 913, 750	5, 972, 551	6, 376, 513	11, 410, 000	234, 282	4, 037, 502	4, 271, 784
1952	7, 233, 199	7, 243, 589	17, 416, 368	8, 634, 000	208, 300	2, 816, 405	3, 024, 705
1953	9, 272, 513	9, 141, 572	27, 923, 573	7, 734, 000	362, 891	4, 334, 710	4, 697, 601
1954: 1st 6 months	5, 910, 000	5, 564, 000	13, 903, 778	2, 453, 000	726, 000	3, 588, 000	4, 314, 000

[1] Not available.

veins or contact metamorphic-replacement deposits. An erratic distribution of tungsten minerals is typical, especially in quartz veins containing ferberite and hübnerite. Tungsten deposits nearly always are associated with granitic rocks, and there often are contact-replacement deposits near a contact between calcareous sedimentary rock and granitic rock in rather typical tactite zones.

Another classification of tungsten deposits could be used, based on mineral associations. In this respect, there are many deposits in which tungsten is the only valuable mineral; other typical deposits contain tungsten-tin (Bolivia, Portugal, Burma, and others), tungsten-molybdenum (Climax, Colo., and nearly all scheelite deposits), tungsten-antimony (Yellow Pine, Idaho), tungsten-silver-gold (scattered quartz veins), tungsten-copper (Mexico), and tungsten-silver-copper-lead-zinc (Idaho). At operating mines tungsten is produced as a byproduct of molybdenum (Climax, Colo.) and tin (Bolivia, Portugal, England) mining. Also, plants such as the Government tin smelter in Texas recover tungsten from imported tin concentrates. Byproducts of tungsten mining are tin, molybdenum, silver, copper, lead, zinc, and gold. Potential byproducts are manganese and fluorspar. The only cases in which an appreciable effect on the supply of tungsten would result from a change in activity level at mines primarily producing other metals would be molybdenum mining at Climax, Colo., and tin operations in Bolivia.

MINING

Mining methods in use in the United States at tungsten deposits are similar to those in use at mines producing gold and other metals. A few opencut operations are active, but most of the tungsten produced in the United States comes from underground workings. A large quantity of tungsten has been produced at the Atolia, Calif. district from placer deposits.

Tungsten mining at many foreign deposits is characterized by the large output of small-scale, primitive mining operations. Spain, Portugal, Mexico, Brazil, Burma, Siam, and China have much output of this type, along with modern mechanized mining at the larger deposits. The use of small-scale operations has led to many internal problems in some nations; for example, the public water supply at Hong Kong was menaced by unlicensed tungsten operations, and farm labor becomes scarce in Portugal during tungsten booms. The concentrates produced by such operations are often the product of equally primitive hand sorting and hand jigging and frequently require cleaning or other re-treatment after shipment.

MILLING

Modern tungsten mills are of the gravity or gravity-flotation type. Most small mills contain only crushing and grinding equipment, jigs, and tables. The recovery at such plants averages about 60 to 70 percent, and the concentrate produced often contains over 70 percent WO_3. Larger-scale plants now tend to have flotation cells in the circuit in various combinations with the gravity-concentration equipment, permitting recoveries as high as 70 to 90 percent. However, introduction of flotation to attain better recovery created the problem of re-treatment of low-grade flotation concentrates. At some mills these are tabled and combined with gravity concentrates; at others they are shipped to chemical plants for digestion and production of synthetic scheelite.

The domestic tungsten industry, in general, produces concentrates of a high quality suitable for industrial use, while many foreign producers ship a semifinished concentrate from the mines. The production of finished concentrates often involves acid leaching to remove phosphorus, roasting or magnetic separation to remove sulfur, and usually magnetic separation to improve the grade of the product. Some mills, such as that of Tungsten Mining Corp. in North Carolina, use flotation to remove undesirable minerals from tungsten concentrates. Another problem of the producers in some instances is separation of scheelite and wolframite, which is most often done by magnetic separation. The tendency of tungsten minerals, particularly scheelite, to slime in grinding has led to many modifications of mill flowsheets and of mill equipment. Frequent association of scheelite and calcite in tungsten ore has complicated flotation, as both minerals tend to float under the same conditions. The mills of United States Vanadium Co., Nevada Massachusetts Co., Bradley Mining Co. (Ima mine), and Tungsten Mining Corp. (*1, 8, 9, 12*)[3] are examples of typical modern practice. Climax Molybdenum Co. has introduced the Humphrey spiral as a means of recovering byproduct tungsten.

TECHNOLOGY

Tungsten concentrates are used to charge direct to steel, to produce ferrotungsten largely used in steel, to convert to tungsten metal powder (most of which is consumed by the tungsten carbide industry), and to produce tungsten chemicals. Up to World War II, most of the tungsten was used in steel, but at present the greatest consumption is in tungsten powder. The type of concentrate produced must be suitable for the particular use and, in some instances, for the particular plant that purchases the concentrate. This has led to establishment of an elaborate re-treatment plant in New York, equipped with nearly all types of ore-dressing equipment, leaching vats where various solvents (primarily hydrochloric acid) are used to remove phosphorus and other impurities, roasting hearths for removing sulfur and arsenic, magnetic separators for separating scheelite and wolframite and for upgrading concentrates, and sintering, digesting, and precipitating equipment for production of synthetic scheelite.

Synthetic scheelite ($CaWO_4$) is used for the same purposes as other tungsten concentrates. This compound results from a chemical process used for upgrading concentrates low in tungsten and for chemical cleaning of impure concentrates. Tungsten minerals are dissolved by pressure digestion in an autoclave at some plants (near Bishop, Calif.; Conshohocken, Pa.; Henderson, N. C.; and Salt Lake City, Utah) and by lixiviation after sintering or fusing with sodium salts (near Clinton, Md., and Glen Cove, N. Y.). Calcium chloride is added to the solution to precipitate calcium tungstate; in some instances, molybdenum is first removed from the solution by precipitating the sulfide. The autoclave method is said to be less costly than sintering if concentrates containing less than 20 percent WO_3 are treated and more costly if a higher grade feed is used; in either circumstance, the cost is considerably more than production of a tungsten concentrate by gravity-flotation methods.

Ferrotungsten (about 78 percent W), melting base (about 35 percent W), scrap high in tungsten content, and high-purity scheelite (both natural and synthetic) are all used as vehicles for charging tungsten to steel. Ferrotungsten is in general use when class B high-speed steels (6.75 to 22 percent W) are manufactured; however, some firms use only the other tungsten products listed above when class A high-speed steels (less than 6.75 percent W) are produced. Class B steels require a low molybdenum content in the charge; most class A steels require several percent molybdenum. This is important to producers of tungsten concentrates that contain molybdenum; nearly every use of tungsten except charging to class A steel requires a product low in molybdenum.

The scrap tool steel used is often material returned to the plant where it was originally produced. However, in emergency periods scrap drives provide large quantities of mixed tool-steel scrap. Some scrap tungsten carbide is also used by the steel industry.

Scheelite concentrates used for direct charging to steel are normally very high grade and must not contain appreciable quantities of impurities that are undesirable in steel. Purchasing agents for the steel firms usually reorder concentrates of this type from a single supplier to obtain a uniform product known to be satis-

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[3] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

factory. However, the following specification is typical for this use:

		Percent by weight
Tungsten trioxide (WO_3)	Minimum	70.00
Tin (Sn)	Maximum	.10
Copper (Cu)	do	.05
Arsenic (As)	do	.10
Antimony (Sb)	do	.10
Bismuth (Bi)	do	.25
Molybdenum (Mo):		
Class A	Not specified	-----
Class B	Maximum	.80
Phosphorus (P)	do	.05
Sulfur (S)	do	.50
Manganese (Mn)	do	.50
Lead (Pb)	do	.10
Zinc (Zn)	do	.10

Copper, arsenic, antimony, phosphorus, sulfur, and manganese are the impurities that are most often a problem to the producer of concentrates intended for direct charging. Some scheelite contains chemically combined copper, which can be removed only by digestion and precipitation; this is also true of molybdenum, which has been the source of much difficulty to producers in periods when class B steels were dominating the market. Synthetic scheelite, if nodulized, is an ideal product for direct charging, but unnodulized material of this type is not desirable because of high dust losses. Natural scheelite concentrates of the all-slime type are not wanted for the same reason. Concentrates of the black-ore minerals (wolframite) are not satisfactory for this direct charging because of the manganese content.

Ferrotungsten is made in electric furnaces with carbon or silicon as the reducing agent or by the aluminothermic or silicothermic methods. Acid or basic slags are used, depending on the ore. A batch-type operation is required, as the high melting point of ferrotungsten precludes tapping. Because of the high temperatures involved and the control of slag type, producers of ferrotungsten can decrease the quantity of undesirable impurities that pass into the product and can therefore tolerate concentrates containing more of these than would be allowable at a steel plant. The producers also use far more concentrates than any one steel plant and have available several types that can be blended to produce a satisfactory charge. All mineral types of tungsten concentrates are satisfactory for production of ferrotungsten. Producers of mixed-type concentrates can market them without separation at this type of plant, whereas for nearly every other use the scheelite must be separated from the black-ore minerals.

Because of the batch process and the high power requirements, ferrotungsten is a considerably more expensive source of tungsten for steel than concentrates for direct charging. (Recent quotations, converted to price per pound of tungsten contained, are ferrotungsten $3.80 and scheelite concentrates $2.25.) However, for many purposes it is superior and, for some, essential. In general, the higher the percentage of tungsten desired in the steel the greater the need for ferrotungsten. In recent years the trend has been toward the low-tungsten-content (class A) steel, and less ferrotungsten has been used, relative to total tungsten consumption.

The standard specification regarding ferrotungsten by industry (ASTM specification A 144–50) and the Government (National Stockpile Specification P–57a) are the same; they are summarized as follows:

a. Chemical requirements:

Ferrotungsten shall meet the following chemical requirements:

		Percent
Tungsten		70 to 80
Carbon	Maximum	.60
Manganese	do	.75
Phosphorus	do	.06
Sulfur	do	.06
Silicon	do	1.00
Copper	do	.10
Arsenic	do	.10
Antimony	do	.08
Tin	do	.10
Sum of arsenic antimony and tin	do	.20

b. Physical requirements:

Ferrotungsten shall be furnished crushed to a size 1 inch and less.

The consuming firm also specifies the maximum molybdenum content allowable; this is often set at 0.6 percent if class B steels are being produced.

Another branch of industry, which now consumes over half of the total tungsten, is the tungsten-metal powder-tungsten chemical group. Here the first step is to dissolve the tungsten from concentrates, and there is a great variation in the type of plant and the specifications for satisfactory concentrates. Many of the largest plants are operated by firms that originally were primarily concerned with tungsten for lamp filaments. However, the rapid growth of the tungsten carbide industry led to great expansion of these plants. This has created a problem in the industry which becomes critical during defense periods, as these plants were designed to use only high-purity (*very* low molybdenum content) black ore, while satisfactory tungsten carbide can be produced from synthetic scheelite and less pure black ore than are satisfactory for filament wire. The industry is gradually constructing plants designed primarily for the tungsten carbide end use.

The chemical plants are of two general types (1) Those dissolving concentrates with acid and (2) those using basic chemicals as solvents. At the former the tungsten remains as precipitated tungstic acid; at the latter the tungsten is in solution as sodium or potassium tungstate. At both types, several additional steps are required to produce an end product of acceptable purity.

A typical description of the basic-type plant designed to produce very pure products for filament wire follows:

Black ore is dissolved in a batch-type autoclave using sodium hydroxide. The charge is then sent to filters. The sludge or filter cake is sent to storage for possible re-treatment to remove residual tungsten, which is often present in appreciable quantities. The filtrate, containing sodium tungstate, is sent to tanks, calcium chloride is added, and the precipitated calcium tungstate is sent to acid-treatment tanks. Hydrochloric acid is added, and the resulting tungstic acid crystals are washed in tanks. This product is then dissolved with ammonium hydroxide, forming ammonium tungstate. After filtering, this solution is heated; the resulting crystals of ammonium paratungstate are dried on a filter and in steam pans and sent to the reduction plant for charging to the furnaces. The plant water is treated in tanks to remove impurities and returned to the circuit.

At other plants using the same general method, the ore is dissolved in open tanks rather than autoclaves; at still others, it is fused or sintered with sodium carbonate before going to solution. The purification steps often involve several stages of recrystallization as sodium or potassium tungstate, and the final product of some plants is tungstic acid or tungsten trioxide rather than ammonium paratungstate.

The processes used at most of these plants do not recover a high percentage of tungsten from scheelite present in the ores and do not remove molybdenum from the end product. As was mentioned previously, much of the existing plant capacity of this type is used to produce tungsten both for wire and for conversion to carbide. Any impure ore introduced into such a plant may foul the circuit and prevent the production of satisfactory wire. Elaborate precautions are taken to prevent this; each lot of ore is tested both chemically and spectrographically.

The other general-type plant uses acid treatment to disintegrate the ore. Scheelite or synthetic scheelite is the usual feed, as this process is not very satisfactory with black-ore minerals. However, some plants of this type have operated satisfactorily on hübnerite. Ferberite is least suited for acid-type plants.

The usual practice at such plants is to purify the tungstic acid resulting from the first step by several stages of recrystallization. As at the basic-type plants, molybdenum is not removed to any great extent and passes into the final product, unless special steps are taken to remove it. Several plants have recently been designed that will remove molybdenum to the degree needed to produce tungsten carbide, but none in the United States is known to have a process that removes molybdenum to the degree needed to produce satisfactory tungsten wire from concentrates originally containing high molybdenum.

Two general types of impurities are a problem at all chemical-type plants: (1) Those that pass through the process and appear in the end product and (2) those that interfere with the process used but do not contaminate the end product.

The most important elements of the first type are molybdenum, antimony, and, to a smaller extent, bismuth. The most important of the second variety are arsenic, phosphorus, and sulfur. Lead and zinc are less important members of this group. Tin, copper, and manganese, which are detrimental in high-speed steel, are not important at chemical-type plants.

The molybdenum problem has been covered in prior paragraphs. Specifications for concentrates used by producers of filament wire call for a maximum molybdenum content of 0.025 to 0.002 percent.

Antimony attacks the "boats" used for reducing tungsten compounds in the hydrogen-reduction furnaces and is very undesirable in filament wire for electronic purposes. Some firms specify maximum antimony in concentrates at 0.005 percent; others consider as much as 0.05 percent satisfactory.

Arsenic and phosphorus lead to a high loss of tungsten in plant solutions. They also promote the formation of colloidal precipitates where crystalline forms are desired; this reduces plant capacity. Many plants will not tolerate arsenic higher than 0.02 percent and phosphorus more than 0.05 percent; nearly all plants will reject concentrates containing more than 0.25 percent of combined arsenic and phosphorus. As these elements are frequently present in the original ore and in imported concentrates, they present a major problem in the tungsten industry. They are also the most important impurities that are obnoxious to *both* the steel users and the chemical plants.

Specifications in table 2 for the National Stockpile cover concentrates intended for chemical-type plants. This specification is not as rigid as those of some industrial users because of the difficulty in obtaining concentrates so pure and high grade in the quantity demanded by the stockpile objective. Nearly all plant managers prefer concentrates containing 70 percent or more WO_3; some report that scheelite concentrate containing more molyb-

TABLE 2.—*National Stockpile specifications*

			Class I.—Chemical requirements			
			Percent by weight—dry basis			
			Type A, B, or C—ferberite, hübnerite, or wolframite ores and concentrates		Type D—natural scheelite ores and concentrates	Type E—synthetic scheelite precipitates
			(1) For carbide powder	(2) Pure metal		
Tungsten trioxide [1]	(WO^3)	Min	65.00	65.00	65.00	65.00
Tin	(Sn)	Max	1.50	1.50	.10	.05
Copper	(Cu)	do	.50	.50	.10	.05
Arsenic	(As)	do	[2].20	[2].20	.10	.05
Bismuth	(Bi)	do	.50	.50	.25	.25
Antimony	(Sb)	do	.05	.05	.10	.05
Molybdenum	(Mo)	do	.10	.025	.10	.10
Phosphorus	(P)	do	.05	.05	.05	.05
Sulfur	(S)	do	.50	.50	.50	.50
Lead	(Pb)	do	1.00	1.00	.10	.10
Zinc	(Zn)	do	1.00	1.00	.10	.10
Calcium	(Ca)	do	.20	.20		
Manganese+iron	($Mn+Fe$)	do			2.00	.50

[1] Corrected for columbium and tantalum, if present.

[2] Maximum, 0.1 percent arsenic for processed Government-owned material returned under beneficiation contracts.

denum and phosphorus than the maximum listed in table 2 is satisfactory.

Chemicals used for manufacturing tungsten pigments are produced as a coproduct at some plants and are the chief product of one firm. Quantitatively, this is not an important phase of the tungsten industry.

There are two divisions of the tungsten metal powder industry: (1) Hydrogen reduction and (2) carbon reduction. Hydrogen reduction is used to produce metal powder for most tungsten carbide and for all filament wire. Carbon reduction is used when the tungsten-metal powder will be used for welding rods, for coating oil-well tools, and for some tungsten carbide.

Hydrogen-reduced tungsten powder is produced in furnaces containing many long, horizontal tubes. The charge—ammonium paratungstate or tungstic oxide—is placed in iron or nickel boats, which are then pushed into the tubes. Each boat inserted pushes one out the discharge end. Hydrogen flows through the tubes; heat from gas or electricity is applied to the outside. Most plants use 2-stage reduction, which requires 2 furnaces operated at different temperatures or 2 passes through the same furnace; however, at least 1 domestic plant uses 1-stage reduction. The grain size of the metal powder is controlled in the reduction furnaces by variations in temperature, depth of charge in boats, and the time the charge remains in the tubes. A furnace has a much greater (about 300 percent) capacity for producing coarse powder than for fine powder.

Carbon-reduced tungsten powder is in demand for hard-facing welding rods and producing tungsten carbide. Carbon-reduced powder is produced from tungstic acid in crucibles or in furnaces similar to those used for hydrogen reduction. Lampblack is the usual reducing agent in crucibles; natural gas or manufactured city gas is the reducing agent in tubular furnaces. Considerable skill is required to produce a powder low in carbon and with a satisfactory grain size.

Production of tungsten wire, rod, and sheet is based on hydrogen-reduced powder. The powder is compressed, sintered, heated to incipient fusion by passing an electrical current through it, swaged, and drawn or rolled.

Tungsten carbide is largely produced from hydrogen-reduced powder, but recently increasing quantities have been made from carbon-reduced powder. The tungsten-metal powder is mixed with lampblack and heated. The final grain size is very important and is dependent on that of the original metal powder. One firm produces a mixed carbide, $WTiC_2$, by fusion in a bath of nickel. Another produces carbide crystals direct from ore.

Tungsten carbide tools are produced by sintering a mixture of tungsten carbide and cobalt and/or nickel in dies that produce the desired shape. Titanium and tantalum carbides are often blended with the tungsten carbide before sintering, and one firm produces a tungsten-titanium carbide ($WTiC_2$). This rapidly growing industry is constantly developing new techniques, and it is now possible to produce nearly any shape and hold very close tolerances. Methods of grinding and drilling sintered carbides have also improved. Tungsten carbide is used as a core in high-velocity armor-piercing shells. Because of the high cost of the tungsten, research continues for a substitute.

SUBSTITUTES

Molybdenum is substituted for tungsten in some tool steel; however, the probable economic limit is at about present practice. At present, there is no satisfactory substitute for tungsten in carbide tools; however, titanium carbide may soon have similar uses. Research may develop a less expensive substitute for tungsten carbide in high-velocity armor-piercing shells. The introduction of transistors as substitutes for vacuum tubes in electronic uses may result in a reduced demand for tungsten wire.

SECONDARY SOURCES

There is a well-established industry practice of returning scrap tool steel to producing plants. No statistical information is available on the percentage of tungsten from this source used in new heats of steel; however, informal reports and consultation with industry indicate that, in periods of stable demand, well over half of the tungsten charged to steel is in scrap.

Scrap is also saved at producing wire works and either sold or returned to the chemical units. Very little scrap from consumers of wire is returned to industry.

A market is building up for scrap tungsten carbide. One difficulty here is the cost of removing the carbide inserts from the steel shanks or supports. Various practices are used at present, ranging from charging the entire tool into a heat of tool steel to removal of the inserts in special furnaces and treating them chemically to produce synthetic scheelite.

All published Bureau of Mines data on consumption refer to primary tungsten.

USES

The following outline summarizes the uses of tungsten:

- I. Ferrous alloys.
  - (*a*) Straight tungsten tool steels, less than 7 percent tungsten.[4]
  - (*b*) Magnet steels.[5]
  - (*c*) Die steels.
  - (*d*) Hot-work steels.
  - (*e*) High-speed steels, tool steels.
  - (*f*) Low-carbon, high-tungsten steels (8 to 30 percent W).[6]
- II. Nonferrous alloys.
  - (*a*) Cutting-tool alloys.
  - (*b*) Alloys for resistance welding.
  - (*c*) Alloys for electric contacts.
  - (*d*) Jet-engine alloys.
  - (*e*) Alloys for resistance wire.[7]
  - (*f*) High-density alloys.
  - (*g*) Miscellaneous alloys.
- III. Carbides.
  - (*a*) Sintered, with or without other carbides.
  - (*b*) Fused, with or without other carbides.
  - (*c*) Crystals, with or without other carbides.
- IV. Metallic tungsten.
  - (*a*) Wire.
  - (*b*) Rod.
  - (*c*) Sheet.
  - (*d*) Powder.
- V. Chemicals.
  - (*a*) Colors and inks.
  - (*b*) Miscellaneous.

Tungsten is used in die steels, hot-work steels, and high-speed tool steels to impart strength and hardness at high temperatures. Other alloying metals, such as molybdenum, chromium, and vanadium, are always used in addition to the tungsten. Table 3 indicates shipments of the various types as reported by the American Iron and Steel Institute.

The producers of high-speed steels list the following standard types, as shown in tables 4 and 5.

The greatest use of high-speed steel is for high-speed drills, milling cutters, and hacksaw blades. However, it is used in a great number of other applications where hot metals are machined or where the machining results in high temperatures.

The most important nonferrous alloys of tungsten are the cutting-tool, hard-facing alloys containing cobalt, chromium, and tungsten in various proportions. They are said to retain a higher hardness at very high temperatures than the best high-speed steels.

Alloys for resistance welding and heavy-duty electric contacts are important because they are the best products for certain of these applications; however, they do not consume important quantities of tungsten. These alloys contain copper and tungsten or silver and tungsten.

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[4] Essentially no tungsten so used at present. These uses have consumed tungsten in the past but are no longer important.

[5] See footnote 4.

[6] See footnote 4.

[7] See footnote 4.

TABLE 3.—*Shipments of high-speed and tool steel (excluding hollow drill steel)*

[Net tons]

(From American Iron and Steel Institute Annual Statistical Reports)

Grade	Class A high-speed steel						Shipments			
	C (min.)	Cr (max.)	W (max.)	Mo (max.)	V	Co	1953	1952	1951	1950
I	0.60	4.5	6.75	5.5	2.1 max	0.0	11,311	11,821	19,398	6,161
I–b	.90	4.5	6.75	6.5	2.25 min	0.0	566	549	557	154
I–c	.60	4.5	6.75	5.5	2.2 max	3.5 min	85	243	239	572
II	.60	4.5	2.0	9.25	1.3 max	0.0	5,283	4,185	7,504	1,559
II–c	.60	4.5	2.0	9.25	2.2 max	3.5 min	176	214	270	42
III	.60	4.5		9.25	2.2 max	0.0	3,684	2,811	2,421	948
III–c	.60	4.5		9.25	2.2 max	3.5 min	44	25	178	73
Total							21,149	19,848	30,567	9,509
	Class B high-speed steel						Shipments			
	C (min.)	Cr (max.)	W (max.)	Mo	V	Co				
IV	0.55	4.5	19.0	0.0	1.3 max	0.0	2,336	1,601	6,061	6,823
IV–b	.55	4.5	19.0	1.25 max	1.75 min	0.0	259	204	499	546
IV–c	.55	4.5	22.0	1.25 max	2.2 max	3.5 min	841	812	929	1,026
Total							3,436	2,617	7,489	8,395
	Other tool steels									
V	All hot-work steel						14,229	12,719	12,995	8,608
VI	High-chromium (4 precent Cr minimum) die steels						8,619	9,700	12,230	7,168
VII	All other alloy tool steels						53,257	57,103	78,621	40,891
VIII	Carbon tool steels, excluding hollow drill steel						16,941	20,233	34,112	19,038
Total							93,046	99,755	137,958	75,705
Grand total							117,631	122,220	176,014	93,609

TABLE 4.—*Tungsten high-speed steels—"T"*

W (percent)	Cr (percent)	V (percent)	Co (percent)	Symbol	W (percent)	Cr (percent)	V (percent)	Co (percent)	Symbol
18	4	1		T–1	22	4½	1½	12	T–6
18	4	2		T–2	14	4	2		T–7
18	4	3¼		T–3	14	4	2	5	T–8
18	4	1	4	T–4	12	4¼	4¼		T–12
18	4	2	8	T–5	13	4¼	5	5	T–15

TABLE 5.—*Molybdenum high-speed steels—"M"*

Mo (percent)	Cr (percent)	V (percent)	W (percent)	Co (percent)	Cb (percent)	Boron	Symbol
8½	4	1	1½				M-1
5	4	2	6				M-2
6	4	3	6				M-3
4½	4½	4	5½				M-4
5	4½	1½	4	12			M-6
4½	4¼	1½	5½		1¼		M-8
8	4	2					M-10
8½	8¼	1½		3¼			M-11
8	4	1		2½		Added	M-20
8	4	1	2	5			M-30
8½	4	2	2	8			M-34
6	4	2	6	8			M-36
5	4	3	6½	9			M-38
8	4	1¾		8		Added	M-40
4¼	4	1					M-50
4¼	4	2					M-52
4¼	4	3					M-54
4¼	4	4					M-56

Alloys for jet-engine use are now the subject of much research and cannot be considered to have reached a standard type that will remain fixed; however, important quantities of tungsten may be consumed in such alloys.

Several alloys are in use for high-density applications, such as radiation shielding and counterweights.

Tungsten carbide is the most important tungsten-consuming product at present and can be expected to be even more important in the future. It is used for applications requiring great hardness, combined in some instances with stability at high temperatures. For many uses it is combined with carbides of other metals, mainly tantalum and titanium. There are two chemical forms of tungsten carbide, WC and W_2C; the former is present in the sintered carbides, and both are present in fused carbides.

Metallic tungsten is most important in the electric lighting, electronics, and contact fields. Some is used for welding rod, some for shielding radioactive material, and some for X-ray tube components. Tungsten-metal powder is the raw material from which most tungsten carbide is produced at present. The filament of an electric lamp and the distributor "points" of an automobile are the best known uses of metallic tungsten.

Tungsten chemicals are used in dyes, inks, and ceramic frits. An important use of the tungstates is in fluorescent lamps.

STRATEGIC CONSIDERATIONS

The United States has imported tungsten since the first development of a consuming industry. However, important quantities are produced domestically; and, in recent months, domestic production has exceeded consumption. Table 1 indicates the historical situation.

Because potential wartime demands are more than the normal supply, the Government has taken steps to increase domestic production and to build up adequate stocks. As a result of these actions, there should be no shortage during an emergency period.

Domestic production has been stimulated by a purchase program that provides a fixed price of $63 per unit for standard-grade concentrates up to July 1, 1958, or up to a total of 3,000,000 units purchased, whichever occurs first. The DMEA program calls for exploration loans up to 75 percent of the total cost. Foreign production was also increased by long-term purchase contracts and by loans.

The present domestic tungsten program will probably terminate in mid-1956 because of delivery of the 3,000,000 units. Some foreign contracts have expired, and in at least one instance (Korea) this has led to closing the mines.

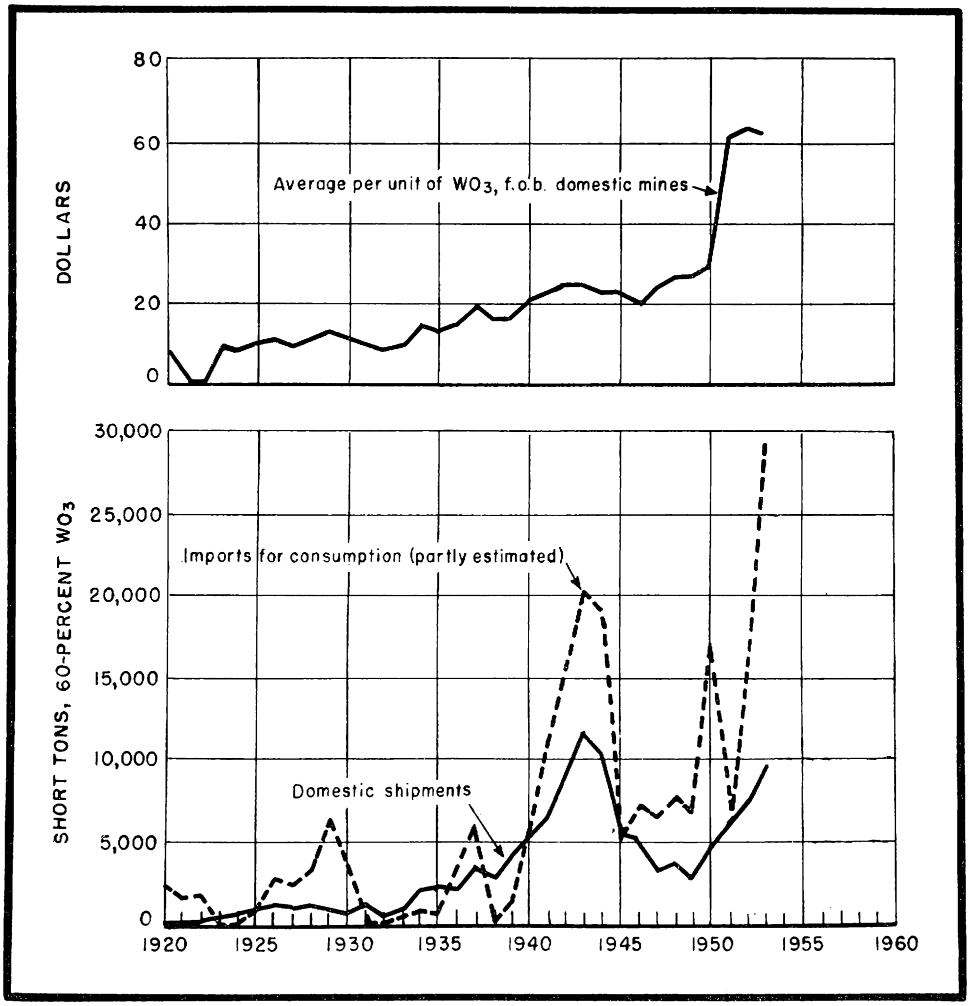

FIGURE 1.—Trends in Shipments, Imports, and Average Price of Tungsten Ores and Concentrates, 1920–53.

PRICES

Figure 1 indicates the historical price situation for tungsten concentrates.

TARIFF

The import duties on tungsten products are shown in table 6.[8]

Table 1 indicates the magnitude of imports of tungsten in concentrates; ferrotungsten imports in recent years have contained the following:

	Thousand pounds, tungsten
1950	1,380
1951	2,018
1952	479
1953	603

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[8] House of Representatives, 82d Congress, 1st session: Rept. 1152.

TABLE 6.—*Import duties on tungsten products*

Paragraph	Commodity	Tariff rate in— Act of 1930	Tariff rate in— 1948	Tariff rate in— 1951
		Cents per pound of metallic tungsten		
302 (c)	Tungsten ore and concentrate	50	[1] 38	[2] 50
		Cents per pound of tungsten content plus percent ad valorem		
302 (g)	Tungsten metal, tungsten carbide and mixtures or combinations containing tungsten metal or tungsten carbide, all the foregoing in lumps, grains, or powder.	60 plus 50 percent.	42 plus 25 percent.[3]	42 plus 25 percent.[3]
	Tungstic acid and all other compounds of tungsten, n. s. p. f.	60 plus 40 percent.	42 plus 25 percent.[3]	42 plus 25 percent.[3]
302 (h)	Ferrotungsten, ferrochromium tungsten, chromium tungsten, chromium cobalt tungsten, tungsten nickel, and all other alloys of tungsten, n. s. p. f.	60 plus 25 percent.	42 plus 12½ percent.[3]	42 plus 12½ percent.[3]
		Percent ad valorem		
316 (b)	Forms containing more than 50 percent tungsten or tungsten carbide:			
	Ingots, shot, bars, or scrap	50	[3] 30	[4] 25
	Wire	60	[3] 40	[4] 30
	Sheets or other forms	60	[3] 40	[4] 30

[1] General agreement (Geneva). Rate effective May 22, 1948.
[2] Effective Dec. 11, 1950.
[3] General agreement (Geneva). Rate effective Jan. 1, 1948.
[4] General agreement (Torquay). Rate effective June 6, 1951.

NOTE. Public Law 869, 81st Cong., suspended the duty on scrap metal (including tungsten) until July 1, 1951. This suspension was continued to July 1, 1952, by Public Law 66, 82d Congress.

RESEARCH AND DEVELOPMENT

In recent years nearly every phase of the tungsten industry has been the subject of considerable research. Special emphasis has been placed on ore dressing, the effect of impurities on products, methods for removing trace impurities, and development of high-temperature alloys. The geology and ore reserves of tungsten deposits have also been investigated. The Bureau of Mines has concentrated its research on ore dressing and hydrometallurgy, especially as regards low-grade ores.

Development of new products has been pushed by many firms; perhaps the most outstanding recent developments have been in pure tungsten articles, such as welding electrodes and carbide, and high-density products.

OUTLOOK

TRENDS

Many trends reflected in tungsten products are masked by defense production now and in the past; however, some are so well established that they can be used to estimate future developments. The relative use of class A (6.75 percent W, maximum) and class B (22 percent W, maximum) high-speed steels has shifted from 36 percent class A in 1944 to 47 percent in 1948, 80 percent in 1951, and 86 percent in 1953. It seems almost certain that industry will continue to use large quantities of class A steel, which will result in saving much tungsten. If 20,000 tons of high-speed steel is assumed and 6 to 18 percent W is used as typical of the tungsten content of class A and B, respectively, the 1944 ratio would require 5,472,000 pounds of tungsten and the 1951 ratio only 3,360,000 pounds.

Another well-established trend is the pronounced shift from high-speed steel to tungsten carbide when referred to total consumption of tungsten; this in part reflects the changed steel type mentioned above and in part the effect of greatly increased use of tungsten carbide. In 1944 steel took 72 percent of the total tungsten concentrates, which contrasts to 26 percent in the second quarter of 1954.

Development of alloys for jet engines may increase tungsten requirements, and development of a substitute for tungsten carbide in armor-piercing shells may greatly decrease requirements; none of these can be considered a known factor at this time.

NEW USES

New uses for tungsten products will no doubt be developed; however, none is known to be advanced to a stage where predictions of substantial future use can be made. It seems probable that the carbides in particular will be extended to many commodities not produced at present and that tungsten and its alloys will appear in articles designed for high-temperature applications.

GENERAL

As productive capacity at mines has been stimulated by defense programs to a point where industrial needs at present levels and deliveries to the National Stockpile can both be supplied, termination of the stockpile program apparently will lead to a surplus of productive capacity. However, ore reserves as now known will not support this capacity to produce for extended periods; development programs will be needed; and, in many instances, such work is lagging. Without aggressive exploration and development, it seems probable that in less than 10 and 20 years, respectively, domestic and foreign mines will be depleted to the extent that 1953–54 production rates could not be duplicated.

Actual production in specific areas is difficult to predict because of many nontechnical factors, such as tariffs, taxes, and wage rates. World output for the area, exclusive of Russia and China, probably will fall in the range of 15 to 30 million pounds of tungsten a year during periods when no actual war is in progress.

United States consumption doubtless will gradually increase; 20 years from now, 12 to 16 million pounds per year may be used.

PROBLEMS

The sporadic nature of demand and price now give particular concern to domestic producers because of the anticipated end of the Domestic Tungsten Program in 1956 and because of the continual fear that Chinese concentrates will again appear on world markets at very low prices. The following problems are also important to producers of concentrates:

1. Because large capital investments are difficult to justify, there is little incentive to develop several years' ore reserves.
2. Rapid amortization of mine and mill plant leads to high fixed charges.
3. The grade of ore worked in periods of peak demand is very low; this introduces technical problems at mills and, to a small extent, at mines.
4. Consumers' specifications change. For example, the molybdenum content of scheelite concentrates is not important to steel plants now, but in the recent past this prevented the sale of concentrates from some mines.

Problems that concern consumers of tungsten concentrates are:

1. Uncertain supply and price; possible future dependence on unreliable foreign sources.
2. Uncertain supply of the type of concentrates needed for a specific plant. This is least important at plants manufacturing ferrotungsten and most significant at those producing tungsten in metallic form.
3. Trace-element impurities have an important effect on end products; this is particularly true at plants producing wire.
4. Government orders in defense periods are so large in relation to normal demands that sudden changes in the amounts of such orders create a need for corresponding changes in plant and labor force.

Problems important to the Government are:

1. Providing for an assured supply of tungsten for defense needs.
   - *a.* From domestic sources.
   - *b.* From imports. This introduces the problem of a supply for other countries.
   - *c.* From stockpiled material. The proper product distribution and the specifications for the various products are difficult to determine.
2. Providing for an assured supply for future years
3. Providing for industrial capacity to produce defense items.

A problem mutual to Government and industry is maintaining present United States technological leadership in mining, processing, and production of products.

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URANIUM

By

John E. Crawford [1]

NATURAL URANIUM, whose immense potentialities were unknown 15 years ago, is probably the most talked about metal in the world today. Although technical developments in the manufacture of atomic weapons are of paramount importance, the construction of prototype uranium-fueled, electrical-power-generation stations is also significant. Commercial nuclear power may become commonplace in 10 years. In addition, there is increasing interest in the application of radioisotopes to industrial, agronomic, and medical problems; and the effect of radioactivity on man and his environment is a vital subject, important to all inhabitants of the earth.

Summary

Natural uranium metal consists of three radioactive isotopes: 99.3 percent U–238; 0.7 percent U–235; and 0.006 percent U–234.

The earth's crust averages about 3 grams of uranium per ton of rock. It is distributed more abundantly throughout the earth's crust than antimony, bismuth, cadmium, gold, mercury, and silver.

There are over 100 uranium-bearing minerals. The three known primary uranium *ore* minerals are pitchblende, uraninite, and davidite. The six known secondary uranium *ore* minerals are carnotite, tyuyamunite, torbernite and meta-torbernite, autunite and meta-autunite, uranophane, and schroeckingerite.

The Colorado Plateau area of Colorado, Arizona, Utah, and New Mexico is the largest domestic source of uranium. The mineralization is primarily carnotite and pitchblende deposited in the sandstones of the Morrison formation. Processing mills in Colorado, New Mexico, and Utah treat the uranium ore by acid or carbonate leaching, after which the uranium is recovered by one of several classified techniques.

Some uranium is being recovered from phosphatic raw material in Florida as a byproduct of normal phosphatic fertilizer and phosphatic chemicals production.

Uranium ore concentrates are treated at Atomic Energy Commission Feed Materials centers (1) to recover the orange oxide of uranium, (2) to convert the orange oxide into uranium tetrafluoride, (3) to convert part of the uranium tetrafluoride into uranium hexafluoride, and (4) to reduce the balance of the uranium tetrafluoride to uranium metal.

The uranium hexafluoride and uranium metal produced at Feed Materials centers are transferred to AEC Fissionable Materials centers for further refinement. Here, fissionable uranium–235 is recovered from the uranium hexafluoride by gaseous diffusion, and fissionable plutonium is produced from the uranium metal by neutron bombardment in reactors.

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[1] Commodity-industry analyst, Bureau of Mines.

The predominant use of uranium at present is in the production of fissionable material for inclusion in bombs and other weapons for the national defense program.

The radioactive element is being consumed in reactor-development research, particularly with respect to the use of reactors for industrial power and nuclear propulsion and indirectly in the production of radioisotopes for medical and industrial use.

In January 1955 the AEC announced that it had established schedules of basic charges for available materials, including uranium, and services needed by private atomic energy developers and users in the United States.

The future market for uranium through March 1962 is guaranteed under the AEC ore-buying schedule. Thus, while the demand for uranium for the next 8 years is known, long-range requirements cannot be as easily predicted. However, the outlook would appear favorable in the face of rapid development of research in the use of nuclear reactors for industrial power purposes and propulsion units.

Industry, in developing peaceful uses for atomic energy, faces the indeterminate problems of price and future availability of uranium metal, fissionable uranium–235, and plutonium.

Of immediate concern is the application of materials to reactor structural and coolant requirements. Radiation and corrosion resistance are prime prerequisites of materials for constructing reactors.

Health and safety precautions will become subjects of increasing interest as power reactors are established in or adjacent to metropolitan areas. Enactment of severe Federal, State, and local health-safety legislation could impose property and insurance requirements at prohibitive costs upon nuclear-power-conscious industry.

BACKGROUND

GENERAL REVIEW

Uranium was discovered by M. H. Klaproth in 1789 during his investigation of the mineral pitchblende and was named after the planet Uranus. It was 53 years later, however, before E. Peligot prepared metallic uranium in laboratory experiments.

The radioactive element uranium is placed with the actinide series in the Periodic Table. It is one of the heaviest metals, is hard, and appears nickel-white; its atomic number is 92 and atomic weight 238.07.

All natural uranium is radioactive and consists of three isotopes. The proportions are: 99.3 percent U–238, 0.7 percent U–235, and 0.006 percent U–234.

OCCURRENCE

DISTRIBUTION

Uranium is widely distributed throughout the earth's outermost strata. The crust of the earth averages 0.0003 to 0.0004 percent uranium, or about 3 grams per ton of rock. Antimony, bismuth, cadmium, gold, mercury, and silver, not considered particularly rare, are not present in as large quantities as uranium (*9*).[2]

MINERALS

Uranium metal never occurs in its pure form in nature but is always present in combination with other elements, forming a uranium-bearing mineral. There are over 100 uranium minerals. Some occur as primary minerals and many others as secondary minerals. Primary minerals are those that have not been altered since their formation by igneous action. Secondary minerals are those that are formed as a result of decomposition of the original primary mineral.

The three known primary uranium *ore* minerals are (*10*):

Uraninite	Crystalline uranium oxide.
Pitchblende	Amorphous uranium oxide.
Davidite	Rare earth - iron - titanium-uranium oxide.

The six known secondary uranium *ore* minerals are (*10*):

Carnotite	Potassium-uranium vanadate.
Tyuyamunite	Calcium-uranium vanadate.
Torbernite and meta-torbernite.	Hydrous copper - uranium phosphates.
Autunite and meta-autunite.	Hydrous calcium - uranium phosphates.
Uranophane	Hydrated calcium - uranium silicate.
Schroeckingerite	Complex hydrated sulfate, carbonate and fluoride of calcium, sodium, and uranium.

DEPOSITS

Listed below are known uranium deposits, all of which have geologic significance, and some of which have economic significance. Those deposits now being actively mined for their uranium content, either primarily or as a coproduct, are indicated by an asterisk. Some of the deposits so noted in the United States are not extensive and may soon be mined out.

Locality:	*Deposit*	*Mineralization*
Africa:		
Belgian Congo	*Shinkolobwe mine	Pitchblende and uraninite.
Mozambique	Tete district	Davidite.
Southern Rhodesia	Rhodesian copper belt—Roan Antelope, Rokana, Mufilira, N'Changa, and N'Kana mines.	Uraninite.
Union of South Africa	*Witwatersrand	Uraninite.
Australia:		
Northern Territory	*Rum Jungle	Do.
South Australia	*Radium Hill	Davidite.
Canada:		
Northwest Territory	*Eldorado mine	Pitchblende and uraninite.
Ontario	Blind River District	Uraninite and brannerite.
Saskatchewan	*Beaverlodge area	Pitchblende and uraninite.
Europe:		
Czechoslovakia	*Joachimsthal region	Pitchblende and uraninite.
France	*Massif Central region	Autunite, torbernite, parsonsite, and pitchblende.
Great Britain	Devon and Cornwall	Pitchblende.
Portugal	*Urgeirica mine	Autunite, uranophane, torbernite, and pitchblende.

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

Locality—Continued	*Deposit*	*Mineralization*
United States:		
Arizona	*Globe district	Carnotite, tyuyamunite, uranophane, and uraninite.
California	*Kern County	Autunite, torbernite.
Colorado	Garo	Carnotite and tyuyamunite.
Do	*Front Range	Pitchblende.
Colorado, Utah, Arizona, and New Mexico.	*Colorado Plateau	Carnotite, torbernite, uranophane, and pitchblende.
Florida	*Bone Valley formation	Uranium associated with fluorapatite.
Montana	*Boulder area	Torbernite, autunite, and pitchblende.
Nevada	Lander County	Torbernite.
New Mexico	*Colorado Plateau (Grants district)	Uranophane and tyuyamunite.
Pennsylvania	*Mauch Chunk	Carnotite.
South Dakota and Wyoming.	*Black Hills area	Do.
Utah	*Marysvale district	Autunite, torbernite, schroeckingerite, uranophane, and pitchblende.
Do	*Colorado Plateau (San Rafael Swell area).	Uranium in asphaltites.
Washington	Spokane Indian Reservation, Stevens County.	Autunite, torbernite.
Wyoming	*Powder River Basin	Carnotite.
Do	*Wind River Basin	Uranocirite, phosphuranylite, and uranospinite.

PRODUCTION

RAW MATERIALS

In September 1954 the Engineering and Mining Journal announced the names of what it said were the 20 largest uranium ore producers in the United States (*5*). All producers listed operate on the Colorado Plateau. Their production may be from one mine or several. The companies are:

Anaconda Copper Mining Co.
Arrowhead Uranium Co.
Bunker & Co.
Climax Uranium Co.
Consolidated Uranium Mines, Inc.
Cooper & Bronson
Dulaney Mining Co.
Walter Duncan
E & M Mining Co.
Garwood & Gerlach
Haystack Mountain Mining & Development Co.
Homestake Mining Co.
La Salle Mining Co.
Frank Nacheenbetah
Navajo Uranium Co.
Vernon Pick
U. S. Vanadium
Utex Exploration Co.
Vanadium Corp. of America
F. W. Williams.

The following industry-operated uranium mills in Colorado, New Mexico, and Utah, process uranium ore to produce a uranium concentrate for shipment to refineries:

Mill:	*Location*
Anaconda Copper Mining Co.	Blue Water (Grants), N. Mex.
Climax Uranium Co	Grand Junction, Colo.
Galigher Co. (Government owned.)	Monticello, Utah.
Kerr-McGee Oil Industries, Inc.	Shiprock, N. Mex.
United States Vanadium Co.	Rifle, Colo.
Do	Uravan, Colo
Vanadium Corp. of America.	Durango, Colo.
Do	Naturita, Colo.
Vitro Chemical Co	Salt Lake City, Utah.

The recovery of uranium from Colorado Plateau carnotite-type ore, the major source of domestic uranium, involves extraction of the uranium minerals by either acid or carbonate leaching, with subsequent extraction of uranium from the solution by one of several methods, the details of which are classified (*5*).

Small quantities of uranium concentrates are being recovered from Florida phosphate material by solvent extraction and precipitation methods. Uranium occurs with the mineral fluorapatite in the economic land-pebble phosphate deposits of the Plant City-Bartow-Lakeland district of west central Florida. Companies actively engaged in the recovery of low-grade uranium from phosphatic material as a byproduct during the course of their normal activities in the production of phosphatic fertilizers and chemicals are (*2*):

Name:	*Location*
Blockson Chemical Co	Joliet, Ill.
Bonnie Chemical Plant, International Minerals & Chemical Corp	Bartow, Fla.
Texas City Chemicals, Inc	Texas City, Tex.
Virginia-Carolina Chemical Corp	Nichols, Fla.

FEED MATERIALS

The uranium concentrates recovered from the raw ore at the mills are refined by chemical treatment. The steps involved in this procedure, known as the AEC Feed Materials Program, comprise processing of the mill concentrates to recover the orange uranium oxide (UO_3); conversion of the orange oxide into uranium tetrafluoride (UF_4); conversion of part of the uranium tetrafluoride to uranium hexafluoride (UF_6) for feed to the uranium–235 concentration plants; and reduction of the balance to uranium metal for plutonium manufacture (*1*).

Industrial firms managing major plants for the refining of uranium concentrates are Mallinckrodt Chemical Works, St. Louis, Mo., and National Lead Co. of Ohio, Fernald, Ohio.

FISSIONABLE MATERIALS

The AEC Fissionable Materials Program (*1*, *2*) involves production of uranium–235 and plutonium from uranium hexafluoride and uranium metal, respectively. Uranium–235 is recovered from uranium hexafluoride by gaseous diffusion, a method of isotope separation. Two plants for concentrating U–235 by gaseous diffusion are in operation, and a third is nearing completion. The plants, all industry-operated, are:

Operator:	*Location*
Union Carbide & Carbon Corp	Oak Ridge, Tenn.
Do	Paducah, Ky.
Goodyear Tire & Rubber Co. (under construction)	Pike County, Ohio.

Natural uranium metal from refineries is required to produce plutonium, the man-made radioactive element, which, like uranium–235, is consumed in nuclear weapons. This transformation of natural uranium to plutonium takes place in nuclear reactors designed for the purpose. The two industry-operated facilities now producing plutonium are General Electric Co., Hanford, Wash., and E. I. du Pont de Nemours Co., Inc., Savannah River, Aiken, S. C.

STATISTICS

Production statistics are not available for security reasons. The United States is not, however, self-sufficient in uranium resources and imports sizable amounts of uranium ores and concentrates from Belgian Congo, Canada, Union of South Africa, and possibly other countries.

CONSUMPTION AND USES

NONENERGY

Consumption of uranium in the United States for nonenergy purposes can be measured in pounds of contained U_3O_8 and is negligible. Nonenergy uses are in the chemical, ceramic, and electrical industries. Total consumption in 1954 was approximately 2,500 pounds of contained U_3O_8.

ENERGY

The use of uranium in the manufacture of nuclear weapons is, of course, an established fact. Since successful testing of the first atomic bomb at Alamogordo, N. Mex., on July 16, 1945, most of the uranium produced has directly or indirectly been concerned with the manufacture of atomic bombs and, more recently, diversified nuclear weapons.

Smaller quantities of uranium are being used in research, particularly with respect to reactor development and its application to industrial power problems, nuclear propulsion units for aircraft and submarines, and possibly other forms of transportation (*2*).

Indirectly, uranium is consumed in producing radioisotopes, a reactor product receiving widespread application in radiotherapy and more recently in industrial tracer applications. Radioisotopes are produced in a reactor at Oak Ridge and are a byproduct of the operation of research reactors at other AEC facilities, such as Brookhaven and Argonne National Laboratories.

PRICES

RAW MATERIAL

Uranium ores mined in the United States must be sold to the AEC or one of its licensed buyers. The AEC offers guaranteed minimum prices for carnotite-type ores of the Colorado Plateau area, assaying 0.10 percent uranium (U_3O_8) or more. The AEC also purchases the vanadium content of the ore with certain limitations; allows for development and haulage costs; and offers premiums for high-grade material. Circular 5 (revised), published by the AEC, describes specifications that must be met to obtain these benefits. A bonus, effective through February 1957, described in Circular 6, provides an allowance for initial production from new mining properties recovering Colorado Plateau type ore.

The AEC will purchase other types of uranium ores and ores from other locations based on the schedule for Colorado Plateau type ore. The uranium ores included in this category are: Carnotite ores of the Black Hills; limestone ores of the Grants district, N. Mex.; ores of the Marysvale, Utah, district; pitchblende ores of the Colorado Front Range area; complex ores from the Idaho, Montana, Wyoming, Nevada, and southern Arizona areas; and copper-uranium ores.

Guaranteed minimum prices, effective through March 31, 1962, for carnotite-type and roscoelite-type uranium ores of the Colorado

Plateau area assaying 0.10 percent U_3O_8 and more and meeting required specifications are as follows (AEC Circular 5, revised):

U_3O_8 assay, percent:	*Payment per pound of U_3O_8 contained in ore*
0.10	$1.50
.11	1.70
.12	1.90
.13	2.10
.14	2.30
.15	2.50
.16	2.70
.17	2.90
.18	3.10
.19	3.30
.20 and more	3.50

For every pound of V_2O_5 contained in such ore, up to a maximum of 10 pounds V_2O_5 for each pound of U_3O_8, the AEC pays $0.31. A premium of $0.75 is paid for each pound of U_3O_8 in excess of 4 pounds U_3O_8 per short ton of ore and an additional $0.25 per pound for each pound in excess of 10 pounds U_3O_8 per short ton.

A development allowance of $0.50 per pound of U_3O_8 is offered; and a haulage allowance of $0.60 per ton per mile from mine to purchase depot, up to 100 miles distance.

METAL

In January 1955, the AEC announced that it had established schedules of basic charges for available materials, including uranium, and services needed by private atomic energy developers and users in the United States. The AEC stated that the schedules contain classified information and hence will be made available only to properly cleared persons. The schedule sets the rates which domestic licensees under the Atomic Energy Act of 1954 must pay the AEC for natural uranium, thorium, heavy water, uranium–235, uranium–233, or plutonium as used in nuclear reactor and other enterprises for peacetime applications of atomic energy. Uranium–235, uranium–233, and plutonium may only be leased, not sold, to licensees.

OUTLOOK

Nonenergy uranium requirements probably will not increase significantly. Allocation of uranium for nonenergy purposes is the responsibility of the Atomic Energy Commission, as directed under the Atomic Energy Act of 1954.

Because the domestic uranium ore-buying schedule is guaranteed through March 31, 1962—presumably to meet military weapons requirements—uranium-ore producers can for the immediate future expect a stable market. Uranium demand after 1962 cannot be foreseen, although in every respect the future seems most encouraging.

Although the Nation's defense effort requires uranium for weapons stockpiling, other practical uses for nuclear energy are being developed by Defense Department and AEC personnel, such as have been recently publicized with sea trials of the atomic-powered submarine *Nautilus*. Propulsion units for surface vessels and aircraft are also being studied, as well as "package power reactors" for use at inaccessible military bases. Should such applications for nuclear energy become widespread in the next 5 to 10 years, they in themselves might offset any decline in demand for uranium that might develop in weapons production.

A more promising outlook for an increased demand for uranium may be in the field of industrial power. Great strides have been made in the preliminary investigations of nuclear reactor application to industrial power generation. The 60,000-kw. power reactor being constructed at Shippingsport, Pa., will be the first large-scale experimental nuclear power plant developed in the United States. Operation of this unit will provide many valuable data relating to construction of economic nuclear power plants. Studies by the AEC and by large industrial firms indicate that within 10 years nuclear power will be competitive, at least in high-cost power areas (*8*).

The Atomic Energy Act of 1954 will provide stimuli to interest industry in the nonmilitary uses of uranium and its unique properties.

PROBLEMS

Industry, in attempting to utilize the great energy potential of uranium, faces indeterminate long-range problems of availability and price. The constant threat of a major war makes doubtful the future supply of fissionable material for industrial purposes. Should industry build atomic power reactors and a war erupt thereafter, there is the possibility that enough uranium might not be available to fuel the power reactors, with a subsequent loss of vital power for military and essential civilian requirements.

Some immediate problems of industry and the Armed Forces in developing power and propulsion reactor systems relate to efficient structural and coolant materials. It is significant that some materials most difficult to handle have proved most applicable to reactor construction. Metallurgical research is necessary to determine the most suitable substances, their resistance to radiation and corrosion, and other peculiar effects nuclear fission might have upon them. Reduction of mass in shielding, required to protect persons from the radioactivity of the reactor fuel system, would appear to be exceedingly important, if reactors are to have practical use in propelling planes, automobiles, and trains. Large tonnages of cement or lead probably would be impractical for shielding purposes in light conveyances.

Also important are the property and insurance requirements involved in a central power station utilizing fissionable material for fuel. Federal, State, and local legislation may necessitate such expensive property and insurance precautions as to make the costs of an atomic power plant prohibitive.

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VANADIUM

By

Hubert W. Davis [1]

FOR MANY YEARS vanadium production in the United States yielded byproduct uranium; but, with the advent of atomic energy, the enormous demand for uranium has made vanadium the byproduct in the United States. In fact, the latter's abundance poses an interesting problem.

Summary

Since the Atomic Energy Commission entered the carnotite region of the Colorado Plateau for procuring uranium, vanadium supply from the United States output plus imports from Peru has exceeded industry requirements, and the surplus has been placed in the National Strategic Stockpile. The possibilities for utilizing this larger supply of vanadium offer an attractive field of research.

Vanadium supplies are normally ample to meet United States requirements, but when defense or war needs call for substitution of vanadium for scarcer metals the expanded demand results in a severe shortage.

The possible recovery of uranium through the large-scale processing of phosphate rock may carry with it a substantial byproduct output of vanadium. A potential source of vanadium, though the cost may be high, is the titaniferous iron ores of the MacIntyre mine in Essex County, N. Y., which is now being worked for titanium.

Most steels in which vanadium is used as an alloying element require only a few pounds per ton, hence a little goes a long way. Development of major applications involving vanadium alloys where vanadium is the primary constituent would result in a substantial outlet for some of the surplus production.

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[1] Commodity-industry analyst.

BACKGROUND

SIZE OF DOMESTIC INDUSTRY

The United States is the largest producer and consumer of vanadium in the world.

The total output of vanadium in ore in the United States from 1910—the year of first commercial production—through 1947 has been 46,533,000 pounds. Figures on production for 1938 through 1947 are shown in table 1. For security reasons, publication of later figures has been suspended at the request of the Atomic Energy Commission.

TABLE 1.—*Vanadium in ores and concentrates produced in the United States, 1938–47* [1]

Year	Pounds	Year	Pounds
1938	1, 613, 155	1943	5, 586, 492
1939	1, 984, 068	1944	3, 527, 054
1940	2, 162, 916	1945	2, 963, 913
1941	2, 513, 051	1946	1, 272, 148
1942	4, 439, 130	1947	2, 117, 962

[1] Data for 1940–47 are receipts at mills and Government purchasing depots.

GEOGRAPHIC DISTRIBUTION OF INDUSTRY

Vanadium is widely distributed in minute quantities throughout the crust of the earth, but deposits in which it is in high enough concentration to justify mining and recovery are few. Currently, world production of vanadium ores is limited almost entirely to four countries—Northern Rhodesia, Peru, South-West Africa, and the United States.

The center of vanadium mining in the United States is the Colorado-Utah region, but small outputs are made in Arizona, Nevada, New Mexico, South Dakota, and Wyoming. Some vanadium is recovered as a byproduct of phosphate rock mined in Idaho; it is also recovered in the United States as a byproduct of chrome ore and of petroleum residues.

Vanadium oxide, ammonium metavanadate, and ferrovanadium are produced at a number of localities in the United States.

Peru ranks second to the United States as a producer of vanadium ore. The famous Mina Ragra mine in the Andes near Ricran, Department of Junin, has been an important source since 1907, when production was begun.

In Northern Rhodesia fused vanadium oxide is produced as a byproduct of lead-zinc ores at Broken Hill.

Vanadium is recovered as a byproduct of the lead vanadates of the Otavi region in northern South-West Africa.

Vanadium occurs in small deposits widely scattered in the Provinces of Córdoba, Mendoza, and San Luis, Argentina, and a small quantity of ore is mined annually.

RESERVES

The principal world sources of vanadium have been the deposits of vanadium sulfide in Peru, the deposits of vanadium-bearing sandstone in the United States, and the vanadate deposits in Northern Rhodesia and South-West Africa. Minor quantities of vanadium have been obtained from other vanadate deposits, from asphaltites and the ash of certain oils and coals, as a byproduct from phosphate rock, and from certain iron and chrome ores and bauxites.

The following information on reserves of vanadium in the United States was prepared by the Bureau of Mines and Geological Survey and is quoted from a report on the mineral position of the United States, published in the hearings before a subcommittee of the Committee on Public Lands, United States Senate, 80th Congress, First Session, 1947.

> The vanadium-bearing sandstone deposits are widely distributed in adjacent parts of Colorado, Utah, Arizona, and New Mexico, and they have yielded about 90 percent of the domestic vanadium production. The average grade of ore mined in recent years ranged from 1.5 to 2 percent V_2O_5. Individual deposits have a wide range in size, but most of them are small; many contain only a few hundred tons of ore each, though some contain many thousand tons. On the basis of geologic studies, the total amount of vanadium contained in these deposits is thought to be large, but because of sporadic local distribution of the deposits and their range in size, the practice of operators is to extend exploration and development only a relatively short distance ahead of mining faces, and for this reason reserves in the usual sense are relatively small. An estimate of available reserves, including inferred reserves which will probably be found with continued mining, and which perhaps can be mined profitably under conditions prevailing in 1943, total several years' supply at the 1943 rate of mining.
>
> Some base metal and precious metal deposits in southwestern United States contain small quantities of vanadates of lead, copper, and zinc. As the vanadates are irregularly distributed in these deposits and the grade of the ore is low, they have produced only as a byproduct and in a desultory fashion. It is unlikely that these deposits will yield significant amounts of vanadium either under peacetime or wartime conditions.
>
> Phosphate rock, mined at Conda, Idaho, for conversion to superphosphate, contains about 0.30 percent V_2O_5. Since 1941 some of the vanadium has been recovered as a byproduct. Reserves of phosphate rock in Idaho and Wyoming are large, and it is likely that

this source will continue to yield vanadium in the future. In addition many millions of tons of vanadiferous shale associated with the phosphate rock contain from 0.70 to 1.30 percent of V_2O_5. No vanadium has been produced from these deposits and it is not certain that it can be produced profitably under conditions such as those prevailing recently. Presumably much vanadium could be obtained from this source under more favorable prices and improved technology.

Vanadiferous shale similar to that in Wyoming and Idaho has been found in the Camp Floyd district, Utah, but the grade is low and the tonnage of indicated reserves small.

Most titaniferous magnetite deposits contain some vanadium, the amount ranging from 0.1 to 0.5 percent of V_2O_5. Although the grade is so low that these deposits cannot be worked for vanadium alone, they represent a large reserve of vanadium, some of which might be obtained as a byproduct. Perhaps the most readily available source is the Lake Sanford, N. Y., deposit, which is now being worked for titanium. The magnetic iron-ore concentrates being produced from this ore contain about 0.7 percent of V_2O_5 * * *. Most of the other titaniferous magnetite deposits in the United States appear to be less favorable as a possible source of byproduct vanadium, either because of the complex metallurgy involved in their treatment or because the vanadium content is too low to permit profitable extraction.

Vanadium-bearing green mica is associated with flake-graphite deposits in Alabama. The average vanadium content of the micaceous rocks and the tonnage of the material present have not been determined.

The mine at Mina Ragra, Peru, has been and still is an important source of vanadium. Information on reserves is not available, but it is understood that on the basis of a rate of production of about 600 tons annually they will suffice for many years.

Figures on reserves of vanadium ore in Northern Rhodesia are not available. According to the Rhodesia Broken Hill Development Co., Ltd., the only producer in Northern Rhodesia:

> It is becoming increasingly difficult to maintain a continuous supply of vanadium-bearing material with a grade suitable for leaching, because of the gradual exhaustion of accumulated surface dumps. Much experimental work has been carried out in an effort to produce a leach grade concentrate from the low grade mixed fines tailings and recent flotation tests have been encouraging.

Full-scale production of vanadium by the company was continued until April 1952, when the plant was shut down. Experimental leaches were conducted later in the year on flotation concentrates from mixed fines tailings, and a small additional production was made. All vanadium-bearing ore produced in 1953 was stockpiled with the mixed fines tailings, pending final evolution of a process for recovering both the zinc and vanadium contents of this material. Tests on the problem continued to give encouraging results.

Information on reserves of vanadium in South-West Africa is lacking.

TECHNOLOGY

Vanadium occurs in igneous rocks as well as in sedimentary formations. Alteration of the original mineral is very common, and alteration products are numerous. Patronite (vanadium sulfide), for example, oxidizes to the sulfate; and, in contact with calcium minerals, it is transformed to various hydrated calcium vanadates (*8*).[2]

The chief ore minerals of vanadium are patronite, VS_4; carnotite, $K_2O.2UO_3.V_2O_5.3H_2O$; roscoelite (vanadium mica); and vanadinite, $Pb(PbCl)(V_2O_4)_3$.

In the United States mining methods are mostly of very simple character, since very few deposits are large enough to warrant much in the way of equipment. Portable two-drill compressors, jackhammer drills, gasoline hoists, half a dozen mine cars, and an air hoist or two make up the average mine equipment (*12*). Virtually none of the mines requires artificial ventilation. The mining method employed by a mine near Rifle, Colo., has been described in detail (*5*).

Vanadium ore is mined in Peru by open-pit methods.

In the United States the usual methods of recovering vanadium from ores include direct reduction and chemical processes, such as roasting with or without reagents (for example, soda ash or salt) and then leaching to remove the vanadium. The process used at the Naturita (Colo.) mill of Vanadium Corporation of America has been described (*12*).

The resulting vanadium pentoxide produced contains 85 to 92 percent V_2O_5 and is consumed largely as a raw material in the manufacture of ferrovanadium, which contains 35 to 55 percent vanadium. Some oxides, however, are intermediates for the manufacture of numerous chemical compounds, such as catalysts, and some oxide is used for the addition of vanadium to steels under certain special conditions.

The process used in treating the vanadium ore of Peru is a combination of roasting and lixiviation. Briefly, the flow comprises 10 principal steps (*13*): (1) Crushing, (2) roasting, (3) quench-water leaching, (4) grinding, (5) filtering of quenched pulp, (6) acid leaching of quenched residue, (7) filtering of leached pulp, (8) combining the 2 pregnant solutions from operations 5 and 7 and precipitating from this combined solution the vanadium values, (9) filtering the precipitate from the barren liquid, and (10) drying, mixing, and sacking precipitates.

The concentrates, which average about 65 percent V_2O_5, are shipped to Cambridge, Ohio,

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

where they are converted to vanadium oxide or ferrovanadium.

Ductile vanadium (99.8 to 99.9 percent pure) was made available in 1950. It can now be obtained in massive form for remelting, as well as in ingots, bars, sheet, and foil. The metal is lighter than iron, has good structural properties, and resists pitting and corrosion by salt spray and sea water. It can be rolled at ordinary temperatures; cold reductions up to 85 percent have been made without annealing. Pure vanadium machines well and is easily welded with regular shielded-arc methods (*20*).

USES

About 90 percent of the vanadium used is consumed as ferrovanadium in the manufacture of tool steels, engineering steels, high-strength structural steels, nonaging rimming steels, and special wear-resistant cast irons. Ferrovanadium is also used in welding-electrode coatings, as a deoxidizer, and in permanent-magnet alloys. Vanadium oxide is also employed for adding vanadium to steels under certain special conditions. Vanadium oxide and ammonium metavanadate are utilized as catalysts, in glass and ceramic glazes, for driers in paints and inks, and for laboratory research.

Vanadium is mainly used in steel for its grain-refining and alloying effects, and a little goes a long way. In high-speed steels the vanadium content ranges from approximately 0.50 to 2.50 percent, although still higher percentages are sometimes employed. Alloy tool steels, other than high-speed steels, contain 0.20 to 1.00 percent vanadium. The quantity of vanadium added to engineering steels is generally 0.10 to 0.25 percent. Most steels containing over 0.50 percent vanadium are for special purposes. Vanadium can be successfully used alone in an alloy of carbon steel; but, in a wide variety of engineering and structural steels, it is more generally combined with chromium, nickel, manganese, boron, and tungsten. A high-temperature steel, which contains neither cobalt nor columbium but, instead, employs titanium and small, controlled quantities of vanadium, has been developed (*22*). In additions from 0.10 to 0.15 percent, vanadium increases the strength of cast iron 10 to 25 percent and adds a considerable degree of toughness. The use of metallic vanadium alone is limited at present largely to alloying with gold in dental alloys, with copper and bronzes (such as for aircraft propeller bushings), and with aluminum for airframe construction (*11*).

SUBSTITUTES

Supplies of vanadium are usually ample to meet requirements of the United States, and it is only necessary to resort to substitutes and alternates when, as in 1941–43, apparent tightness of vanadium was caused primarily by substitution of SAE steels, containing chromium and vanadium, for unobtainable nickel-containing steels. In the great majority of instances where vanadium steels have been substituted, equally serviceable substitutions are said to be had in molybdenum or molybdenum-silicon steels with a grain-refining addition of aluminum (*21*). This saves chromium as well. An extreme example for the saving of vanadium in World War II was its omission from rolled tank armor and the use of manganese-titanium steel for high-strength plate for the Navy in place of a vanadium-containing steel (*16*). Platinum can be used as a substitute for vanadium as catalysts.

PRODUCTION

Because complete information on the quantity of vanadium recovered as byproducts of iron ore and other raw materials is lacking, it is impossible to determine world production of vanadium from all sources. Consequently, table 2 reflects only the production of vanadium in ores and concentrates for the countries listed, plus the quantity recovered in the United States as a byproduct of phosphate rock.

TABLE 2.—*World production of vanadium in ores and concentrates, 1943, 1945, and 1947–53*

[Metric tons]

(Compiled by Berenice B. Mitchell)

Country	1943	1945	1947	1948	1949	1950	1951	1952	1953
Argentina	--------	3	7	(1)	(1)	(1)	(1)	(1)	(1)
Northern Rhodesia	426	219	56	173	153	------	87	43	------
Peru	847	688	435	511	456	436	449	437	317
South-West Africa	577	420	282	187	163	295	529	624	540
United States (shipments)[2]	2, 534	1, 344	961	(3)	(3)	(3)	(3)	(3)	(3)
Total[4]	4, 384	2, 674	1, 741	(3)	(3)	(3)	(3)	(3)	(3)

[1] Figures not available.

[2] Includes also vanadium recovered from phosphate-rock mining.

[3] Bureau of Mines not at liberty to publish figures.

[4] Total represents data only for countries shown in table and excludes vanadium in ores produced in French Morocco, Spain, and U. S. S. R., for which figures are not available; the total also excludes quantities of vanadium recovered as byproducts from other ores and raw materials.

FOREIGN TRADE[3]

Imports of vanadium into the United States during the past 10 years have comprised chiefly concentrates largely from Peru, flue dust from Venezuela and Curacao, and ferrovanadium from Japan, France, Germany, Sweden, and United Kingdom.

Vanadium ore and concentrates enter the United States duty free. The rate of duty on ferrovanadium is 12½ percent ad valorem.

TABLE 3.—*Vanadium imported for consumption in the United States, 1944–53*

[Pounds]

Year	Vanadium ore or concentrates		Vanadium-bearing flue dust		Ferrovanadium: Gross weight
	Gross weight	Vanadium content	Gross weight	Vanadium content	
1944	4, 247, 490	1, 284, 603	191, 901	40, 171	-------
1945	8, 776, 328	1, 550, 479	133, 795	26, 293	-------
1946	2, 784, 349	791, 057	97, 750	20, 931	-------
1947	3, 274, 548	983, 869	143, 124	71, 819	-------
1948	4, 034, 509	1, 051, 675	-------	-------	-------
1949	2, 028, 980	551, 337	-------	-------	-------
1950	5, 110, 403	1, 457, 010	9, 575	804	130, 022
1951	3, 893, 900	982, 878	-------	-------	123, 050
1952	4, 338, 660	1, 043, 797	12, 285	939	21, 396
1953	2, 959, 600	716, 977	9, 822	1, 010	17, 364

Exports of vanadium are usually small and consist chiefly of ferrovanadium, as shown in table 4.

TABLE 4.—*Exports of vanadium from the United States, 1944–53*

[Pounds]

Year	Ore and concentrates[1] (vanadium content)	Ferrovanadium (gross weight)	Vanadium metal, alloys, and scrap (gross weight)
1944	6, 254	1, 191, 996	74
1945	113, 927	171, 314	32
1946	6, 051	113, 058	----------
1947	7, 661	178, 198	25, 436
1948	13, 180	238, 824	2, 602
1949	13, 130	194, 655	2, 754
1950	963	82, 449	4, 106
1951	2, 817	122, 344	1, 712
1952	120, 367	293, 162	103, 036
1953	[2] 12, 319	[3] 156, 952	(4)

[1] Probably also includes fused vanadium oxide.
[2] Comprises vanadium pentoxide, vanadic oxide, vanadium oxide, and vanadates.
[3] Comprises ferrovanadium and other vanadium alloying materials containing over 6 percent vanadium.
[4] Beginning Jan. 1, 1953, not separately classified.

PRICES

For many years vanadium ore was quoted at 27½ cents a pound of contained V_2O_5, but on March 8, 1951, the price was advanced to 31 cents. This quotation, however, disregards penalties based on grade of the ore or the presence of objectionable impurities—matters important to the refiners, inasmuch as impurities vitally affect recovery. During the past 10 years the quotation on ferrovanadium has ranged from $2.70 to $3.25 a pound of contained vanadium (depending upon grade of the alloy) and on vanadium pentoxide (technical grade) from $1 to $1.33 a pound of V_2O_5.

DEFENSE PROGRAM

During the Korean conflict supplies of vanadium were adequate for defense requirements, and no restrictions were placed on its use by the National Production Authority.

During World War II vanadium was placed under complete allocation on December 20, 1941. Allocation control was lifted on December 30, 1943, and the restrictions on the use of vanadium in melting-alloy tool steels were removed in January 1944, thereby allowing producers of alloy steels to use any alloy content desired, except that they were required to employ 50- to 60-percent scrap in all melts.

RESEARCH

The recovery of vanadium from titaniferous magnetites has been investigated by the Bureau of Mines (*1, 14, 15, 23*).

According to Report of Investigations 3679:

> It appears that most of the vanadium in this ore (Essex County, N. Y.) can be recovered in pig iron, which can be blown in a converter to produce a slag containing vanadium in amounts suitable for the production of ferrovanadium.
>
> A recovery in the metal of 87.3 percent of the vanadium was obtained during the operation of the experimental blast furnace on burdens composed of iron ore and titaniferous sinter containing 11.1 percent TiO_2 and 0.26 percent vanadium. Approximately 4 percent of the vanadium was lost in the slag. These results show that the oxides of vanadium are reduced readily under blast-furnace conditions.

According to MacMillan and others (Report of Investigations 4912):

> Tahawus magnetite from the MacIntyre mine at Tahawus, N. Y., was successfully treated as follows: (1) The ore was roasted at 1,050° to 1,080° C. with carbon and soda ash, thereby reducing the iron to metal sponge and retaining the titania in the slag. (2) The metallic iron was separated, after wet grinding, by magnetic or gravity means. (3) The slag fraction was air-roasted at 850° C. and leached with NaOH to re-

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[3] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

cover Na_3VO_4. (4) The leached solids received a dilute-H_2SO_4 leach to remove interfering impurities, chiefly Na_2O and SiO_2. (5) Titania was dissolved from the leached solids of step (4) by a H_2SO_4 bake followed by a water leach. (6) Titania was precipitated by seeding and boiling the resulting solution. The spent liquor was used for the dilute-H_2SO_4 leach in step (4).

With certain modifications, the process described for Tahawus magnetite was successfully applied to a titaniferous iron ore from Iron Mine Hill near Woonsocket, R. I.

According to Report of Investigations 4902, the ore from the deposit at Iron Mountain, Wyo., was roasted with 15-percent sodium carbonate to convert the vanadium to a water-soluble form and fix the soda for the subsequent electric-smelting step. Approximately 90 percent of the vanadium is recoverable in a product assaying 80 percent V_2O_5.

Processes for recovering vanadium from phosphates have been studied by the Bureau (*2*, *3*).

A new process (*17*) for treating high-lime carnotite ores, a method (*19*) for the recovery of vanadium from chromate liquors, and development of a method (*18*) for recovering vanadium directly from alkaline solutions without neutralizing them have been described, and a patent (*6*) was issued for a process that overcomes the objectionable characteristics of other processes for treating vanadium and uranium ores.

OUTLOOK

The increased demand for uranium for atomic energy purposes has carried with it production of byproduct vanadium greatly in excess of industry requirements and offers an attractive field of research to find new uses. Moreover, the nature of the domestic ore is such that vanadium will be produced as a byproduct in greater quantity than uranium. Also, the program of AEC makes economic ores lower in grade than could previously be mined. Furthermore, additional plants primarily for producing uranium are under construction or projected. For these reasons, the potential domestic production of vanadium, plus imports from Peru, is expected to be ample for defense requirements and for appreciable additions to the National Stockpile.

However, if potential supplies should be inadequate for full mobilization needs, other sources, of which there are large reserves, could be brought into production to make up the deficiency. Potential sources of additional vanadium, though the cost may be high, include the titaniferous iron ores of the MacIntyre mine in Essex County, N. Y.; the extensive deposits of phosphate rock, such as those in Idaho and Wyoming; vanadium-bearing slags; and the small lead-vanadium deposits and complex ores of some of the Western States. The possible recovery of uranium through the large-scale processing of phosphate may carry with it a substantial byproduct output of vanadium. More efficient beneficiation methods for recovering vanadium from carnotite ores would add appreciably to its availability.

As a result of extensive research in recent years, some new uses may be developed for vanadium. It has been demonstrated that vanadium has a strong potential as a structural engineering metal, but its utility for this purpose has not yet been fully explored. Production of low-cost, high-purity vanadium for industrial applications might result in a substantial new outlet for the metal.

PROBLEMS

A growing problem of the vanadium industry is the large surplus production and the apparent need to expand uses of the metal. At present there are no major applications involving vanadium alloys where vanadium is the primary constituent.

The high cost of producing vanadium by present methods restricts it use. For example, vanadium can be used in place of chromium, nickel, and molybdenum; but, because it costs substantially more, efforts are not directed toward the maximum use of vanadium. Thus, problems as regards vanadium are its high production cost and resultant high price.

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VERMICULITE

By

Oliver S. North [1]

AS RECENTLY as the late 1920's there were few indications that a \$15-million-a-year industry would ever grow out of the mineralogic curiosity vermiculite. Today, it is an important industrial commodity used in insulating, as a lightweight aggregate and an agricultural material and for a wide range of other applications. The quantity of exfoliated vermiculite produced and used has increased tenfold in the last 15 years.

Summary

From an uncertain commercial beginning in 1925, production of crude vermiculite concentrates in the United States reached 209,000 short tons in 1952. An additional 8,000 short tons was imported from Union of South Africa. This quantity of crude mineral was enough to produce about 200,000 short tons of exfoliated material. In 1953 the consumption of crude vermiculite concentrates from both foreign and domestic mines declined to slightly less than 197,000 short tons.

The Zonolite Co., Chicago, Ill., now produces the bulk of the crude vermiculite used in the United States from mines near Libby, Mont., and Enoree, S. C.

Vermiculite is mined from open pits, crushed, concentrated, dried, and sized. After shipment to the market area it is exfoliated in kilns. Exfoliation is mechanical separation and warping of the individual layers caused by conversion of the combined water component to steam. The material from the kiln is cooled and packed in paper bags. Exfoliated vermiculite has a very low bulk density because of the air spaces between the layers.

Although the major quantity uses of exfoliated vermiculite are in the construction industry—for loose-fill thermal insulation and as concrete and plaster aggregate—many specialized uses have been found for these lightweight, accordionlike particles.

Through the efforts of members of the industry and their trade associations the technology of processing and using vermiculite has been improved, and research along those lines is continuing.

Reserves of crude vermiculite of commercial quantity and quality appear adequate for many years of use, but they are considerable distances from major market areas.

There is need for further investigation of known domestic occurrences of vermiculite, search for additional deposits of vermiculite in areas near major consuming centers, research on mining and milling methods, examination of the efficiency of various types of furnaces, study of lightweight vermiculite concretes, investigation of new uses for finer sizes of exfoliated vermiculite, and dissemination of comprehensive technical, statistical, and economic information on this commodity.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

STATISTICS

Production and consumption of crude vermiculite increased steadily from 1943, when the quantity mined was 46,645 short tons and the value was $471,595, to 1951, when 209,008 short tons of screened and cleaned crude vermiculite valued at $2,679,148, was sold. By far the bulk of this material was produced by a single firm, the Zonolite Co., at its Montana and South Carolina properties. Table 1 shows the quantity and value of screened and cleaned vermiculite produced and sold in the United States for 1935–53.

The Bureau of Mines has not canvassed the production of exfoliated vermiculite on an annual basis. In the annual Minerals Yearbook it has been customary to assume a loss of weight during exfoliation and then assign an average value to the derived figure. Using this method, the estimated quantity and value of exfoliated vermiculite produced and sold in the United States during 1943–53 are shown in table 2. There are approximately 50 vermiculite-exfoliation plants in the United States.

TABLE 1.—*Screened and cleaned vermiculite concentrates sold or used by producers in the United States, 1935–53*

Year	Quantity (short tons)	Value, f. o. b. mill
1935	7, 068	$88, 445
1936	16, 933	185, 787
1937	26, 556	260, 664
1938	20, 700	192, 000
1939	21, 174	174, 587
1940	22, 299	137, 698
1941	23, 438	125, 444
1942	57, 848	319, 931
1943	46, 645	471, 595
1944	54, 116	541, 744
1945	64, 808	648, 077
1946	86, 390	867, 973
1947	131, 385	1, 338, 572
1948	138, 635	1, 387, 233
1949	168, 819	1, 686, 419
1950	208, 096	2, 122, 427
1951	209, 008	2, 679, 148
1952	208, 906	2, 657, 826
1953	189, 535	2, 445, 381

TABLE 2.—*Exfoliated vermiculite sold or used by producers in the United States, 1943–53* [1]

Year	Estimated quantity (short tons)	Value	
		Estimated average, per short ton	Total
1943	43, 333	$75	$3, 250, 000
1944	48, 704	75	3, 652, 800
1945	58, 333	75	4, 375, 000
1946	82, 067	75	6, 155, 000
1947	124, 813	75	9, 361, 000
1948	131, 707	75	9, 878, 000
1949	160, 373	75	12, 028, 000
1950	200, 000	80	16, 000, 000
1951	200, 000	80	16, 000, 000
1952	200, 000	75	15, 000, 000
1953	185, 000	70	13, 000, 000

[1] No estimates of quantity and value of exfoliated vermiculite were made before 1943.

MINERALOGY AND PROPERTIES

The mineralogy of vermiculite is complex. There is a specific mineral named vermiculite but the term also is applied commercially to a number of other closely similar minerals that have the property of exfoliation. Authorities present the mineralogic description of "vermiculite" in a variety of ways. In general, it can be said that the vermiculite minerals are essentially hydrated silicates of aluminum and magnesium. They vary greatly in composition and physical properties. Exfoliation is said to be due to mechanical separation of layers when the contained water is converted to steam. The expansion is axial, normal to the basal cleavage, and change in the other two dimensions is negligible.

For the sake of simplicity the term "vermiculite" as used in this report includes all types.

Vermiculite crystallizes in the monoclinic system, with perfect basal cleavage and a hardness ranging from 1.5 to 3.0 on the Mohs scale. The specific gravity of the crude mineral, as mined, is 2.5. The apparent specific gravity of material crushed to furnace specifications is 0.8 to 1.4; and of exfoliated, 0.1 to 0.2. In nature vermiculite occurs in a wide range of colors from black through various shades of brown to yellow. On exfoliation the color changes to a silvery or golden hue, according to the degree of heat and conditions of exposure to air.

The indefinite and variable chemical composition of vermiculite is indicated by the following ranges in percentages of each of the major components present: Silica, 31–41; alumina, 10–17; iron oxides, 5–22; magnesium oxide, 11–31; and combined water, 4–21. Chemical composition alone has little value in determining whether a particular vermiculite has commercial potentialities; technical trial of the material provides the only satisfactory test.

Vermiculite often occurs at or near the contact of an acid intrusive rock, especially a pegmatite dike, with a basic or ultrabasic country rock, particularly pyroxenite, peridotite, dunite, or serpentine. At Libby, Mont.—location of the largest vermiculite-mining operation in the world—vermiculite is associated with a large pyroxenite-syenite stock and small porphyry dikes that intrude argillites and quartzites. The Loolekop deposit in the Palabora district, Northern Transvaal, South Africa, is reported to be a very large occurrence of altered phlogopite mica.

TECHNOLOGY

Most vermiculite deposits are mined by typical open-pit methods. At Libby the material is quarried on 18-foot benches and trucked to the primary plant. Details of crushing and screening crude ore depend on the quality of the mineral being mined at the time. The material is air-dried in a rotary kiln to eliminate much of the surface water; and the heat is carefully controlled to make certain that no partial exfoliation takes place. A complex system of crushers, screens, and air-separator units accomplishes beneficiation and very close sizing of the different grades.

Exfoliation occurs between 1,400° and 2,000° F. Several types of exfoliating kilns are in use, and although they vary in structural details are similar in principle. The exfoliator in general use in the United States is a modified vertical shaft furnace heated by oil- or gas-fired burners. The charge fed into the top of the furnace is impeded in falling by staggered baffles and, on coming in contact with heat from the burners, exfoliates. The expanded product, which subsequently may or may not be sized, passes to an air separator that removes unexfoliated material and rocks, thence to a cooler which reduces the temperature, so that the material can be handled. Finally, the finished product is packed in 2- to 4-cu. ft. paper bags for shipment. The rate of output is about 320 cu. ft. of exfoliated material per hour per furnace, and most plants are equipped with 2 or more furnaces.

The plant practices of processors using South African crude vermiculite differ somewhat from those described above; an inclined kiln generally is utilized, and the exfoliated material usually is screened to produce a uniform product.

USES

Gradation of the exfoliated vermiculite ranges from ½-inch down to minus-100-mesh. The coarsest product is used for chicken litter; that in the medium-size range is marketed as loose fill insulation, lightweight aggregate for plaster and concrete, soil conditioners, etc.; and finer granules are used as carriers for agricultural chemicals and as an anticaking agent in fertilizer. Numerous other uses have been found for the different sizes from time to time.

Official statistics on the uses of exfoliated vermiculite are not available, but one of the earliest—and one still popular—is its application as a loose-fill medium between suitable sheets, boards, lath, or wire netting for thermal insulation of roofs, ceilings, and walls. The size range desired for this use is minus-3-, plus-14-mesh. One ton of vermiculite will cover an area of 1,120 square feet to a depth of 3 inches and provide thermal insulation equivalent to a 5-foot brick wall or 7 feet of dense concrete.

In addition to its use for insulating domestic, commercial, and industrial structures, loose vermiculite is employed for packing refrigerators, incubators, fireless cookers, thermal jugs, ovens, safes, and water heaters and in many of those uses competes with cork, mineral wool, diatomite, etc.

The use of exfoliated vermiculite, to which a stabilizing agent has been added to prevent segregation, in lightweight concrete is now one of the more important applications of the mineral. Vermiculite concrete applications include monolithic walls, floors, and roofs and prefabricated units, such as panels, planks, beams, precast chimneys, etc. The use of such a relatively high-priced aggregate is justified in part by the reduction in weight of walls, floors, and ceilings, which makes possible appreciable savings in the weight of steel required and therefore in the overall cost. Concrete of this type, in addition to being more fire-resistant, has advantageous heat- and sound-insulating properties.

Vermiculite is used as a plaster aggregate because of its acoustical and thermal insulating properties and the fire-resisting qualities of such plasters. A novel use for vermiculite plaster is in making imitation marble.

An insulating cement of vermiculite and a suitable binder can be applied as a plastic mass and dried in place for insulating surfaces operating at temperatures up to 1,800° F.

Exfoliated vermiculite may be bonded with any of numerous binders, for example, dextrin, fuller's earth, casein glue, wood pulp, fire clay, plaster of paris, portland cement, synthetic resin, waterglass, etc., to produce thermal plasters, fireproof boards, plastic refractory insulation, molded articles, etc. In combination with asphalt, exfoliated vermiculite also is utilized in producing various roofing and flooring compositions and in manufacturing tar adhesives.

Refractory insulation of several types is made with vermiculite. On the roofs of furnaces and kilns vermiculite can be used in the form of loose fill, which in turn is covered with vermiculite cement. Vermiculite brick bonded with such materials as portland cement, fire clay, sodium silicate, etc., are used for high-temperature industrial insulation and are particularly suitable for the insulation arches of open-hearth furnaces where structural strength of the brick is not essential.

Exfoliated vermiculite is used as a soil additive for rooting cuttings and tubers, in hotbeds for sprouting seedlings, and as a general soil conditioner, mainly to lighten clay soil and to improve the moisture-retention qualities of sandy soil.

Many processors find it difficult to dispose of the entire output of "fines" from their kilns, although that fraction has been used to some extent for a number of purposes. Chemical companies purchase it for use as a filler, insecticide carrier, and the like.

Exfoliated vermiculite meets competition in all fields of use. Competitive concrete aggregates include sand, gravel, stone, air-cooled slag, etc., and such lightweight materials as expanded clay, foamed slag, expanded shale, expanded perlite, cinders, pumice, scoria, diatomaceous earth, etc. Competing plaster aggregates include sand and expanded perlite. Other loose-fill insulation materials include mineral wool, diatomite powder, and expanded perlite. A wide variety of materials is used for industrial insulation, including mineral wool, asbestos sheet, 85-percent magnesia, silica aerogel, diatomite products, "foamglas," etc. As a soil conditioner (not a fertilizer) vermiculite competes with expanded perlite, peat, straw mulch, leaf mold, plant stalks, and other agricultural waste products.

In all minor applications vermiculite competes in markets formerly held by other products, and its use in each instance depends on considerations of economy or utility or both.

IMPORTS

Considerable quantities of crude vermiculite are imported into the United States. No import duty is assessed on unprocessed vermiculite. Very little, if any, exfoliated vermiculite (on which the duty would be 15 percent ad valorem) is imported.

Nearly all imports of crude vermiculite come from deposits in Union of South Africa, particularly from the Palabora area, Northern Transvaal, and are used by exfoliating plants in the New York City area and in Northeastern and Southern States. In 1950 the United States imported from Union of South Africa 16,531 short tons valued at $256,152, f. o. b. port of shipment; however, in the following 3 years the quantity imported from that source declined steadily—reportedly due to an increase in ocean freight rates—to 6,930 short tons, valued at $120,000, f. o. b. port of shipment in 1953. With the addition of freight charges the cost of African vermiculite to eastern producers is about the same as that of vermiculite from Libby, Mont.

EXPORTS

Several thousand tons of crude vermiculite is exported annually to Canada by the Zonolite Co.; smaller quantities go to Italy, Mexico, Venezuela, Puerto Rico, Hawaii, Arabia, and Switzerland. Very little exfoliated vermiculite is exported because of high freight charges on bulky expanded material.

RESERVES

Montana's vermiculite reserves are estimated to be large enough for many years of anticipated demand, especially since recent mill improvements have made usable large quantities of marginal ore. Apparently Zonolite Co. has released no information on the extent and quality of its reserves at Enoree, S. C.; but increasing tonnages are mined there, and a new mill is now under construction near that town. Medium-size deposits, many of which have reported irregular commercial operation in the past, are known in Wyoming, Colorado, North Carolina, Georgia, Texas, and Pennsylvania. The mineral also has been identified in many other States.

TRANSPORTATION

Transportation costs are a large part of the delivered cost of crude vermiculite. Freight charges from Libby, Mont., to eastern producers amount to more than the cost f. o. b. mill. Even higher freight charges for the bulkier exfoliated material tend to limit its area of distribution to a radius of 100 to 300 miles around the exfoliating plant. Data on the cost of exfoliating vermiculite are not available.

PRICES

The average value of crude vermiculite, meeting customers' specifications, is $12 to $13 per short ton, f. o. b. mill. Exfoliated vermiculite sells for a wide range of prices, depending on specifications and the use to which it is put. Concrete and plaster aggregates usually are priced at $0.90 to $1.20 per 4-cu. ft. bag, f. o. b. plant.

RESEARCH

The National Bureau of Standards has tested the characteristics of vermiculite concretes and also made fire tests on vermiculite plaster partitions and columns. Most laboratory research on production and use technology has been conducted by the industry, principally the Zonolite Co. and the earlier companies that merged to form that firm. Many research projects are sponsored by the two trade associations of the industry, the Vermiculite Institute and the Vermiculite Association, Inc. Results of those projects are made available to the industry, building contractors, and other interested individuals.

OUTLOOK

The use of exfoliated vermiculite as a loose-fill insulating material, an aggregate in concretes and plasters, refractory insulation, and for many other purposes has been growing steadily. Demand for nearly all of the vermiculite products has increased as more of the material has become available and the use technology better known. Continuing search for new uses and constant endeavor to improve older ones are being carried on by the industry and its trade associations.

Formerly an objection to using vermiculite concrete aggregate was segregation of the ingredients due to differences in specific gravity. Vermiculite exfoliators have largely overcome this problem by incorporating a stabilizing agent in aggregate to be used in concrete, and the quantity used for this purpose is expected to increase in the future.

The immediate and long-range outlook for the vermiculite industry is favorable for two reasons. High-level housing and industrial construction activity is expected to continue for some time, providing a steady or growing market for vermiculite building products. Also, since vermiculite still is a relatively new commodity, it has not reached all of its potential market in the United States, either geographically or in variety of uses. Although the industry faces the usual technical and competitive problems, it is widely expected that consumption of this commodity will increase or at least maintain its present level during the next few years.

PROBLEMS

Problems confronting the vermiculite industry include the development of new uses and markets, improvement of processing and use technology, and the usual economic problems of cost and distribution.

The overall reserve picture in the United States has been improved greatly in recent years by development of new processes that have made usable extremely large tonnages of low-grade ore; however, there is need for additional domestic sources of crude vermiculite in areas near major markets. High transportation costs from present major commercial sources of crude vermiculite to the major market areas are a serious commodity problem.

Most of the major building codes that formerly made no provision for using vermiculite have been revised to permit its use. However, there still are some building codes that restrict the full utilization of vermiculite.

Studies of the physical, chemical, and processing characteristics of vermiculites from different sources for determining their commercial potentialities are needed.

Lightweight and semilightweight concretes, including vermiculite concretes, are being used in increasing quantities, and much study remains to be given to production technology, uses, and merits of the several aggregates, including vermiculite.

An industry problem is the development of a large market for the finer size material produced during the exfoliation process.

The efficiency and advantages of the various types of kilns used for exfoliating vermiculite should be investigated.

Compilation and distribution of more detailed and comprehensive technical, statistical, and economic information are needed to provide industry and Government agencies with basic facts required for the most effective management and guidance of policy decisions.

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WATER

By

Sidney Gottley [1]

WATER—fresh water—is indispensable for human life, because water is 70 percent of man's anatomy. The lavish use of water in most areas of the United States would be wasteful to those living in water-short areas of the world. Misuse of water resources can cause irreparable harm to an economy; indeed the production of almost all commodities necessary to an industrialized economy requires enormous quantities of water for mining, refining gasoline, air-conditioning buildings, generating electricity, agriculture, and steelmaking. Without adequate supplies of this vital commodity, modern civilization could decline as have many historic civilizations whose water supplies failed.

Summary

This chapter presents the salient facts and problems in water resources as they relate to the mineral industries. Water shortages are widespread over the United States and have hampered mining operations in some places; elsewhere, too abundant water has interfered with mining. Many water resources have become polluted, sometimes by the mineral industries.

Water-supply problems can be attacked by conservation of water through reclamation of used water, artificial precipitation of rainfall, and demineralization of saline waters. Present demineralization methods are too costly for wide use, but research on several new methods promises future cost reductions. Much of the research is conducted under contract with the United States Department of the Interior.

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[1] Executive Assistant to the Assistant Director—Programs, Bureau of Mines.

BACKGROUND

SCOPE

In this chapter major emphasis is placed on the newer methods of providing water supplies—methods that might be termed "processing" as contrasted to the traditional methods of solving water problems by "handling" water.

WATER SHORTAGES

Water resources are exhaustible. At many points throughout the United States ground-water supplies have been used faster than they can be replenished by natural means, frequently by overconcentration of wells in small areas.

In the fall of 1954 the southern two-thirds of the United States found its ground-water levels at or near record lows. In 17 of 22 key observation wells of the Federal Geological Survey over the entire country, year-end water levels were below average.

Surface waters were equally deficient. The annual runoff of the major river systems was the second lowest on record. The annual flow of the major continental rivers was alarmingly low. The Mississippi was 54 percent of normal; the Missouri, 58 percent; the Ohio, 51 percent; and the Colorado, 46 percent.

Only in the New England and Northwest areas did ground-water levels remain relatively high, and the St. Lawrence and Columbia Rivers flowed at a rate above normal.

Existing and impending droughts are not a new development. The last two decades have seen many examples of the disturbing effects of man's activities on the natural pattern of water availability.

At Texas City, Tex., wells have been drawn upon until the water level has dropped more than 100 feet below sea level. Intrusion of salt water from the nearby ocean has followed, ruining the wells for further use. In Atlantic City, N. J., the water level in wells has dropped to 85 feet below sea level. At Long Beach, Calif., the water table has been pulled down to 75 feet below sea level, and well water is becoming salty all along the coast as the salt water crowds in to replace the fresh water withdrawn. In the Los Angeles, Calif., district, removal of water bulk from the soil has caused its land surface to sink as much as 8 feet in some spots. At Savannah, Ga., the head of water dropped to 32 feet below sea level.

Bristol, Pa., wells, sunk when the Delaware River became overpolluted, formerly delivered 700 gallons a minute, but in 1950 only 200 gallons a minute could be pumped. The city has been forced to turn back to the Delaware for water supplies at great expense.

Pumping of water for irrigation has lowered the water level in 1 county in the Texas Panhandle 30 feet; in another, 19 feet. In 1941, 2,560 wells were operating there; by the fall of 1948, 8,500 wells were drawing water to irrigate about 1 million acres. Similar water troubles exist in the Salt River Valley and adjacent areas of Arizona.

The list of water-short areas is tremendous. Baltimore, Md., and Mobile, Ala., are worried over salt intrusion in their wells. Long Island, Memphis, Miami, Chicago, Peoria, the Lake Charles area of Louisiana, the Grand Prairie region of eastern Arkansas, and many other places are concerned about water shortages.

Many water shortages are affecting present mineral-industry operations. Lack of water for processing has discouraged some companies from undertaking phosphate-rock production in Idaho and Wyoming. The Rand district of Kern County, Calif., was too dry to permit mining its tungsten resources, but successful wells found clear, good water in 1954. An increase in gold production and full-scale tungsten mining were expected to follow.

EXCESS WATER

Many mining ventures have been hampered by excessive ground water. Often, bailing or pumping has had to be employed at very shallow depths. As greater depths are reached and more territory opened, water flows usually increase. When these flows exceed economical pumping capacity, the extent of operations has been severely limited. In many instances, such as the Fad shaft, Eureka, Nev., shaft-sinking and mining methods have been modified at great expense to meet special water problems posed during otherwise routine operations.

METAL-MINE FLOODING

A situation of this type has existed in the Leadville mining distict of Colorado for many years. Two drainage tunnels, driven in 1895 and 1922, were successful as drainage projects. The first was also valuable for exploration purposes, although ore discoveries in the second were disappointing. During World War II the Federal Bureau of Mines began to drive a tunnel over 2 miles long to drain all or most of the Fryer Hill, Carbonate Hill, and Iron Hill basins. Unforeseen ground conditions slowed the work and increased costs, so that work was stopped when the war ended. When the shortage of lead, zinc, and manganese again became serious in 1950, work was resumed in September of that year. Completed in February 1952, the tunnel has not only unwatered many mines of the area but has also made it much cheaper for mines below the tunnel level to pump out water, since pumping need only be up to the tunnel level instead of all the way to the surface. The tunnel has also proved valuable for exploration; and several promising ore bodies have been encountered or dewatered thereby. By agreement with property owners, the Government will collect royalties on minerals made accessible, produced, and sold as a direct result of the drainage project.

The basic criteria for projects of this type are:

(1) Costs of dewatering by pumping versus the cost of a drainage tunnel.

(2) The amount of ore reserves to be made available by dewatering.

(3) Provision of water supplies.

Excessive water has been the greatest difficulty in developing the Park City mines in Utah. Giant Cornish pumps were used in the early days; but, when the mines reached depths at which the pumps could not handle the large flows of water, it became necessary to drive drainage tunnels. Because water is a community problem, the many small mines in the area had to consolidate into a few large companies able to finance the drainage works and utilize the full benefits therefrom.

At the Yerington copper mine in Nevada the water table is lowered 100 feet to keep the pit dry. Five wells, arranged in a ring around the pit, are pumped, supplying water needed for the operation as well as lowering the water table.

At several places in the Tri-State lead-zinc district known reserves are under water. The Crestline, Kans., field, abandoned in the depression of the thirties, has been inundated. If the Oklahoma-Kansas fields were abandoned and water should fill their miles of underground workings the cost of dewatering probably would be prohibitive. In these fields many million gallons of water must be pumped each day to maintain the drainage status. Several mining companies operate numerous pumping plants that are supported mainly by the larger scale, low-cost, mechanized, mining operations on the predominantly large areas of marginal-ore reserves.

COAL-MINE FLOODING

In the Pennsylvania anthracite region, encroaching water is progressively flooding the mines. This flooding menaces the safety of miners and threatens to curtail present production levels. Since 1949 the Federal Bureau of Mines has been making an engineering study of the problem, seeking information and preparing plans for steps to be taken to combat the encroaching water. As a possible remedial measure one system would employ tunnels to tap the underground pools and conduct the water to central pumping plants where it would be lifted to the surface and turned into existing surface watercourses. Alternatively, the tunnel system could dump into a main tunnel that would conduct the water 300 miles south to the headwaters of Chesapeake Bay. Objections have been raised to the latter scheme, because of possible pollution of the bay by acid mine waters.

NONMETALLIC-MINE FLOODING

Large quantities of water enter the fluorspar shaft mines in the Rosiclare vein in Illinois through channels in the limestone and from the Ohio River where it intersects the vein. The problem is somewhat intermittent, increasing in severity with excessive rains and high river levels. This problem has not been adequately studied; and the only current solution is to maintain large pumping capacity, enlarging it as needed.

OILFIELD BRINES

The disposal of oilfield brines has been a problem for many years because of the harmful effects of mineralized water discharged from oil wells on fresh-water supplies, vegetation, and aquatic and other animal life. The brine-disposal problem is particularly acute in agricultural areas, such as Kansas.

Brines associated with petroleum deposits often contain more dissolved salts than does sea water, and the concentration and kinds of dissolved salts vary considerably. Hydrogen sulfide and carbon dioxide gases also are present in solution, singly or together, and their presence increases the ability of brines to retain certain components in solution. Industry and the Federal Bureau of Mines have studied methods for disposing of brines.

The methods used most generally for the disposal of brines produced with oil are:

(1) Impounding in evaporation ponds.
(2) Controlled diversion into surface streams.
(3) Injection into subsurface formations.

Various States recognize one or more of the methods as legally authorized procedure. As a general rule, impounding brine in ponds and solar evaporation have not been entirely successful, principally because of the large volumes handled, the porous nature of the soil, and the low rate of evaporation. Likewise, diversion of brine into fresh-water streams by controlled dilution has not proved satisfactory in some areas, especially in parts of Kansas and Oklahoma where the rainfall is relatively low, because it has been impossible to obtain enough fresh water for such dilution.

Subsurface injection of oilfield brines has been practiced on a limited scale for a number of years; but many problems remain unsolved, and current studies are concerned with methods of conditioning brines to minimize their corrosive properties to avoid damage to oil-well casings, etc. Subsurface injection can be extremely valuable in secondary-recovery methods for petroleum.

INUNDATION OF MINERAL DEPOSITS BY PUBLIC WORKS

The construction of large dams to provide water supplies, power, and flood control often results in the inundation of mineral deposits. The loss of mineral reserves and production must be evaluated in considering the advisability of initiating such projects. Such losses may be variously calculated, and no set formula has been adopted by Federal agencies, although the former Federal Interagency River Basin Committee has clearly stated that—

> a comprehensive public viewpoint should * * * include consideration of all effects, beneficial or adverse, short range or long range, that can be expected to be felt by all persons and groups in the entire zone of influence of the project.

Each case must be evaluated on its own merits.

SOLVING WATER-SUPPLY PROBLEMS

The traditional approach in this country to solving water-supply problems has been to devise new methods or additional facilities for "handling" water. When new supplies have been needed, more or deeper wells have been drilled; canals or tunnels to carry water from longer distances have been dug; and dams to impound water in new reservoirs have been built. These tremendous construction projects are not cheap. This approach is in use currently by municipal, State, and Federal agencies.

The Bureau of Reclamation of the United States Department of the Interior and the Corps of Engineers of the Department of the Army are continually striving to provide additional supplies of water for irrigation, municipal and industrial water supplies, flood control, and hydroelectric power by means of huge dams, canals, and tunnels, etc. Within the limitations of available, usable water supplies and the costs of tapping them, their work may be considered adequate. Unfortunately, many areas do not have such supplies that are accessible, except at exorbitant costs.

Consequently, there has been much interest in recent activities directed toward new approaches, particularly the conservation of water in use, the artificial precipitation of rainfall, and the demineralization of saline waters.

Because the Nation's water requirements are steadily increasing, prosecution of these approaches is literally imperative. In 1950 the daily average withdrawal of water per person was 1,000 gallons or a 170-billion-gallon daily average for the whole country. The total withdrawal of water in the United States since 1900 has doubled, on the average, about every 25 years. Reliable estimates predict that the Nation's water requirements will double from 1950 to 1975.

CONSERVATION OF WATER

Since water is a renewable resource, water conservation may best be defined as the maximum effective utilization of the commodity. In practice, progress toward this goal can be made in three ways:

(1) Reclamation of used water.
(2) Prevention of pollution of water by wastes.
(3) Use of substitutes for fresh water.

Many problems and their solution in these fields do not come within the scope of the mineral industries and therefore will not be discussed here.

RECLAMATION OF USED WATER

Reclaiming water for reuse provides a means of directly increasing the supply of usable water. At the same time, if the cost of developing new supplies of water is high enough, reclamation of water may mean an actual saving in obtaining additional water supplies.

Most industries circulate water through their plants only once. Where water is scarce some plants have been forced to recirculate it. One oil company has reduced its water requirements to one-fifteenth of its needs by extensive recirculation. A new refinery under construction at Amarillo, Tex., will reclaim its effluent. Many mineral-dressing plants are reusing some fraction of their process water.

Very few plants can use surface waters without some type of clarification. If plant effluents are being treated to meet pollution-control regulations, reusing treated waste water can cut original clarification costs.

Other economies can be effected by reusing process water for cooling and other purposes and by reusing water from heating and cooling circuits at lower and higher heat levels, respectively.

The largest source of water for reuse is that discharged by municipal sewage-treatment works. At the moment, psychological barriers are an insurmountable obstacle to reuse of this water for domestic purposes, despite the fact that the water can be made biologically pure and safely usable; industry, however, need not be so squeamish. The Bethlehem Steel plant in Baltimore, Md., is making satisfactory and economical use of sewage effluent.

In many cities the cost of dual-supply systems for using sewage effluent are prohibitive, but industry should not overlook the possibilities in any area.

Recharging techniques for putting water back into the ground have been developed in quite a number of areas, particularly in California, where seasonal rainfall is caught and distributed over "spreading areas," from which the water seeps into the ground to supply needed water-bearing strata.

In numerous other places, such as western Long Island, water is returned to the ground through wells drilled specifically for that purpose. To comply with New York State laws that require an industrial water user in Kings and Queens Counties to maintain ground supplies if he uses over 100,000 gallons a day, the typical industry installation comprises 2 wells, 1 to supply water to the plant and the other to pump water back into the ground. This recycling enables Metropolitan New York to be one of the largest users of ground water in a concentrated area without disastrous depletion. Many other areas may find recharging of water strata necessary in the near future. Such recharging can often be done seasonally.

PREVENTION OF POLLUTION OF WATER BY WASTES

Much has been written about water pollution. Much has been done about it, and much is being done.

A review of representative river valleys by the United States Public Health Service proved conclusively that—

(1) Even though not all streams are polluted, the major streams of the United States are gravely affected and the problem is nationwide.

(2) Pollution is a factor that already affects or eventually will affect all water resources development.

The survey revealed a predominance of pollution in the heavily populated and highly industrialized Eastern States. Four of the river basins studied—the Connecticut, Delaware, Potomac, and Ohio—have 4,722 cities and factories, or over 67 percent of the total in the 11 basins studied. Nearly *two-thirds* of these eastern cities and factories discharged their wastes wholly untreated.

The industrial waste problem is greater and more pressing than the problem of domestic sewage. Industrial wastes are very complex; and knowledge of how to treat certain wastes or how to eliminate acid mine drainage, except at great expense, is not available. Many industrial wastes are more harmful than sewage. The mineral industries contribute their share of wastes—perhaps more than their share—on a national basis.

The Federal Bureau of Mines did much early work on determining the presence of coal-mine drainage-acid in streams. This was followed by a cooperative mine-sealing program that demonstrated the effectiveness of that method, though expensive, in preventing pollution; but the seals were too few, they remained untended until they fell into disrepair, and newly abandoned mines were not sealed. And so today, mine-acid drainage represents the most serious polluting substance in many streams. A relatively recent mine-sealing program in Pennsylvania has already achieved a measurable diminution in the acid content of some of its streams.

Unfortunately, costs are a major obstacle to the construction of waste-treatment plants by factories. Some firms may lack the reserve capital for such construction. An encouraging note is the increasing civicmindedness of many industries—sometimes stimulated by stringent antipollution ordinances—that has led them to greater efforts to control the discharge of harmful wastes.

In striving for maximum efficiency in water use, the fact that waste prevention can mean recovery of valuable mineral, chemical, and metallurgical products that may have enough value to offset the cost of treatment must not be ignored.

Under the Water Pollution Control Act of 1948, the Public Health Service is the principal Federal agency responsible for combatting the many problems in this field, making financial grants to State and interstate agencies.

State boards, joint commissions of several States, and many others are seeking means of abating water pollution. Elimination of pollution can provide supplies of water in many places where it is needed. The cycle that brought cities to the banks of streams and the resulting pollution as the cities grew and became industrialized, in turn rendering the streams unusable, now needs to be reversed. Other provisions must be made for disposing of wastes, so that the streams may be returned to uses of higher quality than has been possible in recent years.

USE OF SUBSTITUTES FOR FRESH WATER

Since as much as 75 percent of the water used by industry (30 percent of the total water withdrawn in the United States) is used for cooling in industrial plants, substitution of saline water for fresh would relieve some of the strain on existing water supplies. There is a need for better and cheaper corrosion-resistant equipment to make this practice more widespread, although it is believed that at least 15 billion gallons per day of salt or brackish water are already being used.

Air cooling is another alternative to fresh-water cooling and has been proved feasible at petroleum refineries and carbon-black plants.

If the efficiency of closed-circuit refrigeration could be increased, this too could substitute for fresh water.

ARTIFICIAL PRECIPITATION

Experiments on artificial precipitation have sought to demonstrate that rain can be made to fall by seeding clouds with such materials as dry ice and silver iodide. Research on methods of precipitation is being carried on, but the principal unsolved problems seem to be in the area of effects produced by applying any or all methods. Many questions are the subject of controversy or even legal action today. Does artificial precipitation actually increase the average annual precipitation in a given area? Does artificial precipitation in one area reduce natural precipitation to harmful levels in neighboring areas? Does artificial precipitation have harmful effects on the climate of other areas at great distances? What legal recourse does an individual, whose livelihood is adversely affected by artificial precipitation, have against the "rainmakers"?

Despite the work of well-qualified organizations, even controlled experiments have not yielded conclusive results. The final answers to these questions may not be available for many years.

DEMINERALIZATION OF SALINE WATERS

HISTORY

The extraction of potable water from sea water dates back to the ancient Greeks. In 350 B. C. Aristotle noted that pure water could be obtained by evaporating sea water.

In 1595 one Conrad Gesner wrote a book on the distillation of water; and Sir Richard Hawkins, in a sea voyage in 1593, is said to have distilled sea water to obtain fresh water. Notable as the first-known patent on water purification of any type was one granted in 1675 to William Walcott.

Additional patents for water purification were granted in England in 1683, 1770, and 1777. In the 1790's, Thomas Jefferson, then Secretary of State, "examined into the truth and importance of the allegations" of a petition by Jacob Isaacs in which he set forth that he had discovered a method of converting salt water into fresh. Jefferson reported ". . . on the whole, it was evident that Mr. Isaac's mixture produced no advantage either in the process or result of the distillation. . . ."

During the 19th century, particularly during the latter half, distilling apparatus was developed to a high degree of perfection in the character of the distillate, compactness of apparatus, and unit cost of installation and operation. Its use continued to be limited primarily to ships and naval bases. Coincident with opening of the Suez Canal, the British Government installed sea-water distillation plants on or near it. Of these, one at Aden seems to have afforded the main supply of potable water at least until 1927, when a deep well was completed. In the United States distillation has never been applied to municipal supplies of fresh water, except at lumbering camps in Texas where water was distilled as early as 1932.

PRESENT UTILIZATION OF SEA WATER

Efforts to develop economically feasible processes to convert sea water have been continuous. Progress in this direction has been surprisingly great in the past 15 years. Salt water may now be conditioned for the following special circumstances:

(1) Aboard sea-going vessels.

(2) Onshore installations or at military outposts on islands where fresh water is at a premium.

(3) Onshore installations for producing steam and power.

(4) Boiler-feed-water installations for central stations.

The cost of producing fresh water has been progressively reduced for all of these purposes. However, industry and agriculture cannot be supplied with fresh water from the sea unless lower cost methods can be found.

Where the need for fresh water is great enough so that cost can be disregarded there is no problem. Enough distillation units were produced in this country during World War II to supply pure water for the daily needs of 1 million fighting men and constituted the sole source aboard many ships, on Iwo Jima, and on other Pacific islands. A cost of $5 per pint for drinking water produced from sea water by the ion-exchange process aboard life rafts was unimportant when it meant saving men's lives.

Multiple-effect distillation is being used on a fairly large scale in some places where fresh water cannot be otherwise obtained and where low-cost or waste heat is available from oil-development processes. The island of Curaçao, Netherlands West Indies, with a population of 45,000, uses 6-effect distillation units. Ten triple-effect units, each producing 100,000 gallons a day of distilled water from sea water, have been constructed to supply water to Kuwait on the Persian Gulf, and the plant is being doubled in size. This installation uses natural gas for fuel at a negligible cost.

In 1954 the Pacific Gas & Electric Co. announced the first United States industrial application of ocean-water distillation at a new power plant under construction at Morro Bay, Calif. The installation was to consist of 2 triple-effect evaporator sets with a combined capacity of 100 gallons a minute. A high salinity of 150 percent of normal sea water and a low boiling temperature will be maintained to minimize salt deposition. Water entering the first effect will be chemically treated.

METHODS OF DEMINERALIZING SALINE WATER

Although many methods of removing the salts from sea water and brackish water have been proposed and tested, no one has yet developed a technically and economically feasible process for general use.

In all probability the solution will not be found by development of one process. Mineral waters vary greatly, as do the uses for demineralized water. A process that may prove successful in treating a specific water or for a particular use may prove unsatisfactory for other waters and other uses.

Saline water is a comparatively simple system of inorganic salts dissolved in water, having physical and chemical properties on which separation processes may be based. The system is quite stable, because the water itself and the inorganic salts dissolved in it are among the more stable compounds. Hence, removal of salts from saline solutions requires relatively large quantities of energy.

The minimum quantity of energy required for separating salt from sea water may be determined by thermodynamic principles to be 2.8 kw.-hr. (3.8 hp.-hr.) per 1,000 gallons, independent of the type of separation process.

Energy requirements for any applied process are always more than the thermodynamic minimum because other energy requirements, such as those required for evaporating the water itself, heat and friction losses, electrical resistances, etc., must be added to the minimum requirements. The total energy requirement for a process depends upon the extent to which these additional demands can be reduced.

The answer to the problem of demineralizing saline waters appears to lie in development of a new process or processes, possibly based upon hitherto untried principles or undeveloped sources of energy. Some potential separation processes, phenomena, and sources of energy are discussed below.

A number of methods are in use for converting saline waters to a relatively pure state. These fall into two categories, those that evaporate water and condense the vapor formed and those that remove salts by ion exchange.

Evaporation.—*Multiple-effect evaporation* uses boiler steam to evaporate sea water in the first-effect evaporator. Condensation of this evaporated fraction in the second-effect evaporator provides the heat necessary to evaporate more sea water. Repetition of this cycle as many as 6 or 7 times decreases the heat wasted and results in lower fuel costs—but higher equipment costs.

Another current evaporation technique is *vapor-compression distillation.* Compression of the vapor from the evaporator increases its temperature, and the compressed steam helps to heat more of the brine. Condensation takes place directly in the steam chest, eliminating the need for cooling water and a separate condenser. The energy requirements of the compressor far exceed those of the evaporator for steam generation. Units of this type have proved most useful in supplying water to coastal oil operations. Portable varieties have found military application in supplying fresh water to troops.

In a third method, *flash evaporation,* the release of water at a particular pressure and temperature into a chamber of slightly lower pressure causes the liquid to flash into vapor. Subatmospheric pressures and multistage operation have been used. Separation of salts upon flashing decreases scaling problems, which has proved useful in shipboard sea-water distilling plants, having lower weight and space requirements than other processes.

Ion Exchange.—The second category—ion exchange—uses resins to supply less objectionable ions in exchange for those not wanted in the water. Operating at relatively high cost on a large scale, because of the tremendous quantities of acid needed to regenerate the resins, the process is used mostly for water-softening—the removal of calcium and magnesium ions. Development of special-purpose resins extends the possible utility of ion exchange in demineralization.

Because all conventional methods were too expensive to be useful for general purposes, water demands were mounting, and the number of water-short areas was increasing, the Congress in July 1952 authorized a program of research on saline water conversion to be conducted by the United States Department of the Interior.

After 2½ years of research conducted under contract, by various research organizations, universities, and private companies, 4 methods appear to hold promise of appreciably lowering costs of demineralization:

(1) An ion-transfer membrane demineralizer that combines a plastic perm-selective membrane with electrolysis to remove dissolved salts. It appears to be best suited to treatment of brackish water, because its power requirements increase markedly with removal of increased amounts of salts.

(2) A variation of vapor-compression distillation in which heat-transfer coefficients are increased 5 to 10 times by causing turbulence to overcome surface resistance to rapid boiling. Tested thus far only on a laboratory scale, this method, if successful, could lower costs of distilling water to a fraction of those of conventional equipment.

(3) Extraction in the supercritical state—at temperatures near 700° F. and pressures of 3,200 p. s. i.—is theoretically possible at low cost, but corrosion problems at such temperatures and pressures will be exceedingly difficult. This method utilizes the fact that saline water displays both liquid and vapor phases above the critical point of pure water. Pure water does not. As temperatures and pressures increase, the salt concentrates in the liquid phase.

(4) The thermal-difference method avails itself of the temperature differential between the warm surface water of the ocean and the cooler water at depth. Warm sea water is evaporated under reduced pressure, generates power as it passes through a generator, and is then condensed by the cooler sea water.

Other methods are also being investigated, based on such phenomena as freezing, osmosis, differential solubility in immiscible liquids, and others.

The most certain path to success in demineralization of saline water lies in developing a cheap source of energy. Such sources may include waste heat, nuclear fission, marine temperature differences, geothermal energy, and solar energy. Study by the Interior Department program indicated that waste heat had promise only in special cases. Solar energy—free everywhere and plentiful in many areas—appears attractive until the substantial capital investments required are considered. Interior Department investigations show some promise that possible improvements may make utilization of solar energy competitive in some particular locations.

COSTS OF DEMINERALIZED WATER

Costs of water vary greatly over the United States. Irrigation water ranges from $1.50 to $20 per acre-foot in California or $0.003 to $0.06 per thousand gallons. Municipal water costs range from a few cents up to 20 cents per thousand gallons. The cost of industrial water is even more variable, since industry often spends many times the price of raw water for treatment that will make supplies meet quality requirements.

The cost of demineralization by conventional methods ranges from $2 to $4 per thousand gallons. Estimates of cost for the more promising demineralization processes, based only on inadequate laboratory data, are as low as 30 to 70 cents per thousand gallons.

BYPRODUCTS OF DEMINERALIZATION

Minerals from saline water offer little promise of materially influencing the costs of demineralization processes. The worth of minerals dissolved in 1,000 gallons of sea water is much too small compared to the price of water.

However, in certain metropolitan areas, the value of salts recovered could add some byproduct credit. On the other hand, the costs of brine or salt disposal may be prohibitive for some processes, if pollution problems are to be avoided.

Nevertheless, the large number of elements present in sea water presents an intriguing possibility as a new source of minerals and metals.

OUTLOOK

Water problems probably will increase in the immediate future. Industrial growth and concentration, the spread of air conditioning, population increase, and other factors will tend to place greater pressure on the water resources in particular localities. This may force the adoption of such expensive practices as the present high-cost demineralization processess in some places in an effort to provide water supplies. In many instances costly reservoirs and tunnels will be constructed. Reclamation of used water should become more widespread, and the attack on water pollution already underway all over the country will be intensified.

The long-range outlook is not discouraging. Promising processes for demineralization, for pollution prevention, and for conservation of water will continue to be investigated, and the possibilities for successful development of methods that will make vast supplies of good water available are far from remote.

PROBLEMS

Water enters into commerce in a manner unlike many commodities. In fact, many users take its availability as a matter of course. It is produced, transported, and marketed over relatively small areas. Thus, water problems are generally due to the local effects of uneven or changing natural distribution of supplies and rising local demands.

The water problems of the mineral industries sometimes involve excesses as well as shortages of water. Although the physical aspects of such problems are tremendous, the economic aspects often have paramount importance.

Many methods of overcoming water-supply problems have been or are now being considered. These include reclamation of used water, artificial precipitation, and demineralization of ocean and brackish waters. In all these, technologic and economic considerations are closely interrelated.

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ZINC

By

O. M. Bishop [1] and R. L. Mentch [1]

ZINC is a vitally important metal of the group forming the basic framework of modern industry. Its chief use is in preserving steel products and structures, but it is essential and irreplaceable in brass and finds extensive uses in die castings, photoengraving plates, and dry-cell battery cases. As zinc oxide it is used widely in vulcanizing rubber and paint formulas.

Summary

Zinc is one of the nonferrous metals consumed in large quantity. Although used in brass for about 2,000 years, only in the past 100 years or so has it attained additional usage for its own properties. Currently it finds extensive use in galvanizing, followed by die castings, brass, pigments, and rolled products.

The United States is the world's leading producer of zinc ores and its largest producer and consumer of zinc metal. During the past 30 years United States mines have produced one-third of all mine output, its smelters have reduced 40 percent of the world's zinc metal, and its industries have consumed 40 percent of the world's supply. In recent years, however, world mine production has increased markedly while that of the United States has remained relatively constant, annually averaging 629,000 tons—about 24 percent of world mine output in 1950–53.

World War II marked the beginning of United States dependence on imports for a large portion of its needs. For many years the United States had been a net exporter of zinc, but beginning in 1940 and continuing thereafter greatly increased domestic requirements necessitated importation of large tonnages of foreign metal, ores, and concentrates to augment home supplies. Return to self-sufficiency is unlikely.

Canada and Mexico are the second and third largest producers of zinc and historically the chief sources of imports to the United States. Other major producers of zinc on a mine basis in order of output in 1953 were U. S. S. R., Australia, Peru, Belgian Congo, Poland, Japan, Italy, West Germany, and Spain. Together with the United States these 11 countries furnished 87 percent of the total world mine production in 1953.

Total developed world reserves of measured and indicated zinc ore are estimated to contain in excess of 70 million tons of zinc, and in all probability the content of inferred ore reserves is from 2 to 3 times as large. Canada has the largest reserves on a national basis followed by Australia and the United States. Other countries having important reserves are Mexico, Peru, Argentina, Bolivia, Germany, Italy, Spain, Sweden, Poland, U. S. S. R., Algeria, French Morocco, Tunisia, Belgian Congo, Northern Rhodesia, South-West Africa, Burma, and Japan.

The various segments of the zinc industry in the United States—mining, smelting, secondary-recovery operations, fabricating, and marketing—are not

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[1] Commodity-industry analyst, Bureau of Mines.

dominated to any marked degree by a few companies. The six largest producers on a mine basis supply less than half of the total United States output. Thirteen companies are engaged in primary reduction operations, and fabrication is controlled to a large extent by numerous independent companies.

World production of zinc in recent years (1948–54) has been well over world requirements, and consequently, large stocks have accumulated. Much of the excess production was shipped to the United States where accumulating stocks and the prospect of still more imports brought about unusually low prices and the closing of many mines. As a result, domestic mine production in 1954 declined to the lowest rate in about 20 years. In the second half of 1954 resumption of United States stockpiling and increased world consumption tended to alleviate the situation somewhat. One solution of the world problem of oversupply is increased consumption, with careful market research guiding any future production expansion programs; since consumption is stimulated by low zinc prices, efforts to improve extraction techniques and to lower costs per unit of output will help to solve this problem.

BACKGROUND

HISTORY OF ZINC INDUSTRY

Zinc has played a role in civilized man's activities from the time of Caesar Augustus, when brass, an alloy of copper and zinc, was discovered.

The demand for zinc, as for other metals, was greatly increased by the mechanization that characterized the industrial revolution. Uses for brass were multiplied and new uses for zinc and its compounds developed. In response to this demand world smelter production, which totaled only 50,000 tons in 1850, began doubling every 15 years to exceed 800,000 tons by 1910. Thereafter world production of zinc was spurred by strong demand, but at a lesser rate, until a high of 2,557,000 short tons was reached in 1953, or about triple that of 1910.

Before 1860 most of the small requirements of the United States were supplied by imports from England, Belgium, Germany, or Austria. The first recorded smelting of zinc in the United States was that by John Hitz at the Government Arsenal in Washington, D. C., in 1835–36. The ore was from Franklin, N. J., and the zinc metal produced was alloyed to make brass from which standard weights and measures were fabricated. In 1848 the Sussex Zinc & Copper Mining & Manufacturing Co., forerunner of The New Jersey Zinc Co., erected a small smelter at Newark. It was this smelter that first commercially produced zinc oxide directly from ore 4 years later.

In 1858 or 1859 the Lehigh Zinc Co. erected a Belgian-type furnace at Friedensville, Pa., and in 1860 J. Wharton built a similar plant at South Bethlehem, Pa. That same year the Matthiessen & Hegeler smelter was erected at La Salle, Ill., to smelt zinc ores from the Upper Mississippi Valley and southeast Missouri. Important zinc ore bodies had already been discovered in the Joplin area of southwest Missouri, but it was not until 1872 that the railway was completed permitting large shipments of these rich ores to La Salle. By 1860 the slab-zinc production of the Lehigh Zinc Co. had increased to 750 tons. In 1864 zinc was first used in the United States to coat steel. The demand for zinc-coated or galvanized products developed rapidly and aided in stimulating demand for zinc so that by 1880, 13 smelters were producing about 23,000 tons of slab zinc a year in all. Statistics of primary smelter production collected annually since 1882 show a strong, growing industry stimulated by the discovery of rich ore bodies and by the demand for zinc in galvanizing, brassmaking, and the rolling of sheet zinc. In 1890 primary slab-zinc production in the United States totaled 64,000 tons and 10 years later 124,000 tons.

The domestic smelting industry continued to grow, and in 1914 production of primary slab zinc from domestic and foreign ores totaled 353,000 tons. During World War I the demand for ammunition brass increased greatly. Furthermore Great Britain, which had obtained its zinc from Belgium and Germany, was forced to seek a new source. To meet this demand new smelting facilities were built in the United States at an unprecedented rate, and by 1917 primary slab production was 670,000 tons or 90 percent above the 1914 level. For the 5 years 1916–20 slightly over 60 percent of the world supply of slab zinc was produced in the United States.

Beginning with 1914 and continuing to the present zinc production in the United States at both mine and smelter levels has exceeded that of any other country.

SIZE, ORGANIZATION, PRODUCTION, AND PLANT LOCATION

Zinc is 1 of the 5 metals produced and used in greatest quantity by industrialized nations, both in war and peace. In the United States the output of primary zinc in all its forms in the decade 1941–50 approximately equaled the individual production of copper, aluminum, and lead and was about one-hundredth that of steel. In 1925–53 smelter production of primary slab zinc in the United States ranged from 207,000 to 942,000 short tons annually and averaged 646,000 tons. Smelter production for the world during the same years (see table 1) ranged from 861,000 to 2,557,000 tons annually and averaged 1,691,000 tons. In that 29-year period United States smelters produced 38 percent of the world supply of zinc. United States mine production in the same years ranged from about 19 to 48 percent of the world mine production and averaged about 32 percent.

TABLE 1.—*Comparison of zinc industry of the United States with world totals, 1925–1953* [1]

[All data in thousand short tons]

Year	Mine production			Smelter production (slab zinc)[2]			Consumption (slab zinc)[2]		
	Total	United States	Remainder of world	Total	United States	Remainder of world	Total	United States	Remainder of world
1925	1, 471	711	760	1, 249	573	676	1, 323	538	785
1926	1, 715	775	940	1, 343	618	725	1, 372	584	788
1927	1, 748	719	1, 029	1, 440	593	847	1, 454	550	904
1928	1, 670	695	975	1, 545	603	942	1, 575	580	995
1929	1, 793	724	1, 069	1, 599	625	974	1, 614	596	1, 018
1930	1, 612	595	1, 017	1, 536	498	1, 038	1, 415	436	979
1931	1, 220	410	810	1, 099	292	807	1, 159	315	844
1932	996	285	711	861	207	654	894	212	682
1933	1, 250	384	866	1, 083	307	776	1, 123	346	777
1934	1, 491	439	1, 052	1, 287	364	923	1, 283	357	926
1935	1, 600	518	1, 082	1, 467	421	1, 046	1, 504	477	1, 027
1936	1, 779	576	1, 203	1, 613	492	1, 121	1, 683	574	1, 109
1937	1, 933	626	1, 307	1, 789	557	1, 232	1, 805	614	1, 191
1938	1, 940	517	1, 423	1, 726	446	1, 280	1, 641	413	1, 228
1939	1, 962	584	1, 378	1, 818	507	1, 311	(3)	[4] 576	(3)
1940	2, 132	665	1, 467	1, 786	675	1, 111	(3)	[4] 684	(3)
1941	2, 314	749	1, 565	1, 928	822	1, 106	(3)	[4] 767	(3)
1942	2, 314	768	1, 546	1, 984	892	1, 092	(3)	[4] 675	(3)
1943	2, 224	744	1, 480	2, 028	942	1, 086	(3)	[4] 769	(3)
1944	1, 883	719	1, 164	1, 791	869	922	(3)	[4] 840	(3)
1945	1, 664	614	1, 050	1, 404	765	639	(3)	[4] 803	(3)
1946	1, 678	575	1, 103	1, 549	728	821	1, 653	757	896
1947	1, 819	638	1, 181	1, 758	802	956	1, 782	727	1, 055
1948	1, 911	630	1, 281	1, 865	788	1, 077	1, 834	756	1, 078
1949	2, 105	593	1, 512	2, 012	815	1, 197	1, 780	657	1, 123
1950	2, 359	623	1, 736	2, 172	843	1, 329	2, 123	901	1, 222
1951	2, 535	681	1, 854	2, 315	882	1, 433	2, 217	886	1, 331
1952	2, 833	666	2, 167	2, 425	904	1, 521	2, 110	798	1, 312
1953	2, 844	547	2, 297	2, 557	916	1, 641	2, 299	936	1, 363
Average	1, 889	613	1, 276	1, 691	646	1, 045	1, 620	591	1, 029

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[1] SOURCES: National Security Resources Board, Materials Survey—Zinc, 1951; Yearbooks of American Bureau of Metal Statistics, and various Bureau of Mines data. It should be borne in mind that world-production data contain numerous estimates, that some nations report on a content basis, others are gaged by exports, while data for some are almost wholly estimated from incomplete information. Consumption data are also largely estimates.

[2] Primary metal only.

[3] Not available.

[4] Not included in average.

Zinc mineral deposits occur in economic quantity and are mined and smelted in many countries and on every continent. In 1953 no less than 21 countries each mined over 10,000 tons of zinc, and another 19 had smaller productions. The greatest concentration of mine production was in the United States, followed by Canada, Mexico, U. S. S. R., Australia, Peru, Belgian Congo, Poland, Japan, and Italy—each of which produced over 100,000 metric tons and all of which have smelting facilities. World mine and smelter output by countries is given in tables 2 and 3. In 1953 there were over 75 smelters and electrolytic reduction plants in 21 different countries; zinc-reduction plants in the United States and foreign countries are listed in tables 6 and 7.

The domestic zinc industry includes approximately 500 firms which mine domestic ores, import foreign ores, collect scrap, and smelt or otherwise process these raw materials into slab zinc, zinc pigments, dust, alloys or other primary zinc products for sale. This industry group can be broadly classified as mining companies, smelting companies (including secondary smelters), and fabricating companies.

TABLE 2.—*World mine production of zinc (content of ore),*[1] *by countries,*[2] *1944–48 (average) and 1949–53*[3]

[Metric tons]
(Compiled by Pauline Roberts and Berenice B. Mitchell)

Country [2]	1944–48 (average)	1949	1950	1951	1952	1953
North America:						
Canada	266, 616	261, 506	284, 153	309, 450	337, 291	362, 909
Guatemala	----------	(4)	332	6, 500	8, 200	6, 100
Honduras [5]	5	201	94	140	287	578
Mexico	188, 657	178, 402	223, 530	180, 064	227, 375	226, 538
United States [6]	576, 135	538, 142	565, 513	617, 961	604, 183	496, 618
South America:						
Argentina	15, 044	10, 921	12, 699	15, 475	15, 396	16, 089
Bolivia (exports)	18, 444	17, 666	19, 570	30, 535	35, 619	23, 974
Peru	59, 077	72, 037	87, 961	101, 300	127, 845	134, 127
Europe:						
Austria	2, 126	2, 694	2, 970	3, 355	4, 986	4, 378
Finland [7]	2, 200	2, 500	2, 100	3, 000	7, 000	3, 200
France	4, 389	10, 907	12, 178	13, 283	14, 600	[6] 12, 000
Germany, West	69, 565	57, 816	70, 153	75, 294	80, 680	91, 618
Greece	1, 170	3, 100	3, 184	6, 300	7, 300	7, 500
Ireland	----------	----------	762	2, 355	3, 124	(4)
Italy	38, 373	74, 562	87, 026	100, 733	112, 914	101, 540
Norway	4, 631	6, 603	5, 702	5, 469	5, 588	5, 000
Poland [8]	69, 930	85, 300	86, 200	86, 200	95, 300	110, 000
Spain [7]	38, 200	50, 000	64, 000	74, 000	86, 000	83, 000
Sweden	35, 148	35, 158	37, 121	38, 318	47, 162	41, 538
U. S. S. R.[7] [8]	95, 800	110, 000	128, 800	148, 000	186, 000	212, 000
United Kingdom	2, 484	----------	36	194	1, 549	2, 891
Yugoslavia	21, 054	44, 017	38, 092	39, 420	47, 789	59, 970
Asia:						
Burma	----------	----------	----------	----------	750	3, 900
India [7]	----------	----------	300	1, 100	2, 000	2, 100
Indochina	357	----------	----------	----------	----------	----------
Iran	----------	----------	----------	----------	12, 000	5, 000
Japan	35, 712	44, 268	52, 032	64, 416	87, 468	104, 670
Korea, Republic of	2, 282	----------	----------	(4)	500	20
Philippines	----------	----------	50	150	1, 600	750
Thailand (Siam)	[9] 5	70	270	520	500	(4)
Turkey [7]	608	200	60	500	1, 200	(4)
Africa:						
Algeria	3, 961	6, 863	7, 167	9, 466	11, 192	19, 160
Angola	----------	----------	----------	350	40	100
Belgian Congo	33, 037	55, 420	74, 805	88, 705	98, 948	125, 791
Egypt	[10] 60	284	382	1, 432	886	200
French Equatorial Africa	224	44	621	518	377	----------
French Morocco	1, 470	2, 847	11, 412	19, 455	28, 352	35, 460
Nigeria	74	72	----------	----------	52	64
Northern Rhodesia [8]	18, 334	23, 217	23, 080	22, 953	23, 257	25, 737
South-West Africa	3, 197	12, 700	11, 300	14, 800	15, 600	15, 800
Tunisia	1, 630	3, 337	2, 932	3, 548	3, 540	3, 650
Australia	176, 652	184, 919	205, 632	197, 843	204, 380	[6] 204, 200
Total (estimate)	1, 788, 000	1, 910, 000	2, 140, 000	2, 300, 000	2, 570, 000	2, 580, 000

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[1] Data derived in part from the Yearbook of the American Bureau of Metal Statistics, the United Nations Statistical Yearbook, and the Statistical Summary of the Mineral Industry (Colonial Geological Surveys, London).

[2] In addition to countries listed Bulgaria, Czechoslovakia, East Germany, North Korea, and Rumania also produce zinc, but production data are not available; estimates included in total.

[3] Compiled by Division of Foreign Activities for the Zinc chapter of Bureau of Mines Minerals Yearbook, 1953.

[4] Data not available; estimate included in total.

[5] United States imports.

[6] Recoverable.

[7] Estimated.

[8] Smelter production.

[9] Average for 1 year only, as 1948 was first year of production.

[10] Average for 1947–48.

TABLE 3.—*World smelter production of zinc, by countries, 1944–48 (average) and 1949–53* [1] [2]

[Metric tons]

(Compiled by Pauline Roberts and Berenice B. Mitchell)

Country	1944–48 (average)	1949	1950	1951	1952	1953
North America:						
Canada	165,535	186,920	185,398	198,290	201,575	224,715
Mexico	49,057	53,496	53,492	58,750	[3] 50,387	[3] 53,053
United States	717,105	739,154	765,176	799,800	820,525	831,072
South America:						
Argentina	1,601	2,651	[4] 7,530	10,629	10,000	11,600
Peru	1,289	1,261	1,262	870	5,216	8,908
Europe:						
Belgium [5]	77,325	176,568	177,326	200,886	186,799	193,427
Czechoslovakia	[6] 2,466	(7)	(7)	(7)	(7)	(7)
France	29,015	58,916	71,531	74,557	80,064	80,938
Germany:						
East	(7)	(7)	(7)	(7)	(7)	(7)
West	[6] 22,983	86,916	122,796	140,640	147,216	148,261
Italy	14,534	26,917	37,925	47,752	54,851	60,033
Netherlands	5,447	15,614	19,752	22,605	25,905	25,202
Norway	25,559	41,090	43,173	40,825	39,232	37,820
Poland	69,930	85,300	86,200	86,200	95,300	110,000
Rumania	[8] 2,533	[4] 3,200	3,000	(7)	(7)	(7)
Spain	18,792	19,551	21,264	21,345	21,358	22,911
Sweden	944	----	----	----	----	----
U. S. S. R.[4]	95,800	110,000	128,800	148,000	186,000	212,000
United Kingdom	68,863	65,144	71,418	70,851	69,839	73,875
Yugoslavia	4,260	9,903	12,315	13,223	14,463	14,549
Asia:						
China [4]	262	180	180	180	180	360
Indochina	[4] 125	----	----	----	----	----
Japan	27,520	32,232	49,008	56,340	70,032	80,112
Africa:						
Belgian Congo	----	----	----	----	----	7,801
Northern Rhodesia	18,334	23,217	23,080	22,953	23,257	25,737
Australia	79,158	82,255	84,995	78,246	88,841	91,625
Total (estimate)	1,525,000	1,825,000	1,970,000	2,100,000	2,200,000	2,320,000

[1] Data derived in part from the Yearbook of the American Bureau of Metal Statistics, the United Nations Monthly Bulletin and the Statistical Yearbook, and the Statistical Summary of the Mineral Industry (Colonial Geological Surveys, London).

[2] Compiled by Division of Foreign Activities for the Zinc chapter, Bureau of Mines Minerals Yearbook, 1953.

[3] In addition other zinc-bearing materials totaling 3,398 tons in 1952 and 27,477 in 1953.

[4] Estimate.

[5] Includes production from reclaimed scrap.

[6] 1945–48 average.

[7] Data not available; estimate included in total.

[8] 1946–48 average.

MINING

Domestic zinc-mining operations range from small enterprises by a single individual to large mines operated by corporations having more than one mining property and reduction plant. Mine production in 1952 came from approximately 600 mines operating in 22 States; 60 percent of these mines produced about 80 percent of the total output. Variations in size are evident from the fact that for that year the 6 leading mines supplied over 32 percent of the production and the 25 leading mines, 60 percent. In late 1952 and throughout 1953 many mines closed because of low zinc and lead prices, and by the end of 1953 the number of zinc-producing mines had decreased to approximately 400.

Most domestic zinc comes from mixed zinc and lead ores; the remainder is derived almost entirely from zinc and copper-zinc ores.

About two-thirds of the zinc-ore production normally comes from western mining districts or regions, chief of which are the Summit Valley (Butte) district, Montana; the Coeur d'Alene region, Idaho; the Warren (Bisbee) and Big Bug districts, Arizona; the Central and Magdalena districts, New Mexico; West Mountain (Bingham), Tintic, and Park City districts, Utah; Pioche district, Nevada; Red Cliff and Ten Mile districts, Colorado; and the Pend Oreille and Metaline districts, Washington.

Nearly all of the remaining ore production comes from five eastern or midwestern areas: Sussex County, N. J.; Mascot area, Tennessee; Austinville area, Virginia; St. Lawrence County, N. Y.; Upper Mississippi Valley area of southwestern Wisconsin and northwestern Illinois; and the Tri-State area of Missouri, Kansas, and Oklahoma. The Tri-State has been the largest producing district for many years but was exceeded by the Coeur d'Alene region in 1950 and 1953 and by the Butte district in 1953. Indications are that the Butte district or Coeur d'Alene region will, within the next few years, regularly lead in annual production. The 50 foremost producing districts and areas are listed in table 4, in the order of their 1953

TABLE 4.—*Mine production of recoverable zinc in the United States by districts that produced 1,000 tons or more in any of the past 5 years, 1944–48 (average) and 1949–53*

[Short tons]

District	State	1944–48 (average)	1949	1950	1951	1952	1953
Summit Valley (Butte)	Montana	23,337	47,982	63,511	80,500	75,968	75,170
Coeur d'Alene region	Idaho	78,748	74,370	86,103	74,989	70,316	68,650
Tri-State (Joplin region)	Kansas, southwestern Missouri, Oklahoma.	132,552	78,628	80,558	91,553	90,512	55,729
St. Lawrence County	New York	32,343	37,973	38,321	40,051	32,636	51,529
New Jersey	New Jersey	75,867	50,984	55,029	62,917	59,190	45,700
Eastern Tennessee [1]	Tennessee	32,001	29,788	35,326	38,639	38,020	38,465
Upper Mississippi Valley	Northern Illinois, Iowa,[2] Wisconsin,	17,208	17,846	26,793	31,403	34,716	26,286
West Mountain (Bingham)	Utah	16,787	22,311	16,120	18,286	20,395	19,669
Red Cliff	Colorado	17,293	17,450	19,956	29,200	26,000	16,850
Austinville	Virginia	16,766	13,166	[1] 12,396	[1] 7,332	13,409	16,676
Central	New Mexico	37,293	26,376	26,897	41,884	48,043	12,743
Big Bug	Arizona	4,955	8,798	10,416	9,688	10,862	10,476
Upper San Miguel	Colorado	1,960	6,004	8,881	9,228	9,811	10,414
Kentucky-Southern Illinois	Kentucky, southern Illinois.	5,768	6,541	6,642	9,584	7,968	5,589
Park City region	Utah	9,429	8,359	7,425	10,209	7,746	4,848
Harshaw	Arizona	1,945	2,947	4,193	4,076	3,924	4,186
California (Leadville)	Colorado	6,387	6,455	7,392	8,144	8,487	3,945
Cochise	Arizona	2,048	1,760	1,025	3,243	4,266	3,893
Warm Springs	Idaho	2,659	1,653	1,236	1,860	2,142	3,026
Smelter (Lewis and Clark County).	Montana	6,404	1,463	2,358	2,428	2,807	2,924
Pioneer (Rico)	Colorado	3,705	1,354	1,365	2,527	2,734	2,634
Eureka (Bagdad)	Arizona	712	2,304	1,478	2,504	3,520	2,594
Tintic	Utah	3,547	6,082	5,985	3,410	2,951	2,433
Aravaipa	Arizona	382	783	921	1,404	1,315	1,732
Rush Valley and Smelter (Tooele County).	Utah	5,901	2,188	1,219	1,608	916	1,528
Silver Bell	Arizona	46	1	11	--------	364	1,324
Breckenridge	Colorado	720	362	427	366	620	1,200
Warren (Bisbee)	Arizona	21,747	35,393	20,707	4,511	4,791	1,182
Verde (Jerome)	do	--------	4,350	7,800	10,155	4,360	959
Creede	Colorado	22	671	873	892	1,024	858
Animas	do	1,004	1,029	961	1,183	986	541
Magdalena	New Mexico	4,172	2,263	1,677	2,276	2,122	512
Patagonia (Duquesne)	Arizona	688	555	368	601	1,049	257
Pima (Sierritas, Papago, Twin Buttes).	do	4,660	7,177	5,802	5,414	3,472	11
Chelan Lake [3] [4]	Washington	1,902	2,724	2,430	1,879	(5)	(5)
Coso [4]	California	1,501	4,062	5,237	4,720	5,479	(5)
Flint Creek [4]	Montana	63	8	120	392	1,084	(5)
Metaline [4]	Washington	8,091	6,496	11,032	12,753	(5)	(5)
Northport [4]	do	2,139	1,412	1,304	3,496	(5)	(5)
Ophir [4]	Utah	440	1,004	374	341	670	(5)
Pioche [4]	Nevada	16,659	18,651	19,655	14,350	12,493	(5)
Sneffels [4]	Colorado	477	1,053	810	1,094	931	(5)
Cow Creek (Ingot)	California	36	(5)	(5)	(5)	(5)	--------
Heddleston	Montana	1,568	2,026	892	1,395	1,066	--------
Old Hat (Oracle)	Arizona	3,746	5,195	4,603	3,583	3,368	--------
Pioneer (Superior)	do	1,229	--------	2,595	6,240	4,175	--------
Smelter (Cascade County)	Montana	--------	1,278	--------	--------	--------	--------
Ten Mile	Colorado	4,208	9,716	2,925	16	12	--------
Tomichi	do	994	1,456	963	1,011	874	--------
Yellow Pine (Goodsprings)	Nevada	702	447	643	1,332	1,464	--------

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[1] Includes very small quantity produced elsewhere in State.

[2] No production in Iowa since 1917.

[3] Includes Peshastin Creek and Wenatchee River districts.

[4] This district is not listed in order of 1953 output.

[5] Quantity withheld to avoid disclosure of individual company operations.

TABLE 5.—*Mine production of recoverable zinc in the United States, 1944–48 (average) and 1949–53, by States*

[Short tons]

State	1944–48 (average)	1949	1950	1951	1952	1953
Western States and Alaska:						
Alaska	9	2	6	1	----------	----------
Arizona	44, 418	70, 658	60, 480	52, 999	47, 143	27, 530
California	7, 199	7, 209	7, 551	9, 602	9, 419	5, 358
Colorado	39, 157	47, 703	45, 776	55, 714	53, 203	37, 809
Idaho	83, 135	76, 555	87, 890	78, 121	74, 317	72, 153
Montana	35, 015	54, 195	67, 678	85, 551	82, 185	80, 271
Nevada	20, 413	20, 443	21, 606	17, 443	15, 357	5, 812
New Mexico	42, 546	29, 346	29, 263	45, 419	50, 975	13, 373
Oregon	----------	6	21	3	1	----------
South Dakota	21	----------	----------	----------	----------	----------
Texas	13	----------	----------	24	3	----------
Utah	37, 216	40, 670	31, 678	34, 317	32, 947	29, 184
Washington	12, 273	10, 740	14, 807	18, 189	20, 102	32, 786
Total	321, 415	357, 527	366, 756	397, 383	385, 652	304, 276
West Central States:						
Arkansas	91	1	8	50	26	----------
Kansas	47, 375	29, 433	27, 176	28, 904	25, 482	15, 515
Missouri	20, 914	5, 911	8, 189	11, 476	13, 986	9, 981
Oklahoma	65, 037	44, 033	46, 739	53, 450	54, 916	33, 413
Total	133, 417	79, 378	82, 112	93, 880	94, 410	58, 909
States east of the Mississippi River:						
Illinois	9, 485	18, 157	26, 982	21, 776	18, 816	14, 556
Kentucky	397	935	731	3, 457	3, 280	489
New Jersey	75, 867	50, 984	55, 029	62, 917	59, 190	45, 700
New York	32, 343	37, 973	38, 321	40, 051	32, 636	51, 529
Tennessee	32, 001	29, 788	35, 326	38, 639	38, 020	38, 465
Virginia	17, 063	13, 166	12, 396	7, 332	13, 409	16, 676
Wisconsin	13, 095	5, 295	5, 722	15, 754	20, 588	16, 830
Total	180, 251	156, 298	174, 507	189, 926	185, 939	184, 245
Grand total	635, 083	593, 203	623, 375	681, 189	666, 001	547, 430

production, and the mine production of recoverable zinc, by States, in recent years, is given in table 5.

All ores, with some minor exceptions, are milled near the mines to produce concentrates for shipment to the smelters.

SMELTING

The United States zinc-smelting industry consists of 18 primary reduction plants and 12 secondary distilling plants that produce slab zinc. Most of the primary smelting plants are integrated with producing mines that directly supply zinc concentrate. The primary plants also operate as custom smelters, buying concentrates and ores from foreign and domestic mines and selling the resulting slab zinc, zinc dust, or pigment on the market. Such smelters also smelt on a toll or fee basis by which ownership of the raw material and the product remains with the shipper. In addition to ores and concentrates these plants treat some scrap materials. The 18 primary reduction plants in 1953 had a capacity of 1,089,000 tons of slab zinc. They actually operated at only 86 percent of that capacity, producing 916,000 tons of primary slab zinc and 18,000 tons of secondary redistilled zinc in 1953, a total of 934,000 tons. In addition to the 934,000 tons of slab zinc produced at primary smelters, 35,000 tons were produced by redistillation of zinc scrap at secondary smelters that treat only that material. Several primary smelters also produce zinc oxide, leaded zinc oxide, and lithopone, and one company produced a substantial tonnage of zinc dust. Still another important product of many primary smelters is sulfuric acid, which is made from the sulfur dioxide gas produced in roasting sulfide zinc ores.

Table 6, which lists domestic smelters and locations, subdivides the 18 primary smelters into groups based upon the method of reduction and gives the annual slab-zinc capacity of each group. The 5 electrolytic plants had 36 percent, the 9 horizontal retort plants 36 percent, the 4 vertical retort plants 23 percent, and the 13 secondary distillation plants 5 percent of the total industry capacity for slab zinc as of the end of 1953. Table 7 lists known foreign zinc-reduction plants by location, type, and capacity.

TABLE 6.—*United States zinc smelters: Their location and group capacity for slab zinc, in 1953*

[Short tons]

ELECTROLYTIC PLANTS	
American Smelting & Refining Co.	Corpus Christi, Tex.
American Zinc Co. of Illinois.	Monsanto, Ill.
Anaconda Copper Mining Co.	Anaconda, Mont.
Do	Great Falls, Mont.
Sullivan Mining Co	Kellogg, Idaho.
Total capacity	412, 500 tons
HORIZONTAL-RETORT SMELTERS	
American Smelting & Refining Co.	Amarillo, Tex.
United States Steel Corp. (American Steel & Wire Div.)	Donora, Pa.
American Zinc Co. of Illinois.	Fairmont City, Ill.
Do	Dumas, Tex.
Athletic Mining & Smelting Co.	Fort Smith, Ark.
Blackwell Zinc Co	Blackwell, Okla.
HORIZONTAL-RETORT SMELTERS—Con	
Eagle-Picher Co. (Mining and Smelting Div.)	Henryetta, Okla.
Matthiessen & Hegeler Zinc Co.	La Salle, Ill.
National Zinc Co., Inc	Bartlesville, Okla.
Total capacity	416, 500 tons
VERTICAL-RETORT SMELTERS	
Meadowbrook Corp	Meadowbrook, W. Va.
New Jersey Zinc Co	Depue, Ill.
New Jersey Zinc Co. of Pa	Palmerton, Pa.
St. Joseph Lead Co	Josephtown, Pa.
Total capacity	260, 000 tons
SECONDARY SMELTERS	
American Smelting & Refining Co.	Los Angeles, Calif.
Do	Beckemeyer, Ill.
Do	Sand Springs, Okla.
Do	Trenton, N. J.
American Zinc, Lead & Smelting Co.	Hillsboro, Ill.
Bullock, W. J., Inc	Fairfield, Ala.
General Smelting Co	Philadelphia, Pa.
Nassau Smelting & Refining Co., Inc.	Tottenville, N. Y.
Pacific Smelting Co	Torrance, Calif.
Sandoval Zinc Co	Sandoval, Ill.
Superior Zinc Corp	Bristol, Pa.
Wheeling Steel Corp	Wheeling, W. Va.
Total capacity	58, 000 tons
Total slab-zinc capacity	1, 147, 000 tons

TABLE 7.—*Foreign zinc-smelter capacity, 1953* [1]

ELECTROLYTIC PLANTS

Location	Company	Estimated annual capacity for slab zinc (short tons)
Argentina: Zarate	"Meteor" Est. Met. S. A. I.yC	6, 600
Australia: Risdon, Tasmania	Electrolytic Zinc Co. of Australasia	114, 200
Belgian Congo: Kolwezi	Soc. Métallurgique de Katanga	39, 700
Belgium: Baelen	Soc. Anon. de la Vieille Montagne	41, 900
Canada:		
Trail, B. C	Consolidated Mining & Smelting Co	190, 000
Flin Flon, Manitoba	Hudson Bay Mining & Smelting Co., Ltd	69, 350
France: Viviez, Aveyron	Soc. Anon. de la Vieille Montagne	49, 600
Italy:		
Monteponi, Sardinia	Soc. di Monteponi	9, 900
Crotone	Soc. Min. e Met. di Pertusola	25, 400
Nossa, Bergamo	S. A. P. E. Z. Soc. per Azioni Piombo e Zinco	19, 800
Porto Marghera	Montevecchio Soc. Italiana del Piombo e dello Zinco	26, 500
Japan:		
Kosaka, Akita-ken	Dowa Mining Co. Ltd	4, 000
Hosokura	Mitsubishi Metal Mining Co. Ltd	13, 200
Naoshima	do	3, 700
Kamioka	Mitsui Mining & Smelting Co. Ltd	15, 900
Miike	do	8, 600
Aizu	Nihon Soda K. K	11, 200
Annaka	Toho Aen Kogyo K. K	13, 200
Northern Rhodesia: Broken Hill (Sable).	Rhodesia Broken Hill Dev. Co	30, 200
Norway: Eitrheim	Det Norske Zinkkompani, A. S	46, 300

See footnote at end of table.

TABLE 7.—*Foreign zinc-smelter capacity, 1953* [1]—Continued

ELECTROLYTIC PLANTS—continued

Location	Company	Estimated annual capacity for slab zinc (short tons)
Peru	Cerro de Pasco Corp	13, 000
U. S. S. R.:		
Dzhaudzhukau (Caucasus)	State-owned	44, 100
Chelyiabinsk (Urals)	do	44, 100
Total		840, 000
	RETORT PLANTS	
Argentina: Comodoro Rivadavia	Cia. Metalúrgica Austral S. A	17, 600
Belgium:		
Corphalie	Cie. des Métaux d'Overpelt-Lommel et de Corphalie	39, 700
Overpelt	do	39, 700
Lommel	do	39, 700
Prayon-lez-Trooz and Engis	Soc. Anon. Métallurgique de Prayon	60, 600
Rothem	Soc. Anon. de Rothem	33, 100
Angleur	Soc. Anon. de la Vieille Montagne	28, 700
Flone	do	52, 900
Valentin-Cocq	do	28, 700
Czechoslovakia:		
Kutterschitz	State-owned	16, 500
Settenz	do	8, 800
France:		
Noyelles-Godault (Pas de Calais)	Soc. Minière et Métallurgique de Peñarroya	16, 500
Creil, Seine-et-Oise	Soc. Anon. de la Vieille Montagne	11, 000
Auby (Nord)	Cie. Royale Asturienne des Mines	41, 900
Mortagne (Nord)	do	9, 400
Federal Republic of Germany:		
Münsterbusch	Stolberger Zink A. G. f. Bergbau u. Zinkhüttenbetrieb	35, 300
Nievenheim	do	18, 700
Friedrich-August Hütte, Post Nordenham.	Metallwerke Unterweser A.-G	26, 500
Borbeck	A.-G. des Altenbergs f. Bergbau u. Zinkhüttenbetrieb	33, 100
Duisburg-Wanheim	"Berzelius" Metalhütten G.m.b.H	36, 400
Harlingerode (Oker)	Unterharzer Berg-u. Hüttenwerke G.m.b.H	55, 100
Italy: Vado Ligure	Soc. di Monteponi	13, 200
Japan:		
Omuda (Miike Wks.)	Mitsui Mining & Smelting Co	26, 800
Shimonoseki (Hikoshima Wks.)	do	18, 500
Mexico: Rosita, Coah	American Smelting & Refining Co. (Mexican Zinc Co.)	60, 000
Netherlands: Budel	Zincs de la Campine	54, 000
Poland:		
Welnowiec	Hohenlohe Werke	19, 800
Lipiny	Slaskie Kopalnie i Cynkownie	70, 500
Bogucice	do	
Nowy-Bytom	do	
Wirek	do	
Buchacs	do	
Szopienice and Trzebinia	Giesche Spólka Akcyjna	94, 800
Zagorze	Soc. des Charbonnages, Mines et Usines de Sosnowice (Pauline Hütte).	6, 100
Chropaczow	Dyrekcja Kopain i Hut Ksiecia Donnersmarcka	8, 500
Spain:		
Arnao	Cie. Royale Asturienne des Mines	30, 900
Peñarroya (Prov. de Córdoba)	Soc. Minière et Métallurgique de Peñarroya	8, 300
United Kingdom: Avonmouth and Swansea, South Wales.	Imperial Smelting Corp. Ltd. (National Smelting Co. Ltd.).	88, 200
Yugoslavia: Celje	State-owned	17, 600
U. S. S. R.:		
Leninogorsk (Kazakhstan)	State-owned	165, 000
Belovo (Kousbas)	do	27, 500
Konstantinovka (Ukraine)	do	13, 000
Ust Kamenogorsk (Kazakhstan)	do	Not known, reportedly important.
Total		1, 373, 000

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[1] SOURCE: Yearbook of the American Bureau of Metal Statistics, 1953 and Minerais et Métaux Société Anonyme, 1953.

The plants treating secondary-zinc-bearing materials include zinc smelters, remelters, brass mills, alloy manufacturers, and chemical and pigments plants. Secondary zinc reported by the Bureau of Mines is that recovered from old and new scrap. Old zinc scrap consists of metal articles discarded because of wear, damage, or obsolescence, usually after use. New scrap is that generated during the manufacture of articles for ultimate consumption but is limited to the scrap processed for recovery of its metal content elsewhere than at the plant of generation. In other words, new scrap as reported by the Bureau of Mines excludes home scrap and is predominently galvanizers' dross, skimmings and ashes, die-cast skimmings, chemical residues, flue dust, sal skimmings, and metal clippings. Old scrap includes old brass, die castings, and engravers' plates. Secondary zinc recovered in 1925 through 1953 had a range of 75,000 to 368,000 tons annually and averaged 223,000 tons. In 1946–53 the average had increased to 302,000 tons or 23 percent of total supply. Recovery in brass and bronze constitutes the most important form of recovery, averaging well over 50 percent of the total in 1940–53. Zinc recovered in slab form and as zinc dust was about 25 percent of the total, and recovery in alloys, pigments, and chemicals comprised the remainder. Production by form of recovery and the kind of scrap processed in 1952 and 1953 is given in table 10. The annual recovery of secondary zinc in all forms from 1925 through 1953 is shown as an element of supply in table 12.

In addition to the zinc pigments and chemicals produced at primary and secondary smelters, considerable quantities of zinc oxide, leaded-zinc oxide, lithopone, zinc sulfate, and zinc chloride are produced by chemical and pigment manufacturers from zinc ores or zinc scrap. The zinc content of all pigments and salts made from either ores or secondary material totaled 159,000 tons in 1952 and 176,000 tons in 1953. Ores and concentrates were the chief raw material, supplying 109,000 and 118,000 tons of the zinc content of the chemicals and pigments made in 1952 and 1953, respectively.

Leadership in the United States smelting industry is largely vested in companies that combine substantial mine production with large, well-located reduction plants. One such concern, the Anaconda Copper Mining Co., produces about one-quarter of the slab zinc smelted in the United States. This firm not only treats company-mined ores but processes custom ores from other mines in two electrolytic plants at Anaconda and Great Falls, Mont. The St. Joseph Lead Co. and The New Jersey Zinc Co. hold dominant positions in the East, smelting both company-mined and purchased ores, while the American Zinc, Lead & Smelting Co. and the Eagle-Picher Co. are important factors in the mid-continent area. Other smelting firms exerting influence in production and marketing practice throughout the United States are the Matthiessen & Hegeler Zinc Co., the American Metal Co., the American Smelting & Refining Co., and the National Zinc Co.

Smelters are established on good routes of transportation, usually at some point near a source of cheap fuel and between the raw-material sources and the market. Electrolytic plants—requiring, as they do, about 4,000 kw.-hr. of power per ton of zinc reduced, are located in part to secure favorable power rates. Similarly, retort smelters, fired by natural gas, are situated on the basis of a plentiful supply of that cheap fuel and coal-fired smelters on the basis of adequate cheap and suitable coals. The marketing of byproduct sulfuric acid is yet another factor in plant location.

The interrelationship of these various factors has resulted in concentration of 91 percent of the primary slab-zinc capacity in Pennsylvania, Montana, Oklahoma, Illinois, and Texas, with Idaho, Arkansas, and West Virginia making up the remaining 9 percent. Table 8 shows the production of primary slab zinc by States from 1944–53 and table 9 production of slab zinc according to methods of reduction and grades.

TABLE 8.—*Primary slab zinc produced in the United States, by States where smelted, 1944–48 (average) and 1949–53*

[Short tons]

Year	Arkansas	Idaho	Illinois	Montana	Oklahoma	Pennsylvania	Texas and West Virginia [1]	Total	
								Short tons	Value
1944–48 (aver.)	22, 441	37, 674	118, 138	199, 095	116, 769	190, 127	106, 233	790, 477	$158, 481, 977
1949	17, 116	41, 854	86, 823	216, 578	157, 650	156, 920	137, 841	814, 782	202, 391, 849
1950	20, 688	53, 922	108, 301	216, 104	145, 117	162, 539	136, 796	843, 467	240, 050, 708
1951	21, 776	54, 468	108, 544	208, 482	161, 247	189, 177	137, 939	881, 633	321, 619, 718
1952	21, 644	54, 340	115, 331	214, 980	161, 242	193, 811	143, 131	904, 479	300, 829, 715
1953	20, 379	54, 037	129, 904	222, 354	134, 918	192, 279	162, 234	916, 105	210, 154, 487

[1] Includes Missouri, 1944 and 1947–53.

TABLE 9.—*Distilled and electrolytic zinc, primary and secondary, produced in the United States, 1944–48 (average) and 1949–53*

[Short tons]

CLASSIFIED ACCORDING TO METHOD OF REDUCTION

Year	Electrolytic primary	Distilled	Redistilled secondary		Total
			At primary smelters	At secondary smelters	
1944–48 (average)	295, 321	495, 156	22, 890	30, 041	843, 408
1949	326, 152	488, 630	22, 782	32, 259	869, 823
1950	342, 085	501, 382	28, 411	38, 559	910, 437
1951	336, 087	545, 546	16, 251	32, 406	930, 290
1952	351, 106	553, 373	18, 861	36, 250	959, 590
1953	370, 870	545, 235	17, 645	35, 230	968, 980

CLASSIFIED ACCORDING TO GRADE

Year	Grade A		Grade B	Grades C and D		Grade E	Total
	Special High-Grade (99.99 percent Zn)	High-Grade (Ordinary (99.90 percent Zn)	Intermediate (99.50 percent Zn)	Brass Special (99.00 percent Zn)	Select (98.75 percent Zn)	Prime Western (98.50 percent Zn)	
1944–48 (average)	239, 051	202, 102	42, 607	62, 498	14, 605	282, 545	843, 408
1949	230, 576	206, 651	21, 513	56, 388	2, 565	352, 130	869, 823
1950	271, 678	192, 075	21, 571	46, 730	4, 021	374, 362	910, 437
1951	281, 571	175, 499	20, 734	60, 511	13, 494	378, 481	930, 290
1952	295, 801	182, 125	17, 903	48, 817	13, 608	401, 336	959, 590
1953	312, 810	180, 188	14, 720	56, 219	1, 930	403, 113	968, 980

TABLE 10.—*Zinc recovered from scrap processed in the United States, 1952–53*

[Short tons]

Recoverable zinc content of scrap processed		
Kind of scrap	1952	1953
New scrap:		
Zinc-base	108, 273	110, 774
Copper-base	126, 625	117, 611
Aluminum-base	820	1, 985
Magnesium-base	40	73
Total	235, 758	230, 443
Old scrap:		
Zinc-base	24, 997	19, 622
Copper-base	49, 312	42, 888
Aluminum-base	226	1, 604
Magnesium-base	130	121
Total	74, 665	64, 235
Grand total	310, 423	294, 678

Zinc recovered[1] from scrap processed		
Form of recovery	1952	1953
As metal:		
By distillation:		
Slab zinc	54, 560	50, 344
Zinc dust	22, 292	22, 185
By remelting	6, 275	6, 116
Total	83, 127	78, 645
In zinc-base alloys	9, 875	8, 535
In brass and bronze	184, 935	168, 951
In aluminum-base alloys	1, 120	3, 673
In magnesium-base alloys	161	194
In chemical products:		
Zinc oxide (lead-free)	8, 914	11, 430
Zinc sulfare	3, 871	4, 566
Zinc chloride	10, 794	12, 981
Lithopone	6, 922	5, 008
Miscellaneous	704	695
Total	227, 296	216, 033
Grand total	310, 423	294, 678

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[1] Zinc content.

FABRICATION

About 40 percent of the slab zinc produced by smelters is used without fabrication in galvanizing iron and steel products, chiefly at the plants of steel producers; thus galvanizing is largely concentrated in such iron- and steel-producing States as Ohio, Pennsylvania, Illinois, Indiana, Maryland, and Alabama. Somewhat less than 5 percent of the zinc consumed in galvanizing is cast into anodes for electrogalvanizing.

The second largest use of zinc is in the manufacture of zinc-base alloys. Much of the alloy is made by alloyers, but the remainder is made at the die-casting plants. Since the automobile, home-appliance, and office-machine industries are the largest consumers of zinc-base alloys the consumption is largely concentrated in the manufacturing areas of Ohio, Michigan, Wisconsin, New York, Pennsylvania, and California.

The brass industry, the third most important consumer of zinc, is heavily concentrated in the Connecticut Valley, but Michigan, Illinois, and Ohio are important manufacturing areas for brass and consume large quantities of zinc in making the alloy.

Zinc rolling mills—largely concentrated in Illinois, Indiana, Pennsylvania, and New York—annually roll 50,000 to 90,000 tons of slab zinc to produce zinc plate, sheet, ribbon, foil, rod, and wire.

PRODUCTION AND CONSUMPTION TRENDS

Zinc supplies throughout the world slightly exceed total requirements at present. Even in 1951, when demand exceeded zinc supply, the actual consumption in the United States was well below the total supply level, and the shortage was felt by the consuming industries only because large shipments of metal were being withdrawn from the market for the National Stockpile. The emergency stocking of zinc beyond consumers' requirements in Great Britain further aggravated the apparent shortage. In early 1952 the true situation of adequate or excess supplies became apparent. Foreign imports of both ores and concentrates were available in greater quantity, smelter stocks of unsold zinc increased, and United States market prices fell well below the 19.50 cents a pound ceiling to average 16.21 cents in 1952, 10.86 cents in 1953, and 10.68 cents in 1954.

An analysis of the situation (see table 1, p. 980) indicates that the depressed market and excess stocks in the United States were brought about by world production of zinc over world requirements and the consequent flow of ores and metal to United States markets. Although world consumption of primary slab zinc averaged 1,975,000 tons annually in 1946–53 and had increased 39 percent, world output of slab zinc in the same years averaged 2,042,000 tons and had increased 65 percent. As a result stocks accumulated to depress the price. The fact that much of the foreign production is from large, high-grade deposits is a distinct advantage to foreign producers in competing in the United States zinc market against production from domestic mines in which costs are high and ore bodies relatively low grade. Details of domestic mine costs and ore grade for 1952 were published by the Tariff Commission (*32*),[2] and it was noted that although average labor costs per ton of ore are lower in the United States than in Canada or Mexico, the average Canadian and Mexican ores contain a greater value in metal and hence the cost per unit of metal produced is less.

In response to the lower prices beginning in mid-1952 mine production of recoverable zinc in the United States dropped from an annual rate of 720,000 tons during the first half of 1952 to an annual rate of 610,000 tons in the latter half of the year. Production dropped further to 547,000 tons in 1953, to 240,000 tons in the first 6 months of 1954, and to 464,000 tons for all of 1954. Purchases of newly mined domestic zinc by the United States Government beginning in June 1954 brought about a considerable reduction in domestic smelter stocks and have resulted in a 15-percent increase in the price of zinc.

Consumption of slab zinc declined from 986,000 tons in 1953 to 876,000 tons in 1954. In the last 3 months of 1954, however, consumption totaled 244,000 tons or an annual rate of approximately 975,000 tons.

TECHNOLOGY

ECONOMIC GEOLOGY OF ZINC

Zinc ores are aggregates of minerals, one or more of which contains zinc in economic quantity. The most common zinc mineral is sphalerite or zinc blende (ZnS), which, with its oxidation products smithsonite ($ZnCO_3$) and hemimorphite ($(ZnOH)_2SiO_3$), forms the chief zinc minerals of the world. Zincite (ZnO), willemite (Zn_2SiO_4), and franklinite ($(Fe, Zn, Mn)O: (Fe, Mn)_2O_3$) occur in a unique and very important zinc deposit at Ogdensburg, N. J., but apart from this one occurrence have little economic importance.

Sphalerite almost always occurs in association with galena, the sulfide of lead. It may also be associated with copper or other base-metal sulfides or occur alone.

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter

Most zinc ores occur as cavity fillings and replacements deposited by hydrothermal solutions, considered to be of igneous origin (*4*). They commonly occur in limestones and dolomites but also occur in important amounts in other rock types at many mines, including those of Butte, Mont.; the Coeur d'Alene, Idaho; Ogdensburg, N. J.; Cerro de Pasco, Peru; the Rammelsberg deposits, Germany; and the Broken Hill lode, Australia.

Although contact metasomatic deposits such as the one at Hanover, N. Mex., are locally important, they are uncommon and usually small. The principal classes of zinc deposits, with some examples of each follow:

1. Contact metasomatic: (Hanover, N. Mex.; Long Lake, Ontario).
2. Cavity fillings:
    - (*a*) Fissure veins (San Juan County, Colo.)
    - (*b*) Breccia (Jefferson City and Mascot, Tenn.; Austinville, Va.)
    - (*c*) Cave fillings (Mississippi Valley area)
    - (*d*) Pitches and flats (Upper Mississippi Valley)
3. Replacements:
    - (*a*) Massive (Leadville, Colo.; Bingham and Tintic, Utah; Sullivan mine, British Columbia; Santa Eulalia, Chihuahua, Mexico; Trepca, Yugoslavia)
    - (*b*) Replacement lodes (Park City, Utah; Coeur d'Alene, Idaho; Franklin Furnace, N. J.; Broken Hill, Australia; Edwards, Balmat, N. Y.)
    - (*c*) Diseminated (Tri-State district)
4. Surficial oxidation (residual hemimorphite and smithsonite in Virginia, Tennessee, Arkansas, and Missouri)

In general terms prospecting is the search for ore, while exploration is the work done to get data on the size, shape, location, and quality of an ore body. Early prospecting and exploration were confined to examination of rock outcrops in the search for gold, silver, copper, and lead. As the exposed commercial ore was largely discovered and the known ore bodies were worked, it became necessary to employ new prospecting techniques, involving careful geologic mapping of rock formations and structural features that control or may control ore deposition. Through intensive study mining geologists and exploration engineers have learned the mechanics of replacement, structural control, contact metamorphism, zonal distribution, structural barriers, wallrock alteration, oxidation, and tectonics both before and after ore deposition. More recently emphasis has been given to a greater utilization of physics, chemistry, and mathematics, not only to obtain an understanding of how ore bodies came into being, but, more important, where they were localized. Utilizing geochemical and geophysical techniques largely developed in petroleum exploration, geologists and engineers have embarked on trace-element analyses of plants and top soil and on various physical measurements in areas judged favorable (*5*, *17*).

MINING

A variety of mining methods, which vary with the type of ore body, is used in extracting zinc ore. Underground methods employing either open or supported stopes yield most production, but open-pit methods have found limited application. Underground stoping methods in use include block caving, room-and-pillar with and without roof bolts, shrinkage stopes, cut-and-fill stopes, and timbered stopes. Zinc deposits amenable to open-pit methods are restricted to relatively few areas, and at the present the only mines of any size employing the method are the Van Stone in Washington (*14*, *24*) and the Barvue (*23*) in Quebec. A description of mining methods, their advantages and disadvantages, is given in Bureau of Mines Bulletin 419 (*16*).

Improvements in mining practice in the past 40 to 50 years have been based largely on the development and extensive use of power equipment. Thirty years ago almost all zinc ores were loaded by hand shoveling, whereas now almost all loading is done by power shovels, scrapers, or mucking machines. Transportation also has undergone great changes, as hand tramming and mulepower have been replaced by motortrains operating on heavy-gage track. More recently trackless mining, which utilizes electric- or diesel-powered units, is finding wide usage in the zinc and lead-zinc mines of Washington (*12*, *18*), the Tri-State district, the Upper Mississippi Valley (*1*), and Tennessee (*35*).

Other improvements in equipment that have done much to raise mine output per man-shift include better lighting at working places, better ventilation (including air conditioning), better and more efficient pumps, and harder hitting light rock drills using carbide bits.

It is believed the United States will continue to be a major zinc-ore- and metal-producing country as important ore bodies continue to be discovered and developed and more efficient operating techniques are devised. Major exploration, development, modernization, and expansion projects undertaken since January 1950 include mine and mill expansion at the Eagle mine, Gilman, Colo.; deep-level exploration at the Crescent and Hercules mines and elsewhere in the Coeur d'Alene region, Idaho; and introduction of block caving in the same area; development and plant construction, Indian Creek mine, Mo.; exploration and development work, Alice-Lexington mine, Butte, Mont.; mine development and plant construction, Sterling mine, Ogdensburg, N. J.; expansion of Balmat mine and mill, Balmat, N. Y.; exploration and development, Eureka Corp., Ltd. mine, Eureka, Nev.; development and construction at Friedensville mine, Friedensville, Pa.; development of North Friends mine, Tenn.; development of Ivanhoe mine, Wythe County, Va.; development and construction, Van Stone mine, Wash.; mine and plant expansion, Pend Oreille mine, Wash.; and mine development and plant construction, Calumet and Hecla mine, Shullsburg, Wis.

MILLING

Milling (*30*), the physical process by which the zinc minerals are concentrated, consists of crushing and grinding to liberate the valuable mineral particles, followed by separation of the zinc mineral particles by gravity, flotation, or magnetic methods or some combination of them.

The particular combination of ore-dressing equipment used varies with the quantities and kinds of ore minerals present, their values, the character of the gangue minerals, and the mineral particle sizes. Simple ores, such as coarsely disseminated zinc or zinc-lead minerals occurring with a low specific gravity gangue, are readily treated in heavy-medium cones, jigs, and tables after being crushed and rolled in closed circuit with vibrating or trommel screens and classifiers to give properly sized feed. Collective or differential flotation of the slime products or of a reground middling product completes the flowsheet. Ores of this kind are common in the mines of the Mississippi Valley and eastern United States. In some instances the ores are concentrated wholly by flotation, but the Sussex County, N. J., zinc deposits are concentrated magnetically.

The more complex sulfide ores consist of disseminated mixtures of fine-grained lead and zinc sulfides, usually accompanied by pyrite, some copper sulfides, and some gold and silver in a quartz or quartz-calcite gangue. Such ores may be complicated further by partial oxidation of the sulfides and by high-specific-gravity gangue minerals, such as barite, siderite, rhodochrosite, or fluorite. The usual procedure on such an ore is to crush and grind in closed circuit with classifying equipment to a mesh at which the ore minerals are preponderantly separated from the gangue minerals. If the ore minerals are interlocked, it is usually good practice to make a bulk sulfide concentrate first rather than grind all gangue and ore minerals to a mesh at which selective flotation can take place; the bulk concentrate is then treated by regrinding and selective flotation.

The low capital and operating costs of the sink-float method promise to extend its field of use for pretreatment of certain ores to reject rock of noncommercial grade resulting from the cheaper, nonselective mining methods.

Zinc ores are concentrated and reduced with considerable loss of metal content. Present ore-dressing practice results in recoveries of about 80 to 92 percent of the sulfide zinc and from only a few percent to perhaps 85 percent of the oxidized zinc under very favorable circumstances. Recent Bureau of Mines investigations on improvement of zinc recoveries in ore dressing include hundreds of flotation tests to evaluate the efficiency of different reagent combinations on zinc and lead sulfide slimes and basic research into fine grinding and the physical chemistry of the resulting slimed minerals. In treating oxidized zinc and lead-zinc ores various degrees of success have been achieved with flotation and leaching methods. The very wide divergence in ore characteristics requires diversified techniques. Among the most successful in terms of metallurgical recovery are the leaching methods. One of these, termed the caustic leach-electrolytic process (*3*) is applicable to considerable tonnages of oxidized zinc-lead ores in Nevada, Utah, Arizona, and California that are not amenable to standard concentration processes and are uneconomic when shipped to standard smelters. By this process zinc is extracted from the ores with sodium hydroxide solution, after which the resulting electrolyte is purified with zinc dust and lime and the zinc electrodeposited. The caustic consumed in dissolving zinc is regenerated in electrolysis, while that consumed in dissolving carbonate and silicate gangue minerals is easily regenerated by causticizing with lime.

The process is also applicable to mixed sulfide ores of zinc and lead, which when roasted can be treated as satisfactorily as oxidized ores. Current research is based on precipitation of the zinc by chemical means. Leaching with sulfur dioxide under acidmaking conditions has also been very effective on certain complex oxidized lead-zinc ores from the southwestern United States.

Considerable attention in Italy and France has been given to the flotation of oxidized zinc ores with fatty amines (*27*), and 2 small mills in Italy and 1 in France are treating over 400 tons of oxidized zinc ores daily. The process has been tested at the Touissit and Zellidja mines of French Morocco since early 1953. If the method proves successful at the Toussit and Zellidja mines the output of Tunisia, Algeria, and Morocco may be expected to increase further as greater recovery will be made at existing plants and many oxidized zinc-lead ore bodies now deemed uneconomic will become profitable.

SMELTING (REDUCTION PLANTS)

Zinc smelting (*20*), in common usage, is the term applied to all treatment processes by which zinc ores or concentrates are reduced to refined metal. Actually zinc is either recovered from its ores and concentrates by distillation in some one of several types of retorts or furnaces or by electrolysis.

Sulfide zinc concentrates are roasted to eliminate sulfur, and in the process the zinc is converted to the oxide and minor quantities of zinc sulfate. The roasted concentrate may either be leached to recover zinc solution for electrodeposition of the metal or combined with coke breeze, anthracite, or other solid fuel and retorted.

Distillation retort plants are classified as batch horizontal retorts, continuous vertical retorts fired by fuel, and continuous vertical retorts heated electrothermically. The zinc vapor and hot reducing carbon monoxide from the retorts pass into condensers of various types where the zinc is collected as liquid metal ready for casting into slab form. One large producer refines a portion of its production by further fractional distillation and rectification, but other distillation plants depend largely upon high-grade feed to retorts or liquation of the condensed zinc to secure the grades of slab zinc offered for sale. Several retort smelters produce zinc oxide and zinc dust as well as slab zinc.

The electrolytic zinc plants as noted above leach the roasted zinc concentrates to recover zinc solutions, which are purified to remove contained copper, cadmium, cobalt, iron, etc. The purified pregnant zinc solution is piped to the tankhouses, where the zinc is electrodeposited upon cathodes of high-grade aluminum. At intervals the cathodes are lifted from the tanks and stripped of the zinc coatings, which are then charged into a melting furnace to be melted for casting into slab form. The electrolysis of the solution regenerates sulfuric acid, which is used in another cycle of leaching.

The residues from distillation and leaching are shipped to a lead smelter for further processing when they contain economic quantities of lead, gold, or silver or processed in a Waelz kiln when economic quantities of zinc with or without lead remain.

The slags from lead blast furnaces often contain economic quantities of zinc and lead, which may be recovered as the oxides in slag-fuming plants. The lead slag from the Herculaneum (Mo.) smelter is electrothermically smelted for direct production of slab zinc. The oxides from both the Waelz kiln and slag-fuming plant are commonly deleaded and the resultant impure zinc oxide reduced to metal in an electrolytic or retort plant.

As shown in table 9 domestic smelters produce slab zinc in several grades, the degree of refining depending upon the type of producing plant and current demand. The electrolytic plants produce Special High-Grade or High-Grade slab zinc but may degrade their product to meet various market requirements. The slab zinc produced by the horizontal retort plants is largely Prime Western grade, although smaller quantities of all other grades are produced. The vertical retort plants preponderantly produce Regular High Grade, but all other grades are produced as demand warrants.

Metallurgical recoveries at zinc-reduction plants range from less than 85 percent in some instances to as high as 96 to 97.5 percent at others, the range in recovery being governed by the nature of the smelter feed, the treatment process, and the economics of recovery. The new Sterling process furnace (*13*) for electrothermic reduction of zinc ores is of much interest, because of its potentiality in treating high-iron-content zinc ores. The utilization of the blast furnace in reducing zinc ores at the Avonmouth plant of Imperial Smelting Corp. Ltd. also holds promise in treating complex zinc concentrates. Recent industry-modernization programs have included many improvements in roasting, among which are the use of Dwight-Lloyd sintering equipment to replace roasting hearths at the Blackwell (Okla.) smelter (*19*), introduction of a suspension roasting process at Trail, B. C. (*22*), and experimental use of the fluosolids roasting process at Amarillo, Tex.

The Bureau of Mines is doing research on secondary recovery in the electrorefining of zinc from anodes of zinc dross, an electrochemical process for recovery of zinc in sal skimmings, and the mechanical separation of zinc from dross and impure zinc condenser metal utilizing centrifugation.

RESERVES

Zinc-ore reserves (*8*) are widely distributed; important deposits occur in the United States, Canada, Mexico, Argentina, Bolivia, Peru, Chile, and Greenland in the Western Hemisphere and in many countries of Europe, Africa, and Asia, as well as in Australia. Zinc reserves in the measured and indicated classification are estimated to total over 70 million short tons of metal. Of this amount about 35 percent occurs in North America, 8 percent in South America, 30 percent in Europe, 15 percent in Australia, and about 8 percent in Africa and Asia. Partial information at hand suggests that the total zinc in measured, indicated, and inferred ore reserves approximates 200 to 300 million tons In addition there are doubtless very large quantities of metal in potential resources that are undiscovered or currently uneconomic.

TABLE 11.—*Estimated zinc reserves of the United States as of January 1950 (8)*

[Short tons of metallic zinc]

	Measured and indicated [1]		Inferred [2]		Total [2]	
	Gross content in ground	Recoverable content [3]	Gross content in ground	Recoverable content [3]	Gross content in ground	Recoverable content [3]
(1) Zinc in deposits that could be worked under technologic conditions similar to those in 1950	8,480,000	6,530,000	12,700,000	9,800,000	21,200,000	16,300,000
(2) Zinc in deposits workable under possible future technologic or industrial conditions	3,440,000	-----------	400,000	-----------	3,800,000	-----------

[1] This includes individual estimates of measured and indicated ore in some properties where such ore is known, but for which the tonnage figures are unavailable.

[2] Figures rounded.

[3] Milling and smelting losses are considered to be roughly 23 percent.

North America.—The United States, Canada, and Mexico have an estimated third of the world's developed reserves of zinc. The unwillingness of many mining firms precludes inclusion of a statement of reserves for the United States and Mexico on a current basis. An estimate of total zinc reserves in the United States as of January 1950 is given in table 11; it shows measured and indicated reserves of 8.48 million tons of zinc content. Developed reserves in Canada are estimated to contain 14 million tons of zinc. A recent discovery and partial development (1952–54) in Greenland suggests a considerable ore potential there.

Inferred ore in North America is believed to contain 65 to 100 million tons of additional zinc.

South America.—The combined reserves of developed ore in South America are estimated at 6 million tons of zinc or 8 percent of the world supply. Peru and Argentina have the largest reserves; those of Bolivia and Chile are somewhat smaller. Lack of development in such mineralized countries as Chile, Peru, Bolivia, Brazil, and Argentina suggests that very large reserves might well be developed.

Europe.—Europe produced about 800,000 short tons of newly-mined zinc in 1953 or roughly 28 percent of the world total. Measured and indicated reserves are estimated to contain 22 million tons of zinc, or about 30 percent of the world total. Of this quantity about 11 million tons is in eastern Europe and 11 million tons in western Europe.

Africa.—Africa has important zinc deposits in Algeria, Belgian Congo, French Morocco, Northern Rhodesia, South-West Africa, and Tunisia. Reserves contain about 4 million tons of zinc. The operation of the Société des Mines de Zellidja and Société Nord Africaine du Plomb at and near Oudjda, Morocco, extend into Algeria. Active exploration has shown large reserves capable of supporting a production of 120,000 tons of zinc concentrates (55 percent zinc) beginning in 1954. The Belgian Congo has important developed zinc reserves, a large part (450,000 to 600,000 tons of zinc) being in the copper slags at Lumbumbashi where the Prince Leopold ores are smelted. South-West Africa and Northern Rhodesia each have an important zinc-mining industry, with reserves that reportedly are large.

Asia.—Developed zinc reserves in Asia are estimated to be 4.5 million to 5 million tons, about half in Burma and the remainder divided among Japan, China, Korea, Viet Nam, and the Philippines.

Australia.—Australia has developed reserves containing about 11 million tons of zinc, or roughly 15 percent of the world total. The important deposits are at Broken Hill, and Captain's Flat, New South Wales; Mount Isa, Queensland; and the Read-Roseberry mine, Tasmania.

CONSUMPTION AND USES OF ZINC IN THE UNITED STATES

The consumption of slab zinc, of zinc in ore and concentrates for the direct manufacture of pigments and salts, and of zinc in scrap metal to make zinc alloys, zinc dust, chemicals, and pigments during the 29-year period 1925–53 averaged about 940,000 tons annually and in the last 5 years of that time period averaged 1,240,000 tons. An itemization of each element of annual supply and distribution for that 29-year period is given in table 12. The consumption of slab zinc, by industries and classes of product, is given in table 13.

The largest single use of zinc—galvanizing—employs about 40 percent of the total slab

TABLE 12.—*Salient statistics of the zinc industry in the United States, 1925–53* [1]

[All figures in thousand short tons]

Year	Supply				Distribution				
	Total	Recoverable mine production	Recoverable secondary	Imports [2]	Total	Slab zinc consumed	Consumed as ore in pigments, etc.	Exports [2]	Secondary [3]
1925	879	711	156	12	942	579	111	157	95
1926	956	775	168	13	996	622	130	140	104
1927	897	719	169	9	905	583	122	96	104
1928	878	695	181	2	898	626	124	38	110
1929	914	724	177	13	907	634	138	24	111
1930	746	595	128	23	646	451	105	11	79
1931	513	410	102	1	518	370	75	5	68
1932	362	285	75	2	380	259	55	11	55
1933	509	384	121	4	502	350	72	7	73
1934	550	439	97	14	515	360	76	13	66
1935	658	518	129	11	640	473	86	8	73
1936	749	576	161	12	778	582	96	7	93
1937	839	626	166	47	831	610	112	8	101
1938	659	517	119	23	574	421	69	8	76
1939	829	584	183	62	871	626	85	14	146
1940	1, 057	665	222	170	1, 075	719	96	90	170
1941	1, 314	749	284	281	1, 272	827	135	97	213
1942	1, 452	768	331	353	1, 250	728	115	140	267
1943	1, 631	744	368	519	1, 350	817	115	106	312
1944	1, 493	719	345	429	1, 346	889	142	26	289
1945	1, 404	614	360	430	1, 297	852	131	14	300
1946	1, 215	575	301	339	1, 242	801	134	62	245
1947	1, 279	638	311	330	1, 293	786	146	120	241
1948	1, 283	630	325	328	1, 280	818	133	77	252
1949	1, 167	593	238	336	1, 046	712	88	71	175
1950	1, 345	623	326	396	1, 375	967	134	25	249
1951	1, 347	681	314	352	1, 376	934	134	50	258
1952	1, 478	666	310	502	1, 278	853	109	66	250
1953	1, 520	547	295	678	1, 368	986	118	26	238
Average	1, 032	613	223	196	991	663	110	52	166

[1] Sources: National Security Resources Board, Materials Survey—Zinc, 1951, and various Bureau of Mines data.

[2] Imports and exports are factored as follows: 100 percent of the metal is added to 85 percent of the zinc content of ores and concentrates.

[3] Consumed in dust, alloys, chemicals, and pigments.

zinc consumed. Galvanizing is recognized as the most economical means of protecting steel products from atmospheric corrosion. Zinc coating exposed to normal atmospheres forms an insoluble, adhering, impervious layer of zinc carbonate that resists further attack. The usefulness of this as a protective coating on steel is enhanced further by the fact that zinc is electropositive with respect to iron; hence, when a galvanized article is subjected to corrosion the zinc is sacrificed in favor of the iron. Zinc-coated steel products include roofing and siding sheets, wire and wire products for outdoor exposure, articles fabricated from sheet steel (such as range boilers, pails, cans, and tanks), hardware for outdoor use, pipe and conduit, and exposed structural steel.

In galvanizing the more widespread introduction of continuous hot-dip galvanizing (*10*, *21*), with a new and superior zinc-alloy coating containing a small quantity of aluminum, is expected to cut costs of production and yield a better product having a considerable cost advantage over aluminum sheet and aluminum coatings.

The second largest class of slab-zinc usage is in die casting—the art of producing castings in quantity by forcing molten alloy into steel dies or molds. The equipment consists of a melting pot, an injecting device, and the die, with a device for opening and closing the die during operation. The zinc alloy is injected into the die at temperatures up to 900° F. and at pressures up to 2,500 pounds per square inch. Die casting requires a good structural metal that can be melted and used at relatively low temperatures, has little shrinkage, is dimensionally stable, and solidifies to a smooth finish without gas inclusions. Because of their excellent properties die-cast alloys are used for parts as small as zipper elements or as large as automobile radiator grilles. The automotive

TABLE 13.—*Consumption of slab zinc in the United States, 1944–48 (average) and 1949–53, by industries* [1]

[Short tons

Industry and product	1944–48 (average)	1949	1950	1951	1952	1953
Galvanizing: [2]						
Sheet and strip	120, 818	146, 923	188, 406	144, 329	145, 875	164, 601
Wire and wire rope	46, 746	39, 231	47, 317	51, 792	48, 645	44, 100
Tubes and pipe	67, 041	78, 030	91, 877	79, 221	82, 043	88, 428
Fittings	11, 845	11, 487	15, 948	21, 186	10, 366	10, 330
Other	94, 595	75, 209	98, 138	103, 751	90, 759	99, 529
Total galvanizing	341, 045	350, 880	441, 686	400, 279	377, 688	406, 988
Brass products:						
Sheet, strip, and plate	112, 185	43, 157	68, 737	67, 815	71, 706	94, 826
Rod and wire	51, 677	23, 651	43, 413	46, 056	49, 831	47, 312
Tube	19, 957	12, 816	17, 385	15, 927	17, 057	18, 136
Castings and billets	8, 361	2, 620	4, 170	7, 098	7, 262	8, 145
Copper-base ingots	8, 458	2, 701	4, 081	5, 743	8, 223	7, 659
Other copper-base products	1, 741	589	1, 587	653	1, 529	2, 104
Total brass products	202, 379	85, 534	139, 373	143, 292	155, 608	178, 182
Zinc-base alloy:						
Die castings	169, 123	199, 665	285, 022	282, 812	225, 877	297, 280
Alloy dies and rod	5, 763	2, 024	2, 929	11, 135	9, 235	7, 140
Slush and sand castings	447	492	1, 576	2, 487	1, 577	3, 025
Total zinc-base alloy	175, 333	202, 181	289, 527	296, 434	236, 689	307, 445
Rolled zinc	82, 772	55, 200	68, 444	64, 085	51, 318	54, 649
Zinc oxide	18, 303	10, 292	18, 187	18, 223	17, 205	20, 675
Other uses:						
Wet batteries	1, 686	1, 359	1, 527	1, 749	1, 396	1, 417
Desilverizing lead	2, 254	2, 448	2, 947	2, 186	2, 370	2, 425
Light-metal alloys	1, 158	1, 060	1, 356	3, 132	3, 266	5, 939
Other [3]	4, 325	2, 887	4, 087	4, 591	7, 243	8, 207
Total other uses	9, 423	7, 754	9, 917	11, 658	14, 275	17, 988
Total consumption [4]	829, 255	711, 841	967, 134	933, 971	852, 783	985, 927

[1] Excludes some small consumers.

[2] Includes zinc used in electrogalvanizing and electroplating, but excludes sherardizing.

[3] Includes zinc used in making zinc dust, bronze powder, alloys, chemicals, castings, and miscellaneous uses not elsewhere mentioned.

[4] Includes 2,394 tons of remelt zinc in 1949, 3,035 tons in 1950, 4,505 ton in 1951, 4,144 tons in 1952, and 3,710 tonsi n 1953.

industry uses a great number of zinc die castings, including such items as carburetors, bodies for fuel pumps, parts for windshield wipers, speedometer frames, horns, heaters, parts for hydraulic brakes, door handles, instrument panels, and many others. Die castings are also used extensively in electrical appliances, business and other light machines, tools, building hardware, toys, and novelties.

In the consumption of zinc it is considered likely that die-casting uses will increase further, for although much is heard of the inroads of aluminum in this field zinc has inherent mechanical, physical, and chemical advantages over aluminum that make it the preferred metal for many die casting uses when zinc is selling at about two-thirds of the price of aluminum. American Die Casting Institute statistics on die-cast alloy shipments for various end uses indicate that there has been little substitution of aluminum but rather that zinc has maintained its position in most instances where the metals might be considered competitive. It is believed that the present strong supply position of zinc, the favorable price relationship, and the inherent advantages of the common zinc die-casting alloys will result in continued increases in demand.

Large quantities of slab zinc are consumed for brassmaking, especially in wartime. During World War II consumption of zinc for brass, principally for cartridge and shell cases and other military articles, composed nearly 50 percent of the total slab zinc consumed. Since 1946, however, zinc consumed for brass has averaged less than 15 percent of the total slab zinc used. Zinc recovered from copper-base scrap (averaging about 175,000 tons annually) is consumed in making brass and bronze.

Developments in weapons and ammunition have lessened the probable requirements for brass in future wars, but civilian and military requirements remain large.

For use in sheet and rolled forms zinc is given the desired properties through alloying. Such uses include dry cells, jar caps, weather stripping, photoengraving plates, and roofing, as well as heavy plates for cathodic protection of steam boilers, ship hulls, and pipelines (*11*). Rolled zinc in building construction gives long service at reasonable cost and does not stain painted walls.

Zinc pigments and chemicals, most important of which are zinc oxide, leaded zinc oxide, lithopone, zinc chloride, and zinc sulfate, have literally hundreds of uses. Although the combined average annual shipments (zinc content) of these compounds is about 200,000 tons, only a small proportion (some 15,000 to 20,000 tons of zinc oxide) is produced from slab zinc, and the remainder of the compounds is produced directly from ore and secondary materials. The pigments and salts find many uses. The major uses for zinc oxide are in rubber, paints, ceramics, cosmetics and pharmaceuticals, textiles, and floor coverings. Leaded zinc oxide is used in paints. Lithopone is used in paints, varnish, lacquers, coated fabrics, textiles, floor coverings, rubber, paper filler, and printing ink. Zinc chloride goes into wood-preserving solutions, dry battery cells, refining oil, and soldering flux. Zinc sulfate is used in rayon, fertilizers, glue, textile dyes, electrogalvanizing solutions, insecticides, fungicides, chemicals, and soap.

About 25,000 tons of zinc in the form of dust is used annually as a reducing agent in the synthesis of dye intermediates, the manufacture of sodium and zinc hydrosulfite, sherardizing iron and steel products, metallic paints, and certain other uses.

SUBSTITUTES

There are few suitable substitutes for zinc in its major uses. For galvanizing iron and steel products there is no adequate substitute. Ceramic and plastic coatings are now employed in a narrow field that may gain in future importance. Cadmium is used for electroplating, but its cost and the quantity available limit it as an extensive substitute for zinc in galvanizing. Other metals, particularly aluminum, may be substituted for zinc coating on steel in some instances, but the cost to date has inhibited extensive use. Sheet aluminum is a strong competitor of galvanized steel, although it has poorer structural qualities and its cost is higher.

There is increasing consumption of wrought aluminum and magnesium, but the data available indicate that such consumption is largely in new uses and not at the expense of brass.

Aluminum competes with zinc in certain die-casting applications, and the competition between the two metals is influenced to some extent by metal price considerations. For the most part selection of an alloy depends upon the properties desired in the finished casting. In addition, where the use of electrodeposited coating is desired, zinc has a great advantage because of the ease with which finishes can be applied over die castings of zinc alloy.

Relatively few substitutes for zinc are available in chemical applications. Aluminum and magnesium could replace zinc to some extent as reducing agents in chemical reactions. In the paint industry lead and titanium pigments can be used instead of zinc pigments in many instances. Titanium pigments have replaced lithopone to a marked extent, but they supplement rather than compete with zinc oxide in paint in most formulations. Titanium dioxide has made inroads on the use of zinc oxide in porcelain enamels, but in opacifiers zirconium compounds have replaced zinc oxide only in part.

In general, there is little necessity to seek substitutes for zinc, as ample quantities of zinc in the various grades are available at relatively low prices in world markets.

FOREIGN TRADE[3]

The foreign trade of the United States, as it relates to zinc, consists chiefly of imports of zinc concentrate and slab zinc and exportation of relatively small quantities of slab zinc.

IMPORTS

Imports were a neglible proportion of the total United States supply of zinc from 1925 to 1936 and consisted largely of ores and concentrates imported under bond for subsequent export. In 1937 imports increased markedly, initiating a period of greater dependence upon foreign supplies. Beginning in 1940 and continuing to the present, the greatly expanded demands for military and industrial purposes have been met in large part by greater volumes of imports. For the 14 years 1940–53 imports have averaged 418,000 tons a year or about 29 percent of the total supply.

Ores and concentrates consistently have been the principal category of imports, averaging about four-fifths of total zinc imports.

Canada and Mexico were the principal sources

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[3] Figures on imports and exports compiled by Mae B. Price and Elsie D. Page, Division of Foreign Activities, Bureau of Mines, from records of the U. S. Department of Commerce.

of United States zinc imports throughout 1925–53 and during the last 14 years supplied about three-fourths of all imports. Mexico has been the chief source of ores and concentrates, supplying nearly half of the total; approximately 20 percent came from Canada. Canada supplied about 60 percent of the imported slab zinc and Mexico about one-third during the 29-year period.

In 1937 Peru began to export zinc ores and concentrates to the United States and, especially since 1941, has been a consistent source, averaging slightly less than 10 percent of total imports. During World War II considerable quantities of zinc in ores and concentrates came from Australia and Argentina. Chile, Belgian Congo, and Bolivia supplied smaller tonnages during the war years.

Table 14 shows imports of ores and concentrates (zinc content) and slab zinc by country of origin for 1937–53, and table 12 gives the total imports factored for smelter losses.

TABLE 14.—*Imports of zinc and zinc-bearing materials into the United States by countries of origin, 1937–53* [1] [2]

[Short tons]

Year	Zinc ores and concentrates (zinc content)							Blocks, pigs, slab and zinc scrap			
	Canada	Mexico	Argentina	Peru	Australia	Others	Total	Canada	Mexico	Others	Total
1937	84	338	--------	8, 373	--------	17	8, 812	6, 695	8, 375	24, 058	39, 128
1938	--------	7, 253	--------	11, 330	--------	--------	18, 583	2, 345	3, 746	1, 395	7, 486
1939	1, 613	23, 221	1, 544	9, 722	--------	--------	36, 100	6, 908	16, 554	7, 961	31, 423
1940	33, 393	93, 789	6, 723	17, 285	1, 749	27, 381	180, 320	6, 938	8, 948	582	16, 468
1941	49, 764	148, 201	20, 579	28, 833	8, 933	32, 903	289, 213	7, 236	25, 621	1, 697	34, 554
1942	78, 104	160, 959	23, 395	27, 506	32, 212	46, 232	368, 408	1, 749	34, 603	--------	36, 352
1943	110, 310	166, 168	53, 452	21, 901	122, 305	64, 958	539, 094	8, 570	45, 344	2, 241	56, 155
1944	112, 299	177, 625	22, 253	34, 889	42, 216	24, 414	413, 696	18, 099	39, 703	5, 824	63, 626
1945	90, 200	177, 003	--------	35, 415	15, 377	63, 724	381, 719	46, 594	36, 105	14, 417	97, 116
1946	57, 298	127, 685	8, 295	48, 791	3, 780	26, 207	272, 056	85, 198	15, 777	3, 772	104, 747
1947	42, 230	163, 726	--------	49, 952	864	41, 187	297, 959	54, 954	332	17, 026	72, 312
1948	55, 371	142, 134	77	22, 475	495	43, 651	264, 203	77, 660	5, 737	9, 835	93, 232
1949	61, 314	144, 101	--------	14, 901	4, 956	15, 907	241, 179	109, 708	14, 191	3, 026	126, 925
1950	77, 525	155, 283	8	16, 946	2, 366	26, 445	278, 573	108, 937	26, 293	20, 744	155, 974
1951	96, 568	143, 769	5, 546	29, 136	2, 825	24, 749	302, 593	85, 066	760	2, 217	88, 043
1952	149, 130	200, 647	603	44, 337	2, 398	52, 521	449, 636	69, 775	18, 686	27, 244	115, 705
1953	165, 737	168, 937	--------	84, 365	10, 820	83, 524	513, 383	107, 925	33, 878	92, 773	234, 576

[1] All data compiled from records of the U. S. Department of Commerce.

[2] Data include zinc imported for immediate consumption plus material entering the country under bond.

EXPORTS

Zinc exports have reached substantial proportions only twice since 1925, once from 1925–1927 and again from 1940–49, except for 1944 and 1945. During the first period the annual average was 131,000 tons, composed about equally of ores and concentrates and slab and sheet zinc. In 1940–49 exports averaged 80,000 tons a year and were predominantly slab and sheet. The United Kingdom and India received the largest quantities of metal during this period. Exports totaled 66,000 and 26,000 tons, respectively, in 1952 and 1953. Exports of many manufactured materials contain an unmeasured but substantial amount of zinc.

TARIFF

Zinc has been subject to an import duty for over 100 years (*32, 33*). The highest rates were in effect from 1930–39 when the duties on slab zinc and ores and concentrates (zinc content) were 1.75 and 1.50 cents per pound, respectively. In 1939 these tariffs were reduced to 1.40 and 1.20 cents per pound, respectively. In 1943 a reciprocal trade agreement between Mexico and the United States reduced the duty on slab zinc and zinc in ore to 50 percent of the amount set by the act of 1930, or 0.875 cent per pound on slab and 0.750 cent per pound on zinc in ore. This agreement was terminated December 31, 1950.

Following agreements at the Geneva Trade Conference of 1947 import duties were established, effective January 1948, as follows: 0.750 cent per pound for ore and concentrates (zinc content) and 0.875 cent per pound for blocks, pigs, or slabs. These rates applied equally to all nations with whom the United States has "Most-Favored Nation" agreements.

On June 6, 1951, import duties on slab and ore were reduced to 0.7 and 0.6 cent per pound, respectively. Owing to the increased United

TABLE 15.—*Zinc articles: United States rates of duty imposed under Tariff Act of 1930, in specified years 1930–54*

(United States Tariff Commission)

Item	Tariff rate in—				
	1930	1945	1948	1951	1954
	Cents per pound; percent ad valorem				
Par. 77:					
Zinc oxide and leaded zinc oxides containing not more than 25 per centum of lead:					
In any form of dry powder	1¾	1⅒[1]	⁶⁄₁₀[2]	⁶⁄₁₀	⁶⁄₁₀.
Ground in or mixed with oil or water.	2¼	1½[1]	1[2]	1	1.
Lithopone, and other combinations or mixtures of zinc sulfide and barium sulfate:					
Containing by weight less than 30 per centum of zinc sulfide.	1¾	1½[3]	⅞[2]	⅞	⅞.
Containing by weight 30 per centum or more of zinc sulfide.	1¾+15 percent.	1¾+15 percent.	⅞+7½ percent.[2]	⅞+7½ percent.	⅞+7½ percent.
	Cents per pound of zinc content				
Par. 393: Zinc-bearing ores of all kinds, except pyrites containing not more than 3 percent zinc.	1½	¾[1, 4]	¾[2]	⁶⁄₁₀[5]	⁶⁄₁₀.[6]
	Cents per pound				
Par. 394:					
Zinc blocks, pigs, or slabs	1¾	⅞[1, 4]	⅞[2]	⁷⁄₁₀[5]	⁷⁄₁₀.[6]
Old and worn-out zinc, fit only to be remanufactured, zinc dross, and zinc skimmings.	1½	¾[1, 7]	¾[2, 7]	¾[7]	¾.[7]
Zinc dust[8]	1¾	⅞[1, 4]	⅞[2]	⁷⁄₁₀[5]	⁷⁄₁₀.
Zinc sheets	2	1[1]	1[2]	1	1.
Zinc sheets coated or plated with nickel or other metal (except gold, silver, or platinum), or solutions.	2¼	1⅛[1]	1⅛[2]	1⅛	1⅛.

States need for foreign zinc, import duties were suspended on February 12, 1952, until March 31, 1953, or until the end of the emergency, whichever came first. The act stipulated that the rates of June 6, 1951, were to be reimposed if the average market price of zinc fell to 18 cents a pound for a calendar month. The Tariff Commission notified the President on July 3, 1952, that the average price of slab zinc for June was below 18 cents a pound, and on July 23 the President signed the order ending the duty suspension. In effect the general inflation in which the price of zinc has participated resulted in an additional lowering

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[1] Trade agreement with Mexico, effective Jan. 30, 1943 through Dec. 31, 1950.

[2] General Agreement on Tariffs and Trade (GATT) (Geneva), effective Jan. 1, 1948.

[3] Trade agreement with the Netherlands, effective Feb. 1, 1936, through Dec. 31, 1947.

[4] Rate previously reduced in the trade agreement with Canada, effective Jan. 1, 1939, through Dec. 31, 1947, to 1⅕ cents per pound of zinc content on zinc-bearing ores, and to 1⅖ cents per pound on zinc blocks pigs, and slabs, and on zinc dust.

[5] GATT (Torquay), effective June 6, 1951.

[6] Duty suspended from Feb. 12, 1952, to July 23, 1952, inclusive (Public Law 258, 82d Congress).

[7] Duty on metal scrap suspended for practically the entire period from Mar. 14, 1942, to June 30, 1953, inclusive (Public Law 497, 77th Cong.; Public Laws 384 and 613, 80th Cong.; Public Law 869, 81st Cong.; and Public Laws 66 and 535, 82d Cong.).

[8] Since enactment of Public Law 497 (77th Cong.), effective Mar. 14, 1942 and subsequent amendments (see note 7 above), providing for temporary suspension of duties on metal scrap, quantities of zinc dust have been entered free of duty under this law. No information is available as to the distinction between the zinc dust which has entered free of duty and that which has entered as dutiable.

of the tariff if considered on an ad valorem basis.

The high level of imports established in 1952–53, together with the depressed state of the domestic zinc- (and lead-) mining industry stimulated much study of various protective tariff proposals and other plans to aid the industry. A committee representing a large segment of the domestic lead- and zinc-mining industry petitioned the Tariff Commission for an investigation under the "escape-clause" provisions of the Trade Agreements Extension Act of 1951, which permits the President upon recommendation of the Tariff Commission, to increase duties by up to 50 percent of the rates prevailing on January 1, 1945. Congressional committees also adopted resolutions instructing the Tariff Commission to investigate imports of lead and zinc and their effects on domestic output. The report of this investigation (*32*) was factual and contained no recommendations. It showed that the existing tariff structure restricted imports but slightly and that during the past decade a substantial part of all zinc imports were exempted from duty.

The report on the "escape-clause" provisions of the Trade Agreements Extension Act of 1951 (*33*) recommended that import duties on most lead and zinc materials be increased 50 percent above the rates existing on January 1, 1945. The President did not accept the recommendations of the Tariff Commission (*36*), instead, he outlined an expanded stockpiling program for strengthening the lead and zinc industry as an integral part of the Nation's defense-mobilization base.

MARKETING AND PRICES

Slab zinc is sold on the standard specifications adopted by the American Society for Testing Materials and by industry generally, which provides for six standard grades.

Market prices of zinc are usually quoted in cents per pound, f. o. b. East St. Louis, although relatively little zinc is actually delivered at that point. Zinc is also quoted on a New York basis. The London price is the principal basis for foreign transactions. Sales agreements usually stipulate market price at time of sale or shipment, as published in some specified trade publications in the United States, usually the Engineering and Mining Journal. Special High-Grade and High-Grade zinc are sold on a delivered basis at a premium of 1.50 and 1.35 cents, respectively, above the price of Prime Western, East St. Louis. Intermediate, Brass Special and Select grades are sold at smaller premiums above the Prime Western price, f. o. b. East St. Louis.

The selling prices of zinc pigments and salts and zinc dust do not follow the minor fluctuations in the price of slab zinc but are affected by the major price movements. In the raw-materials group (including zinc concentrates and byproducts) negotiations by individual contracts and spot sales are the practice, except for concentrates in the Joplin, Mo., area, where a market price for concentrates produced from ores mined in the district is published weekly.

Zinc prices have fluctuated widely during 1925–53. The average price for Prime Western grade, East St. Louis, was 6.76 cents per pound in 1925–29 and 5.10 cents in 1935–39, with an intervening low of 2.88 cents in 1932. The upswing in prices created by the demands of World War II was halted in October 1941 by the Government, which set the controlled price at 7.25 cents per pound. In October 1942 the price was established at 8.25 cents per pound, at which level it remained until controls were removed in the fall of 1946. Prices increased substantially in the postwar period, reaching a high of 17.50 cents in December 1948 and averaging 12.53 cents for 1947–50.

Following the outbreak of war in Korea in June 1950 zinc prices advanced steadily, attaining a peak of 17.50 cents in September. Ceiling prices were established for zinc on January 26, 1951, at the highest price (for each seller) at which sales were made between December 19, 1950, and January 25, 1951. Thus a number of different ceiling prices were maintained. The bulk of zinc sales, however, were at 17.50 cents per pound, Prime Western grade, East St. Louis. This price remained in effect until October 2, 1951, when the Office of Price Stabilization permitted a rise of 2 cents per pound. Increased supplies and lower consumption brought about marked reductions in zinc prices in the summer of 1952. On June 2 the price dropped to 17.50 cents and by August 6 was down to 13.50 cents. The price advanced slightly thereafter but as of October 21 was again down to 13.50 cents. Subsequently on October 23 it declined to 12.50 cents, at which point it remained throughout the remainder of 1952. Excess supplies brought about further reductions in price in 1953; the market opened at 13.00 cents a pound for Prime Western f. o. b. E. St. Louis but thereafter declined until September 11, when the price was 10.00 cents a pound. This price remained in effect to the end of the year. Thus in the 16-month period from June 2, 1952, to September 11, 1953, the price of zinc dropped from 19.5 to 10 cents a pound, the greatest drop in so short a period since 1916. Prime Western grade zinc dropped further in 1954, reaching a low of 9.25 cents on February 15, but thereafter the price increased without interruption to 11.50 cents on September 7, at which it remained for the rest of 1954.

SELF-SUFFICIENCY AND STRATEGIC CONSIDERATIONS IN THE UNITED STATES

The United States is not self-sufficient in zinc. Over the past 29 years about 59 percent of the Nation's supply has been met from domestic mine production, 22 percent from secondary recovery, and 19 percent by imports, but in the 4 years ended in 1953 mine production has been only 44 percent of the total supply, imports 34 percent, and secondary 22 percent.

The differences in supply and distribution figures given in table 12 (p. 994), however, show that supplies and hence imports were considerably over requirements during the World War II period and again in 1952–53 and thus caused the accumulation of large stocks. If stock accumulations and exports are discounted it may be seen that United States import requirements averaged about 238,000 tons annually to meet industrial needs in 1940–53. Acually imports (metal equivalent) averaged 388,000 tons a year during the past 14 years or 150,000 tons a year over consumption.

In 1952–53, imports of recoverable metal totaled 1,180,000 tons or 444,000 tons over the quantity needed to balance supply and requirements exclusive of exports (see table 12, p. 994). Moreover, imports of such record tonnages exerted a depressing effect on domestic mine production, so that this component of supply furnished a considerably smaller proportion of the total than in previous years. In retrospect, it would seem that for 1952–53 imports of about 230,000 tons a year would have been ample to fulfill domestic needs which could not have been supplied from domestic sources.

Thus, while the United States is not self-sufficient in zinc, it does not depend strategically on foreign sources to anywhere near the extent suggested by imports in recent years.

GOVERNMENT PROGRAMS

The Government regulated the zinc industry during World War II and again in 1950–52 to assure adequate supplies for all essential purposes. In World War II measures were taken to increase imports, particularly of ore and concentrates, to expand or at least maintain domestic mine production, expand smelter capacity, establish a stockpile, and control consumption and exports.

The first measures taken included the expansion of imports, partly by purchase agreements (for example, with the Government of Mexico) and partly by agreement with other Allies to divert to the United States exports from Canada and Australia, which normally went to Great Britain and Belgium. This program was a notable success. Imports of ore, which shortly before the war averaged about 9,000 tons a year, had by 1943 reached a high of 539,000 tons, and the annual average for the entire wartime period was over 400,000 tons.

Imports of metal, which in prewar years totaled about 19,000 tons, added an average of about 70,000 tons of metal annually to the total supply during the war years.

Various measures were adopted to increase or at least maintain domestic mine production. The most important of these was the Premium Price Plan, designed primarily to increase production from submarginal ores by paying premiums for production above a certain quota based on 1941 production. Through payment of these premiums, marginal and lower-grade ores were mined, and tailing piles previously considered worthless were reworked. In addition, new and idle mines were brought into production, and more extensive development work was initiated in some existing mines. Despite increasing costs and labor difficulties, mine production in the war years 1942–44 averaged 744,000 tons—about 25 percent above the 1936–38 average.

During the 5½ years the plan was in operation (February 1942–June 1947) 56 percent of all the zinc produced received premiums. In all, a premium of about 177 million dollars was paid for the 2,090,000 tons of premium-produced zinc. The weighted average price paid for all zinc during the period was 10.92 cents a pound, of which premium payments came to 2.36 cents a pound.

The plan for the Government to purchase zinc developed into a stockpiling program dating from the spring of 1941, when the Metals Reserve Company was authorized to purchase up to 100,000 short tons of zinc concentrates from Argentina and Australia. The requirements were increased 150,000 tons in June 1941. In May 1942 the Metals Reserve Company was authorized to purchase up to 600,000 tons of foreign zinc concentrates annually. Total stocks at the end of 1942 totaled 464,000 tons and a year later had reached 617,000 tons.

Early in 1944 further consideration was given to stockpile objectives. An overall total of 800,000 tons was approved, comprising both Government and industry-owned stocks of slab zinc and recoverable zinc in concentrates. By the end of the year the total objective had been exceeded, with total stocks at 810,000 tons. Government stocks approximated the goal of 550,000 tons. From the beginning of

1945 to the end of the war stocks were gradually reduced.

At the end of the war, with a heavy demand from industry for reconversion needs, Government stocks were made available to consumers unable to obtain normal requirements in the market, except for quantities transferred to the United States Department of the Treasury, Bureau of Federal Supply, under the Stockpiling Act of 1946. Thus the wartime stockpiling program of the Government, which was generally effective in meeting wartime needs, carried over in part into a peacetime stockpile.

Soon after the onset of the war in Europe it became evident that the demand for zinc by the United Kingdom and its Allies would exceed the resources of the United States. At first voluntary allocations by industry of available supplies of zinc were tried. Owing to the growing pressure of defense orders these voluntary measures soon became inadequate.

Control of exports had already been initiated by the Government early in 1941. In June of that year the Office of Production Management placed zinc under full priority control, with issuance of General Preference Order M–11. This order, with subsequent modifications and additions, controlled the allocation of zinc, including dust and scrap, for essential military and civilian uses, prohibited its use for certain purposes, established inventory controls, and provided for detailed reports to be filed by producers and users of zinc. It was administered after January 1942 by the War Production Board, successor to OPM.

Through these control measures enough zinc was provided to meet the most urgent military and civilian requirements during the war years. Controls were continued until the end of the war, although they were relaxed somewhat in September 1944, when supply appeared adequate. Full controls were reinstated in April 1945, mainly because of a shortage of the higher grades of zinc, and were continued until August 20, a few days after the end of the war with Japan.

The Korean War in 1950 and the prompt acceleration of the United States defense program were reflected in the introduction of Government controls over the zinc industry in the latter part of the year.

National Production Authority Order M–15 issued December 1, 1950, limited civilian consumers of zinc and zinc-metal products to an average quarterly consumption rate not to exceed 80 percent of their quarterly average, as established in the 6-month period ended June 30, 1950. The same order specified that inventories should not exceed a 45-day supply or a "practicable minimum working inventory," whichever was less. On June 15, 1951, Order M–15 was amended to limit quarterly consumption of Special High-Grade zinc to 70 percent of that used in the basing period, and allowable inventories of zinc and zinc-metal products were restricted to 30 days or the "practicable minimum working inventory," whichever was less.

On July 5 NPA issued a new order, M–9, which placed slab zinc under allocation effective August 1. Under this order no dealer or consumer might accept delivery of 20 short tons or more of slab zinc during any calendar month without an allocation authorization. Amendment 1 to this order, effective January 1, 1952, reduced allowable receipts without allocation authorization from 20 to 10 short tons in any calendar month. Several provisions of Order M–15 were incorporated in Order M–9 as amended March 7, 1952. All NPA restrictions on zinc were revoked on June 27, when it was apparent that zinc was not in short supply.

BUREAU OF MINES EXPLORATION PROGRAM

Under authorization of the Congress, the Bureau of Mines during World War II undertook a strategic minerals development program that included zinc. Over 2,200 zinc and zinc-lead properties were examined, and 20 million tons of ore in all were indicated at about 85 of the 140 deposits drilled or otherwise explored. Outstanding accomplishments of the program with respect to zinc were the ore discoveries in the Metaline district of northeastern Washington and near Galena, Ill. Subsequent private exploration greatly increased the zinc reserves as well as the zinc-ore output of the two areas.

GOVERNMENT EXPLORATION AND EXPANSION PROGRAMS UNDER THE DEFENSE ACT OF 1950

On December 4, 1950, the Defense Minerals Administration was established by the Secretary of the Interior under the provisions of the Defense Production Act of 1950. The function of DMA was to increase the supply of necessary minerals through exploration and expansion programs. In mid-1951 the exploration function was transferred to the Defense Minerals Exploration Administration, and soon thereafter the production expansion and procurement function was transferred to the Defense Materials Procurement Agency under the General Services Administration. DMA–DMEA by the close of 1953 had approved and entered into 175 mineral-exploration contracts for zinc and lead, obligating the Government for up to 50 percent of the cost, or a total of $7,297,000. In conformance with an order by the Office of Defense Mobilization no loans could be made on zinc and lead applications received after May 15, 1953. This order was amended, however, March 23, 1954, to again permit loans on approved exploration projects.

DMPA, in fulfilling its responsibility for production expansion, made purchase contracts (including subsidies where warranted), granted priority ratings for production machinery and equipment, and recommended production expansion loans, operating loans, and certificates of necessity for accelerated tax amortization. The expansion program was not confined to domestic producers, and a number of foreign contracts were negotiated to bring in additional supplies of zinc. In early 1953 the DMPA program was greatly reduced, and on August 14 of that year the remaining functions of DMPA —including completion of contracts in negotiation and the servicing of existing contracts— were transferred to the Emergency Procurement Service of General Services Administration. In total, purchase commitments at guaranteed prices were made with 9 domestic and 3 foreign producers for 124,526 tons of zinc; loans totaling $1,820,000 were certified for 9 domestic producers and 1 foreign producer, and 41 certificates of necessity were granted for accelerated tax amortization in plant expansions expected to result in 182,000 tons of additional zinc capacity per year.

OUTLOOK

World reserves of measured, indicated, and inferred zinc ores appear adequate to meet the foreseeable requirements of the next 40 years, and the potential of new discoveries in known mining districts and undeveloped areas of the world is believed to be much larger. World consumption will increase, requiring an annual mine output of about 3,000,000 tons of zinc by 1965 compared to about 2,500,000 tons at present.

United States reserves will permit continuation of present or somewhat greater domestic mine output for many years. It seems probable, however, that a larger part of the supply will be met by imports, for not only will requirements increase greatly, but the ratio of scrap generation to total consumption is declining. Government purchases of newly mined domestic zinc should stimulate domestic mine production to about 500,000 tons of recoverable metal in 1955. In the ensuing decade mine production can be expected to average 550,000 to 600,000 tons a year. Total distribution, which includes slab-zinc consumption, direct consumption of ore in pigments and chemicals, consumption of secondary metal and exports totaled about 1,370,000 tons in 1953. Projecting requirements and assuming a high level of industrial activity, total zinc requirements are expected to approximate 1,450,000 tons by 1960 and 1,650,000 tons by 1975.

PROBLEMS

To produce zinc at a competitive price, industry undertakes numerous research problems involving ore finding and improved mining, milling, and smelting practice, as well as market research and development. Government also conducts numerous research projects in exploration, ore dressing, and extractive and process metallurgy, as well as in mineral market surveys.

The easily discovered deposits have been or are being exploited. New ore discoveries will require intensive scientific search, utilizing advance prospecting and exploration techniques involving improved geological techniques and exploration tools. Research to lower the unit cost of exploratory openings and to improve the accuracy of sampling is particularly needed.

Research problems in mining that need particular study include lower cost ore breakage, ground support, and ore transport. An element in the solution of this and all other industry problems is widespread publication of acceptable solutions, so that good practice will become widespread. There is constant need for research to improve fine crushing and grinding, frequently the highest cost items in the ore-dressing process. Research should be particularly focused on the recovery of slime sulfides and the overall recovery of the complex and difficult-to-recover oxide minerals.

At present zinc ores are reduced to metal by five commercial processes that vary widely in metallurgical efficiency, labor, and capital requirements. Although much research has been done in developing and improving them, much remains to be done to improve metal extraction and unit cost.

Market research offers a large area in developing better alloys and improved or new applications for them. The ample supply and low cost of zinc justify much research on products and markets by both producers and consumers.

The zinc industry needs better information on which to base industry decisions. The solution of industry problems often requires tabulation of adequate up-to-the-minute statistics to permit more accurate predictions, so that productive capacity and stocks can be maintained at optimum levels.

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ZIRCONIUM AND HAFNIUM

By

Horace T. Reno[1]

ZIRCONIUM is the metal that made possible the Navy's atomic-powered submarine, *Nautilus*. Increased production of pure zirconium resulted in the production of hafnium, removing it from the list of those metals classified as laboratory curiosities.

Summary

Zirconium, in the form of the oxide, was discovered by Klaproth in 1789; it was not, however, until 1914 that the metal was obtained. Hafnium was discovered in 1911 in X-ray spectra, and pure hafnium salts were isolated in 1931.

Zirconium and hafnium occur together in nature. The commercial source minerals, zircon and baddeleyite, are obtained from beach sands and river gravels in the United States, Australia, Brazil, French West Africa, and India. Ore reserves are greater than the demand foreseeable within the next 100 years.

The Federal Bureau of Mines, Carborundum Metals, Inc., Foote Mineral Co., and Zirconium Metals Corp. of America are the principal producers of ductile zirconium and hafnium. In production of the ductile metals, the ore is first decomposed, then the zirconium compounds are separated from the hafnium compounds, and finally the compounds are treated to yield pure zirconium and hafnium. The van Arkel iodide process and the Kroll magnesium-reduction process are employed in treating the compounds to produce ductile zirconium and hafnium.

Zircon and zirconium compounds that are made directly from zircon and baddeleyite—not zirconium and hafnium metal—are the principal end products derived from zirconium-hafnium ores. Zircon is chemically inert and a good thermal conductor and has high specific gravity and melting point, properties that make it an outstanding material for use in refractories and foundry sands. Stabilized zirconium oxide is a poor conductor of heat and can withstand temperatures to 2,535° C. It is used to insulate high-frequency induction or resistance furnaces and to line jet engines. Other zirconium compounds are important; the boride, for example, is an excellent high-temperature material; soluble salts precipitate dyestuffs and render textiles water-repellant; and the phosphate and carbonate are used in medicines. Zirconium metal is used in nuclear reactors, in steel alloys, and in surgery as human-body-replacement material. Hafnium is also used in nuclear reactors.

Nuclear energy development, current commercial research, and more efficient processing methods doubtless will increase the market for zirconium and hafnium. If this increase causes the demand for zircon to exceed the supply of ore mined as a byproduct of rutile, ilmenite, and monazite mining, a sharp rise in the cost of zircon ore can be expected. The chief problem of the zirconium-hafnium industry is to reduce the cost of the pure metals by reducing the cost of treating the ores.

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[1] Commodity-industry analyst, Bureau of Mines.

BACKGROUND

HISTORY

Zirconium was discovered in 1789 by the German chemist, Klaproth, while analyzing jargon, a colorless, smoky zircon gem stone. He separated zirconia, the oxide of zirconium, but he and other chemists who confirmed his discovery could not isolate the metal. It was not obtained until 1914, when Lely and Hamburger, Netherlands chemists, succeeded in producing a small quantity of impure zirconium in the laboratory. Two other Netherlands chemists, van Arkel and de Boer, developed a satisfactory method for producing the pure metal in 1925 (*3*).[2] This method, known as the van Arkel or iodide process, although slow, expensive, and not adaptable to large scale production, was the only one used commercially until 1947, when Dr. W. J. Kroll, working in Federal Bureau of Mines laboratories, adapted his titanium-magnesium reduction process to the production of zirconium.

The history of hafnium is vague. Svanberg, the Swedish chemist, announced in 1845 that zircon was composed of two distinct earths. Others, failing to verify Svanberg's conclusion, pronounced this conclusion erroneous. Urbain in 1911 and Coster and Heresy in 1922 claimed to have discovered hafnium by examination of X-ray spectra (*3*). Pure hafnium salts were not isolated, however, until 1931.

MINERALOGY

Zirconium is associated with other metals in many minerals but is recovered only from the minerals zircon and baddeleyite.

Zircon, a zirconium orthosilicate ($ZrO_2.SiO_2$), grades from colorless to pale yellow, to brownish yellow, to grayish green or reddish brown. It has a hardness of 7.5, a specific gravity of 4.2 to 4.86, a conchoidal fracture, and an adamantine luster. Some colorless zircons are valued as gems.

Baddeleyite, the zirconium dioxide, grades from colorless, to yellow, to brown, to black. It has a hardness of 6.5, a specific gravity of 5.5 to 6.0, a conchoidal to uneven fracture, and a greasy to vitreous luster.

Hafnium is invariably associated with zirconium. It has not been found in minerals that do not contain zirconium.

SOURCES

Zircon occurs in all plutonic rocks but is more common in granite, syenite, and diorite. It is a common constituent of river gravels and beach sands because it is hard and resists corrosion. Beach sands are the principal source of zircon where it is recovered as a coproduct of ilmenite, rutile, and monazite.

Baddeleyite usually occurs in phonolite and, like zircon, is present in river gravels and beach sands. Commercial deposits are known only in Brazil.

The cost of mining zircon and baddeleyite is low compared with the cost of transporting and marketing the ores; these circumstances prevent exploitation of many remote desposits that are as rich or richer in zircon than those being mined.

Zirconium ore is produced commercially in five countries: United States, Australia, Brazil, French West Africa, and India. The principal mines in the United States are along the eastern coast of Florida, where zircon, with ilmenite, rutile, and monazite, is recovered from beach sands.

Australian zircon mines are on the beaches of the eastern coast between Bollina, New South Wales, and Stradbroke Island, Queensland. Australian mines compete with United States mines, probably because of a favorable price differential and a tie-in with rutile sales. Brazilian zirconium ore is produced from the baddeleyite mines at Cascata, near Pocos de Caldas. French West African ore is produced from the beach deposits of Senegal and Cascamance where zircon is recovered with ilmenite.

RESERVES

World reserves of zirconium ore are immense. United States reserves alone are greater than the demand foreseeable within the next 100 years.

INDUSTRY

The companies comprising the zirconium industry are well financed and progressive. Their plants, located to take advantage of proximity to raw materials and market, are well able to supply the requirements of the United States for zircon, zirconia, and zirconium alloys; but, if they are to supply the demand for ductile zirconium, they must be expanded.

The Federal Bureau of Mines Electrodevelopment Laboratory at Albany, Oreg., is the major producer of Kroll-process hafnium-free zirconium sponge, zirconium-alloy ingots, and hafnium sponge. Bureau production is consigned to the Atomic Energy Commission. Carborundum Metals, Inc., Akron, N. Y., began production late in 1953 of Kroll-process zirconium sponge for the Commission. This company and others planning expansion to produce Kroll-process zirconium will eventually allow the Bureau of Mines to withdraw from production and devote all its efforts to research.

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[2] Italicized figures in parentheses refer to items in the bibliography at the end of this chapter.

Other processors of zircon and manufacturers of zirconium and hafnium products are:

Producer and plant location	Products
Zirconium Metals Corp of America (subsidiary of National Lead Co.) New York 6, N. Y.	Ductile zirconium; zirconium compounds.
Titanium Alloy Mfg. Division of National Lead Co., New York 6, N. Y.	Stabilized zirconia refractories; ground zircon.
Orefraction, Inc., Pittsburgh, Pa.	Granular and milled zirconium silicate and zirconium porcelains; enamels; refractories; glass; pottery.
Foote Mineral Co., Philadelphia, Pa.	Iodide-process zirconium crystal bar; hafnium crystal bar; zirconium metal shapes.
Allegheny-Ludlum Steel Corp., Watervliet, N. Y., and West Leechburg, Pa.	Zirconium ingots and shapes; melting and rolling mills.
Firth Sterling, Inc., 3113 Forbes St., Pittsburgh 30, Pa.	Zirconium ingots and shapes; melting and rolling mills.
DeRewal International Rare Metals Co., Philadelphia 5, Pa.	High-purity zirconium metal powder; oxide; compounds. Hafnium metal powder; oxide; compounds.
Electro Metallurgical Division, Union Carbide & Carbon Corp., New York 17, N. Y. (plants at Niagara Falls, N. Y., Sheffield, Ala., and Alloy, W. Va.)	Zirconium alloys and briquets.
Metal Hydrides, Inc., Beverly, Mass.	Zirconium-metal powder; zirconium hydride; zirconium alloys.
Westinghouse Electric Corp., Pittsburgh, Pa.	Zirconium crystal bar; metal shapes.
Ceramic Color & Chemical Mfg. Co., New Brighton, Pa.	Zirconium porcelains; enamels; refractories; glass; pottery; compounds.
Corhart Refractories Co., Louisville, Ky.	Refractories.
Kawecki Chemical Co., New York 17, N. Y.	Zirconium fluorides.
Massillon Refractories Co., Massillon, Ohio.	Refractories.
Metal & Thermit Corp., New York 17, N. Y.	Zirconium compounds for pottery industry.
Morton Co., Worcester 6, Mass.	Fused stabilized zirconia refractories; granular zirconia.
Pacific Graphite Co., Inc., 40th and Linden, Oakland, Calif.	Foundry facings.
Rohm & Hass Co., Philadelphia 5, Pa.	Zirconium sulfate solution (tanning agent).
Shieldalloy Corp., New York 17, N. Y.	Milled and granular zircon.
Stauffer Chemical Co., New York 17, N. Y.	Zirconium tetrachloride (custom chlorination).
Chas. Taylor & Sons (subsidiary of National Lead Co.) Cincinnati, Ohio.	Refractories.
Titanium Zirconium Co., Inc., Flemington, N. J.	Zirconium salts and compounds.
Zirconium Corp. of America, Solon, Ohio.	Stabilized zirconia; zirconium compounds.

TECHNOLOGY

Zirconium and hafnium closely resemble each other, in that they are ductile, crystalline metals highly resistant to corrosion (*6*) (*8*). Some of their physical properties are compared in table 1.

TABLE 1.—*Physical properties of zirconium and hafnium*

	Zirconium	Hafnium
Atomic number	40	72.
Atomic weight	91.22	178.6.
Specific gravity	6.52	13.3.
Melting point	1,857° C	2,200° C.
Boiling point	Above 2,900° C	Above 3,200° C.
Specific heat (cal./gm./° C.)	0.067 (20°)	0.0351 (25°).
Hardness	105 [1]	152.[1]
Modulus of elasticity	11.3	20.
Neutron cross section (barns)	0.4	105.0.

[1] Vickers hardness number.

Zirconium and hafnium are extracted from their ores in three operations: (1) The ore is decomposed, (2) zirconium compounds are separated from hafnium compounds, and (3) the compounds are treated to yield the pure metals.

The ores can be decomposed by heating to volatilize the silica, by sintering and digesting with acid, by fluxing with alkali, by fluxing with nitric cake, or by carbiding followed by direct chlorination. Zirconium tetrachloride, the end product of the direct-chlorination method, is the starting compound of the Kroll process (*7*).

Complete separation of hafnium and zirconium has been impossible because, insofar as is known, they react chemically in much the same way. Practical separation is made, however, by taking advantage of slightly different chemical actions and repeating the process until the desired purity is obtained.

Current hafnium-zirconium separation methods have not been published. Older methods include the fractional phosphate-precipitation process; the fractional crystallization process, using ammonium hexachloride and oxychloride; and the fractional distillation process (*7*). The phosphate-precipitation process is the most satisfactory of the three—hafnium phosphate is less soluble than zirconium phosphate, and because phosphate crystals are granular and easily filtered, separation is obtained rapidly with few precipitations.

Several methods are known for producing zirconium and hafnium; but only two commercial processes, the van Arkel (iodide) process and Kroll (magnesium-reduction) process, produce metals pure enough to be ductile (*7*).

In the van Arkel process, zirconia is reduced with calcium, yielding impure zirconium metal, which is combined with iodine at 200° C. to form zirconium iodide. The zirconium iodide is heated to 1,300° C., at which temperature it decomposes to pure zirconium and iodine.

The Kroll process utilizes molten magnesium to reduce purified zirconium tetrachloride in an inert atmosphere of argon or helium.

The van Arkel and Kroll processes, with minor modifications, work equally well to produce ductile hafnium.

USES

Zirconium and hafnium have been highly publicized in the past few years, leading to the popular belief that most of the zirconium ores consumed in the United States are treated to recover the metals. However, zirconium ores consumed in the United States in 1953 were distributed 28 percent to refractories, 27 percent to foundry sands, 20 percent to ceramics, 12 percent to metals, and 6 percent to chemicals.

Zircon in the principal zirconium ore. It is an outstanding refractory because it is chemically inert, is a good heat conductor, has a high melting point, and has high specific gravity. Zircon refractories are particularly useful in aluminum-melting equipment because they are not wetted by molten aluminum. Zircon foundry sands are superior to silica sands because they provide rapid chilling, being better heat conductors. They are supplanting metal shot as a chilling medium in magnesium castings.

Zirconia (ZrO_2) is unstable, changing crystal form with changing temperatures, but it can be stabilized by adding small quantities of the oxide of magnesium, calcium, scandium, or yttrium. When stabilized, it is a poor conductor of heat and will resist thermal shock. It can withstand temperatures to 2,535° C. and, at temperatures above 1,500° C., it is an excellent electrical conductor. These characteristics make it a good material for insulating high-frequency inductor or resistance furnaces and lining jet engines.

Zirconium boride has been reported to withstand temperatures as high as 6,000° F. At 2,800° F. it has a shear strength of 75,000 pounds per square inch, but it oxidizes rapidly and therefore can not be used at these high temperatures without an inert coating. Zirconium phosphate is used to treat stomach ulcers, and zirconium carbonate is used in poison-ivy ointment and as an antiperspirant. Soluble zirconium salts are used to render textiles water-repellant and to precipitate acid dyestuffs.

Impure zirconium is hard and brittle. It has been used little since electric flash bulbs replaced zirconium flash powder. Pure zirconium is ductile and malleable but expensive. It is used widely in nuclear apparatus because it is strong, resists corrosion and has a high melting point and because of its low neutron absorption factor. Molten zirconium wets graphite, making possible direct graphite-metal contacts in rectifier tubes.

Zirconium is suitable for use in surgery. Body fluids will not corrode it, and human tissue grows to it without decay or deterioration. It is being investigated as a spacer to isolate the void left after a lung-removal operation and is replacing tantalum in bone screws, suture wire, and cranial plates.

Zirconium alloys are becoming increasingly important in industry. Zirconium imparts strength to steel and causes it to harden under heat treatment—it acts as a deoxidizer and scavenger of nonmetallic intrusions. Zirconium-magnesium-cerium alloys are used for lightweight stress members. A gold-zirconium alloy can replace platinum-iridium in fountain-pen tips.

Small quantities of hafnium have been used in carbide cutting tools, and fluorescent powders. In general, however, hafnium has been produced in the past only for use in the laboratory. The high neutron-absorption factor of hafnium makes it of interest to designers of nuclear energy equipment, but security regulations have prevented the publication of detailed data on its use.

PRICES

Quoted by E & MJ Metal & Mineral Markets during November 1954

Zircon ore (sand) min. 65 percent $Zr\ O_2$ per long ton c. i. f. Atlantic seaboard	$47.00 @ 48.00
Zirconium alloy 12–15 percent Zr; 39–43 percent Si bulk, carload lots, per pound	.08
Zirconium-metal powder, per pound	7.00

Quoted by Zirconium Metals Corp. of America (subsidiary of National Lead Co.), late 1954

Zirconium-metal sponge and briquets, per pound	$10.00
Zirconium hot-rolled plate and bars, per pound, base price	27.00
Zirconium cold-rolled strip, per pound, base price	35.00
Zirconium cold-drawn wire 0.060–0.375 in. diameter, per pound	42.50 –32.50

Quoted by Foote Mineral Co., late 1954

Iodide-process ductile zirconium metal:	
Zirconium crystal bar, lots over 100 pound, per pound	$70.00
Zirconium wire annealed, 0.050–0.005 inch diameter, per kilogram	450.00–600.00
Zirconium sheet, 0.010–0.002 inch thick, per kilogram	425.00–750.00
Zirconium powder, pyrotechnic grade, 100-pound lots or over, per pound	10.50

Quoted by Electro Metallurgical Division of Union Carbide & Carbon Co., late 1954 f. o. b. railroad freight cars at destination

Zirconium–ferrosilicon:	
12–15 percent Zr per pound, depending on quantity and quality	$0.08 –0.1075
35–40 percent Zr per pound, depending on quantity and quality	0.2525–0.3025
Zirconium briquets (11 percent Zr, 38 percent Si) per pound, depending on quantity	0.075 –0.10
Nickel zirconium (40–50 percent Ni, 25–30 percent Zr) per pound, depending on quantity	$1.25–1.35

Quoted by DeRewal International Rare Metals Co., late 1954

Hafnium-metal powder (99.3 percent), per gram	$25.00
Hafnium oxide (99.5 percent), per gram	17.00
Hafnium tetrachloride (99 percent), per gram	16.00
Hafnyl sulfate, nitrate, and chloride (99 percent), per gram	15.00

OUTLOOK

Apparently no material has been found that is as satisfactory as zirconium as a structural material in nuclear reactors. Nuclear energy development undoubtedly will increase the market for zirconium and probably for hafnium, notwithstanding their high price.

Zirconium will doubtless replace tantalum in dental and surgical equipment. It is cheaper than tantalum and as satisfactory, or more satisfactory, in some equipment.

Many manufacturers—chemical, electrical, and mechanical—are investigating ductile zirconium. They undoubtedly will find many places where zirconium can replace more expensive material or where zirconium can be applied to improve their products. This research should increase zirconium consumption.

More zirconium production should result in lower prices, which, in turn, would stimulate research to improve technology and develop new uses. Because it is impossible to produce pure zirconium without, at the same time, making hafnium compounds, hafnium will be more plentiful for nuclear applications but might not be available commercially.

The consumption of zircon, zirconia, and the cheaper zirconium compounds, although fluctuating with the economy, apparently will gradually increase, but the prices of these materials probably will not change much unless demand, coupled with the demand for zirconium and hafnium metal, exceeds the supply from ore mined as a byproduct of rutile, ilmenite, and monazite. If the demand does exceed this supply and mines are operated solely to recover zircon, a sharp rise in the price of ore can be expected.

PROBLEMS

The chief problem of the zirconium-hafnium industry is to reduce the cost of the pure metals. Mining, transporting, and selling costs compare favorably with those of other metals; attention therefore must be directed to reducing the cost of treating the ores.

Zircon and baddeleyite ores are chemically stable, and high temperatures are required to decompose them. The cost of decomposing these ores probably cannot be reduced much, but no fundamental reason exists to preclude reducing the cost of purifying and separating zirconium and hafnium compounds after the ores are decomposed.

The Kroll process, although much cheaper than the van Arkel process, which it has largely supplanted, is too expensive a method of producing zirconium and hafnium to allow these metals to compete successfully in commercial markets. More than likely, Kroll-process cost can be reduced to some extent; but, because it is a batch process, the costs tend to be relatively high. One of the problems of industry therefore is to develop a cheap, continuous process for producing pure zirconium and hafnium.

Zirconium, although available and in many instances cheaper than competing metals, is not being used extensively because it is one of the relatively unknown, newer ones. Industry has the problem of publicizing the properties and availability of zirconium.

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By

Mabel L. Winslow

U. S. GOVERNMENT PRINTING OFFICE: 1956