ERRATA

Staff, Bureau of Mines. AN APPRAISAL OF MINERALS AVAILABILITY FOR 34 COMMODITIES. BuMines B. 692, 1987, 300 pp.

INTRODUCTION

Page 2 Figure 1: Hafnium should be added to the group of mineral commodities depicted on the graph.

Page 5 Figure 3: Shading to the right of each of the availability curves to indicate expected, but unknown resource additions (referred to in the last whole paragraph, page 10) was omitted.

Page 7 Figure 5: The figure caption should be "Example mine cost plot from CES."

Page 9 Table 4: The commodity price categories for chromium (i.e., Chromium: Concentrate, Ferrochromium High grade and Low grade, Metallurgical grade, Refined grade, and Chemical grade) are incorrect and should be replaced as follows.

Commodity	Unit	Unit Price
Chromite:		
high-chromium	mt	110.00
high-iron	mt	50.00

Page 11 The graphs of figures 6 and 7, but not the figure captions, are transposed. The lower graph should be "Figure 6. -- Annual resource availability curve"; the top graph should be "Figure 7. -- Potential annual production from producing mines".

ALUMINUM CHAPTER

Page 17 Second paragraph: The recently closed aluminum refinery at Hurricane Creek, AR, is owned by Reynolds, not Kaiser.

Page 19 The second sentence, second paragraph should read: "Weighted-average mining and drying costs in MEC's are only $0.02/lb (not $0.20/lb) of aluminum."

IRON ORE CHAPTER

Page 135 The last sentence, second paragraph should read: "At this price, approximately 0.35 billion lt is potentially available from foreign producers at 15-pct DCFROR, and less than 100 million lt from nonproducers."

Page 137 First sentence, last paragraph: The "140 million" should be "160 millio-"

LITHIUM CHAPTER

Pages 157-163	The chemical formula for lithium carbonate is incorrectly written throughout the chapter. The correct formula, Li_2CO_3, should replace $LiCO_3$, wherever the latter occurs, except: Page 157, sixth paragraph and eighth paragraph (first sentence) where $LiCO_3$ should be lithium.
Page 159	Table 2: The chemical formula for lithium oxide in PEGMATITES (product grade in percent LiO_2) should be Li_2O.

MOLYBDENUM CHAPTER

Page 192	First paragraph below Table 1: "The United States accounts for 53 pct (not 50 pct) of the analyzed demonstrated resources..."

NICKEL CHAPTER

Page 203	Table 2: The ore types of the New Caledonia nickel properties are laterites, not sulfides.

SILVER CHAPTER

Page 250	Paragraph no. 1 in Mining: The outputs of Mexican and U.S. surface mines should be 9.125 (not 9,125) and 3.865 (not 3,865) million tr oz of silver, respectively.
Page 251	Third paragraph: The upper bound of the capacity range for the five flotation mills should be 1.797 (not 1,797) million mt/yr.

Bulletin 692

An Appraisal of Minerals Availability for 34 Commodities

Compiled by Staff, Bureau of Mines

UNITED STATES DEPARTMENT OF THE INTERIOR
Donald Paul Hodel, Secretary

BUREAU OF MINES
David S. Brown, Acting Director

As the Nation's principal conservation agency, the Department of the Interior has responsibility for most of our nationally owned public lands and natural resources. This includes fostering the wisest use of our land and water resources, protecting our fish and wildlife, preserving the environment and cultural values of our national parks and historical places, and providing for the enjoyment of life through outdoor recreation. The Department assesses our energy and mineral resources and works to assure that their development is in the best interests of all our people. The Department also has a major responsibility for American Indian reservation communities and for people who live in island territories under U.S. administration.

Library of Congress Cataloging-in-Publication Data

An appraisal of minerals availability for 34 commodities.

(Bulletin ; 692)

Includes bibliographies.

Supt. of Docs. no.: I 28.3: 692

1. Nonfuel minerals industry. 2. Mines and mineral resources. 3. Nonfuel minerals industry—United States. 4. Mines and mineral resources—United States. I. United States. Bureau of Mines. II. Series: Bulletin (United States. Bureau of Mines); 692.

HD9506.A2A72 1987 333.8'5 87-600244

For sale by the Superintendent of Documents, U.S. Government Printing Office
Washington, DC 20402

PREFACE

The Bureau of Mines assesses the international availability of selected minerals of economic significance, most of which are essential materials to the United States. The Bureau identifies, collects, compiles, and evaluates information on producing mines, developing and explored deposits, and on mineral processing plants in market economy countries. Objectives are to classify domestic and foreign resources by economic evaluation methods, and to prepare analyses of minerals availability.

This publication summarizes and updates a series of availability appraisals, published between 1979 and 1987, that analyzed the availability of 34 minerals from market economy countries. Updated availability information on these minerals is summarized in commodity chapters. The introduction to this bulletin provides a detailed explanation of the assessment procedures. Questions about, or comments on, the appraisals should be addressed to Chief, Division of Minerals Availability, Bureau of Mines, 2401 E Street, NW., Washington, DC 20241.

ACKNOWLEDGMENTS

This bulletin was prepared under the direction of first, Robert L. Marovelli (now retired), and Donald G. Rogich, Chief, Division of Minerals Availability, and Harold Bennett, Chief, Minerals Availability Field Office. Numerous Bureau individuals associated with the Minerals Availability Program over many years made significant contributions to the results contained herein. Personnel in the Minerals Availability Field Office, Denver, CO, and Division of Minerals Availability, Washington, DC, prepared the availability assessment and wrote the reports. Data development was the primary responsibility of Bureau field evaluators and contractors. Mineral commodity specialists of the Directorate of Minerals Information also provided data and reviewed the commodity chapters. Much of the geologic information referred to in this report was derived from U.S. Geological Survey (USGS) Professional Paper 820 and consultation with USGS mineral-resource geologists. Special recognition is due Gary Kingston, former Chief, Division of Minerals Availability who, in large part, conceived of and directed the development of the assessment capability and reviewed a portion of the bulletin.

Hermann Enzer
Assistant Director—Mineral Data Analysis

CONTENTS

UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

cm	centimeter	lb/yr	pound per year
¢/lb	cent per pound	lt	long ton
°C	degree Celsius	ltu	long ton unit
°F	degree Farenheit	m	meter
$/d	dollar per day	mi^2	square mile
$/flask	dollar per flask	mill/kW•h	mill per kilowatt hour
$/kg	dollar per kilogram	min	minute
$/lb	dollar per pound	mm	millimeter
$/lt	dollar per long ton	mt	metric ton
$/mt	dollar per metric ton	mt/d	metric ton per day
$/tr oz	dollar per troy ounce	mt/h	metric ton per hour
ft	foot	mt/yr	metric ton per year
g/mt	gram per metric ton	pct	percent
g/tr oz	gram per troy ounce	pct/yr	percent per year
gal/min	gallon per minute	ppb	part per billion
h	hour	ppm	part per million
in	inch	st	short ton
kg	kilogram	st/d	short ton per day
km	kilometer	st/yr	short ton per year
kW•h	kilowatt hour	tr oz	troy ounce
kW•h/mt	kilowatt hour per metric ton	tr oz/yr	troy ounce per year
L	liter	wt pct	weight percent
lb	pound	yr	year

INTRODUCTION

PROGRAM OVERVIEW

The Bureau of Mines, through its Minerals Availability Program (MAP), investigates the mineral supply potential of the United States and market economy countries (MEC's) for a number of minerals considered essential to the U.S. industrial base. Major activities include identification of important mines and mineral deposits, estimation of their mineral resources and production costs, and availability and sensitivity analyses. Products include data printouts of geological and engineering information, discounted-cash-flow rate of return (DCFROR) analyses of individual mining operations, commodity availability appraisals, and analytic services to Government agencies. These products are used in assessing the mineral supply position of the United States; in setting priorities for research aimed at reducing production costs and minimizing adverse environmental impacts and regulatory compliance costs of mining and mineral processing operations; and in formulating and evaluating Government policies that affect minerals.

Bureau MAP data and analyses have been provided to the executive and legislative branches to aid in consideration of trade, tax, and stockpile policy issues. A study of the economic potential of selected seabed mineral resources supports the regulatory responsibilities of the Department of the Interior in the U.S. Exclusive Economic Zone. Analytic studies of mineral project viability, industry competitiveness, and import dependencies were also undertaken for, or in cooperation with, U.S. and foreign government agencies, industry and trade organizations, and international lending institutions.

Program results are based on a comprehensive data base of resource, cost-engineering, and economic information on approximately 2,900 major world mines and mineral processing plants. Automated analytic and data updating systems and a multidisciplinary staff in the physical and computer sciences, engineering, economics, and operations research ensure informed and comprehensive analysis of the many variables in mineral supply analysis.

This Bulletin summarizes and updates availability information published in individual mineral commodity appraisals since 1979. It also marks achievement, after a 15-yr systems and data development history, of a major Bureau goal to build computer-based systems and the necessary staff infrastructure to perform timely, commodity-wide assessments within comparable cost frameworks. The basic approach of site-specific, feasibility-type evaluations of mines and processing plants permits not only commodity availability analyses, but also sensitivity analyses of the impacts of physical, technical, and economic variables on costs of individual producers.

Program emphasis is now on maintaining and refining basic data and analytic capabilities, performing more detailed analysis of a larger portion of the materials cycle for about a dozen of the 34 minerals, and performing issue analyses using this data system.

HISTORY OF THE AVAILABILITY STUDIES

Bureau attempts to inventory the national mineral resource base date from the 1930's, with studies of manganese, copper, and other commodities. Subsequently, with the War Minerals reports of World War II and the Materials Surveys beginning in the late 1950's, the Bureau began an evolutionary trend towards systematic commodity inventorying. A comprehensive attempt at defining the national resource positions for copper, lead, zinc, gold, and silver was conducted about 1954 with the classification of U.S. resources according to measured, indicated, and inferred categories. Although these unpublished studies were of limited scope, they contributed to the conceptual development of the current availability studies. They did not, however, aggregate or economically classify resources over a range of prices. The current studies, which depict mineral production capability at various prices, began about January 1967, under a program entitled "The Availability of Minerals Under Various Economic Conditions." That program yielded reports documenting the availability of silver, molybdenum, chromium, nickel, uranium, tungsten, and other commodities (*1-4*).[1]

The design of an automated minerals availability system (MAS), began in 1971, at the Bureau's regional field operations centers. The objective was to build a capability for timely and comparable evaluations of mineral supply potential under actual and hypothetical economic scenarios. The initial work involved developing the data input format and coding instructions for an automated resource classification system and manual (*5*). Data entry pilot projects and systems design activities were conducted for copper, fluorspar, and tungsten deposit data. To augment the Bureau's data development, grants were awarded to State governments and universities for supplemental evaluations of aluminum, gold, lead, silver, titanium, tungsten, uranium, vanadium, and zinc deposits.

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

The rapid accumulation of evaluation data necessitated establishing, in 1974, project coordination in Washington, DC, to centralize management and planning. Short-term goals were to automate data and basic analytic procedures. Longer term planning goals included producing short- and long-term commodity supply schedules, sensitivity studies of supply response as a function of physical and economic variables, supply-balance models, and linear programming network models of supply. All were deemed necessary to ultimately achieve a complete analytic capability (*6*).

An important step toward realizing these capabilities was the establishment, in 1975, of an operations group to undertake systems design and development and to manage the expanding data base. This office was located in Denver, CO, in proximity to the Bureau's automated data processing division. The first project was the development of a cost estimating system to assure consistent engineering cost data inputs, particularly for undeveloped deposits. The cost estimating system (CES) handbook, completed in 1978, is a standard program estimating tool. Systems design work also continued on the mine simulation (MINSIM) program, an economic evaluation software program, started in the late 1960's to evaluate individual mining operations by DCFROR methods (*7*). This software program evolved through several stages into the supply analysis model (SAM), an integrated system that aggregates the resources of multiple properties under cost (availability) curves. The international mining cost indexation system, started in 1982, is used to update and convert to U.S. dollars, foreign operation cost estimates to achieve comparability across international boundaries. The SAM resides on a Burroughs B-6800 mainframe computer and a WANG VS-100 microcomputer.

The first data development step was compiling a comprehensive list of mines, mineral properties, and occurrences to serve as a foundation for detailed availability asssessments. This resulted in a data subsystem of identification and location information which became known as the mineral industry location system (MILS). Although MILS-type data were intended to guide the development of the minerals availability data base, the MILS found wider utility as a comprehensive source of minerals location data for Federal land managers and minerals exploration groups (*8*).

The initial projection of the ultimate data file content was 100,000 to 200,000 deposits worldwide, of which, 2,000 to 3,000 would have short-term production significance, another 5,000 to 10,000 would have paramarginal supply potential, and the remainder would have long-term potential for supply development. To date, approximately 2,900 world properties (including processing plants) are fully evaluated, while 200,000 location entries, predominantly in the United States, compose MILS. These data also reside on the Burroughs B-6800. Figure 1 profiles the program data holdings.

Prior to 1978, the development of complete mine property cost evaluations was impeded by still evolving data requirements and evaluation procedures, and a lack of dedicated program funding. In 1978, the Government Accounting Office recommended that MAS be upgraded from a research and development type project to a Bureau program to assure the commitment of adequate resources to attain its developmental and policy support goals (*9*). Subsequently, the Bureau established the MAP as one of four programs in the Mineral Data Analysis Directorate, consisting of a headquarters office in Washington, DC, the Minerals Availability Field Office in Denver for system development and supply analysis, and minerals availability branches in the field operations centers for primary data development. The increased program resources and the establishment of firm data specifications made possible the accomplishment of data development goals. Approximately 2,400 major U.S.

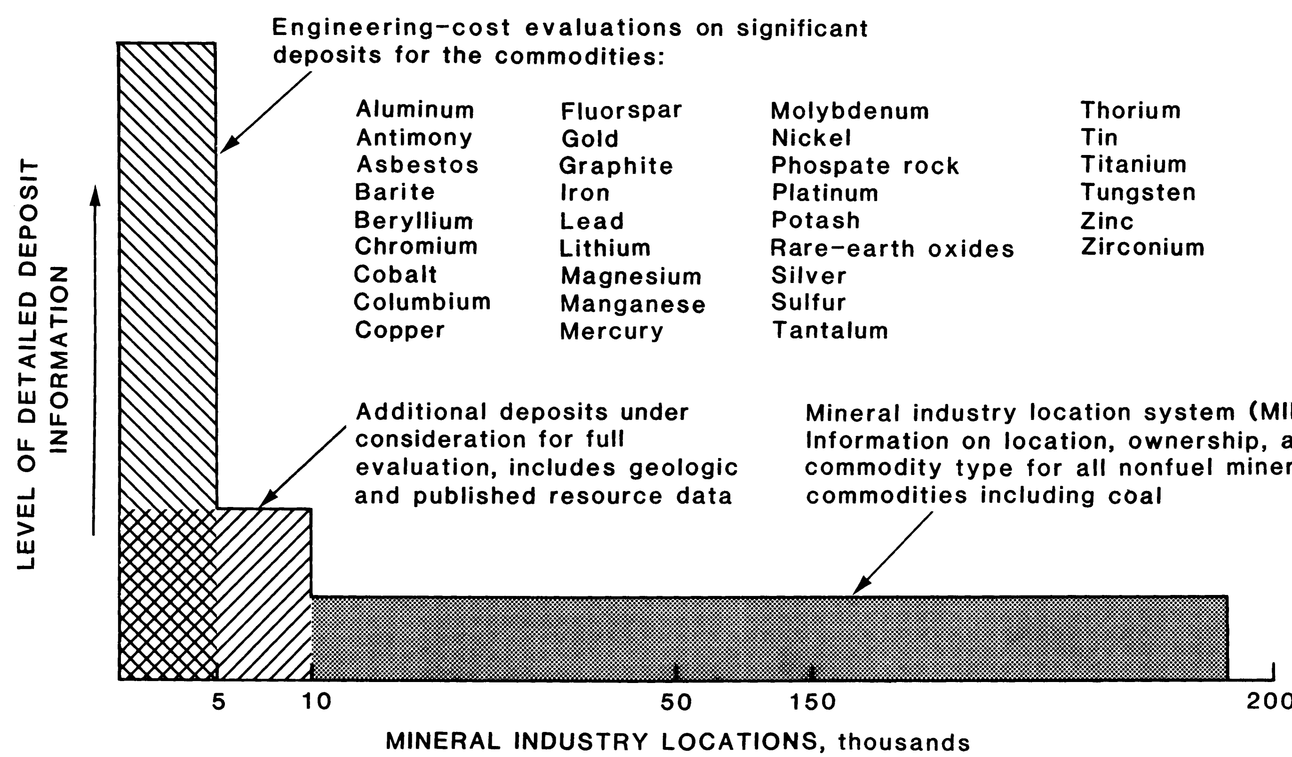

Figure 1.—Minerals availability data profile.

and foreign mines and deposits were fully cost evaluated by 1982. Foreign properties have been evaluated largely under contract by internationally experienced engineering firms. The advice and recommendations of consultants and program evaluators (*10-11*), with respect to analytic methods, policy utility, public access, and linkage with other minerals information systems, have been sought throughout the development of methodologies and systems.

Since 1982, emphasis has been on reporting the results of the availability studies, refining data and analytic support systems, and on making data and systems available to users through technology transfer seminars and professional presentations and publications. Wider public knowledge of the availability studies and Bureau capability have resulted in requests for issue analysis and have provided opportunities for outside review and comment on program results and methods. Intended originally to support the minerals policy and information responsibilities of the Department of the Interior, the Bureau's Minerals Availability Program now provides analyses of a wide range of U.S. and international minerals issues to a wide range of users.

MINERAL RESOURCE CLASSIFICATION

Mineral resources are generally classified on the basis of the geologic certainty of the tonnage and quality of the valuable minerals and on the estimated profitability of their extraction. However, there is no universally accepted formula for the classification of mineral resources; and, while terms such as resources, reserves, availability, ore, etc., are in common use within the U.S. and international minerals profession, they are inconsistently applied and have different connotations. Resource estimators, whether private or government, typically have wide latitude to calculate and report reserves according to different circumstances and objectives. Hence, methods of estimation and reporting vary among companies and operations and are influenced by factors as diverse as local tax policy and mineral deposit type. For example, the knowledge of geologic potential for ore extension at a particular site may be so favorable that the expense of developing long-range reserves is not necessary. Additional considerations that may bias mining engineering estimates and reporting of reserves include lender requirements and securities regulations. Political motivations may also influence resource reporting in certain countries. The Bureau follows the classification principles and terminology agreed to with the U.S. Geological Survey to evaluate and describe resources (*12*). This system was developed to improve the basis for national mineral resource planning and policy analysis by improving the comparability and consistency of resource reporting by Government minerals analysts.

Figure 2 illustrates the relational categories of this classification system. Resources is an all-inclusive, conceptual term encompasing concentrations of identified (and undiscovered) mineral resources from which economic minerals extraction is currently or potentially possible. Identified resources range geologically from well defined to inferred, and economically from highly profitable to highly unprofitable. Demonstrated resources are identified resources whose tonnage and quality have been estimated by engineering measurements. Reserves are select resources whose existence and nature are well established and which, at the time of classification, can be mined at a profit.

Reserve base is a more subjective category. It encompasses demonstrated resources that are, or are presumed to be, technically and economically recoverable in the foreseeable future. By definition, reserve base is less sensitive to commodity prices than are reserves. The term was introduced to overcome inherent problems in aggregations of commodity reserves. These include implications that because a mine is operating it is profitable, that reported reserves are steadfast when, technically, they are price (and cost) sensitive, and that mineral producers (hence, reserves) are equally viable. Qualitiative differences among mines and deposits and the variable effects of price fluctuations on operations are not apparent from most reserve estimates. Also, reserve data strictly reported would yield estimates that would vary widely over time, and would represent a too conservative supply perspective for national purposes. The MAP evaluates mineral properties within the reserve base to provide a comprehensive resource and production cost information base for policy analysis and long-term planning decisions.

Any assessment of economic resource potential is inevitably a point-in-time estimate, characterized by incomplete geological knowledge of resources. Assessments are subject to the professional judgment of the evaluator to make assumptions with respect to extraction technology and the impact of interplay of political, economic, and institutional factors on minerals supply. While determination of tonnage and grade is a relatively straightforward process of exploration, economic value can be estimated only by consideration of production costs and commodity prices, which are variable over the life of the mine. Economic evaluation is complicated by the presence of coproduct and byproduct elements that enhance or detract from the value of the primary commodity. For some important minerals, e.g., cobalt, which is produced almost exclusively as a byproduct of copper and nickel mining, the availability of the byproduct commodity is determined by the production of the primary commodity. Thus, the challenge to the resource classifier is to identify and consistently analyze the signficant deposit- and commodity-specific physical, technical, and economic variables that determine mineral commodity availability.

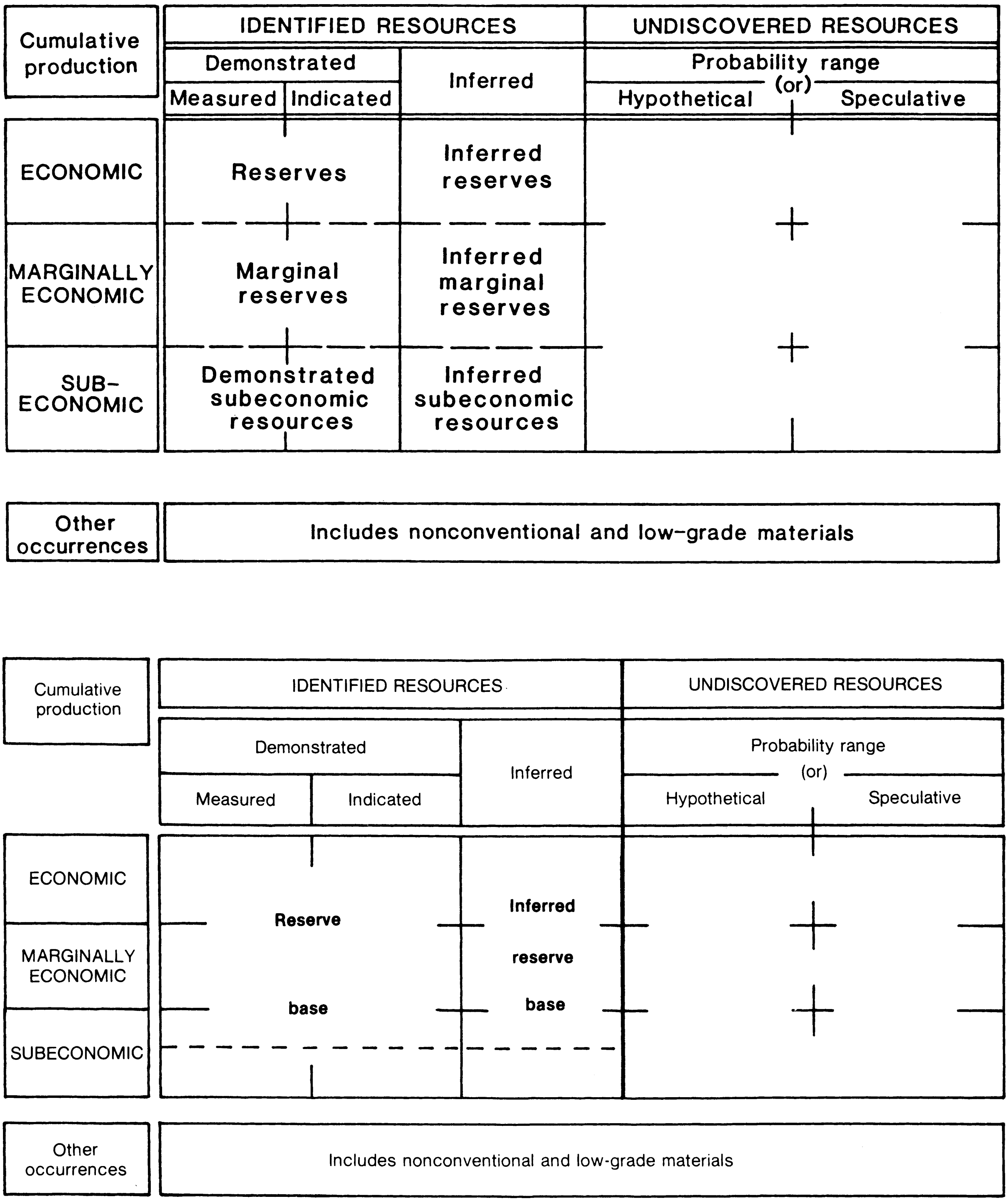

Figure 2.—Mineral resource classification diagram (*12*). Top, major elements of mineral resource classification (excluding reserve base and inferred reserve base); bottom, reserve base and inferred reserve base classification categories.

AVAILABILITY ANALYSIS APPROACH

The Bureau's availability analysis objectives are to depict the relative economic viabilities of significant MEC mineral resources via availability curves. Figure 3 is an example availability curve illustrating the estimated total quantity of recoverable material as a function of the average total production cost of each property. These curves represent commodity-wide aggregations of deposit-specific resource, cost, and economic data (*13*). Significant deposit- and commodity-dependent costs are estimated through data intensive, feasibility-type evaluation procedures assisted by the CES engineering handbook (*14-15*) and the MINSIM financial analysis program. Internal study comparability is achieved by imposing consistent data requirements, cost evaluation methods, and economic analysis assumptions on all deposits and commodities. This objective requires evaluation of all cost of mining and processing mineral raw materials to the same stage, normally a product that can be traded internationally, and adjustment of MEC production costs to a common denominator currency. The SAM is used to ensure consistent application of all these factors (*16*). The data development and analytic procedures, as well as limiting assumptions and factors inherent to availability analysis, are discussed in the following sections, which correspond to the main steps of the availability analysis workflow depicted in figure 4.

This basic approach is also used (but not described here) in engineering optimization and comparison studies for specific mines and in sensitivity analyses of the impacts of physical, technical, and economic variables on viability of individual mine properties (*17-18*).

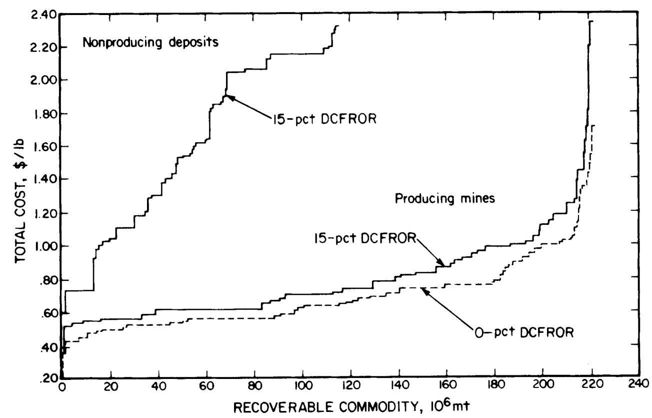

Figure 3.—Total resource availability curve.

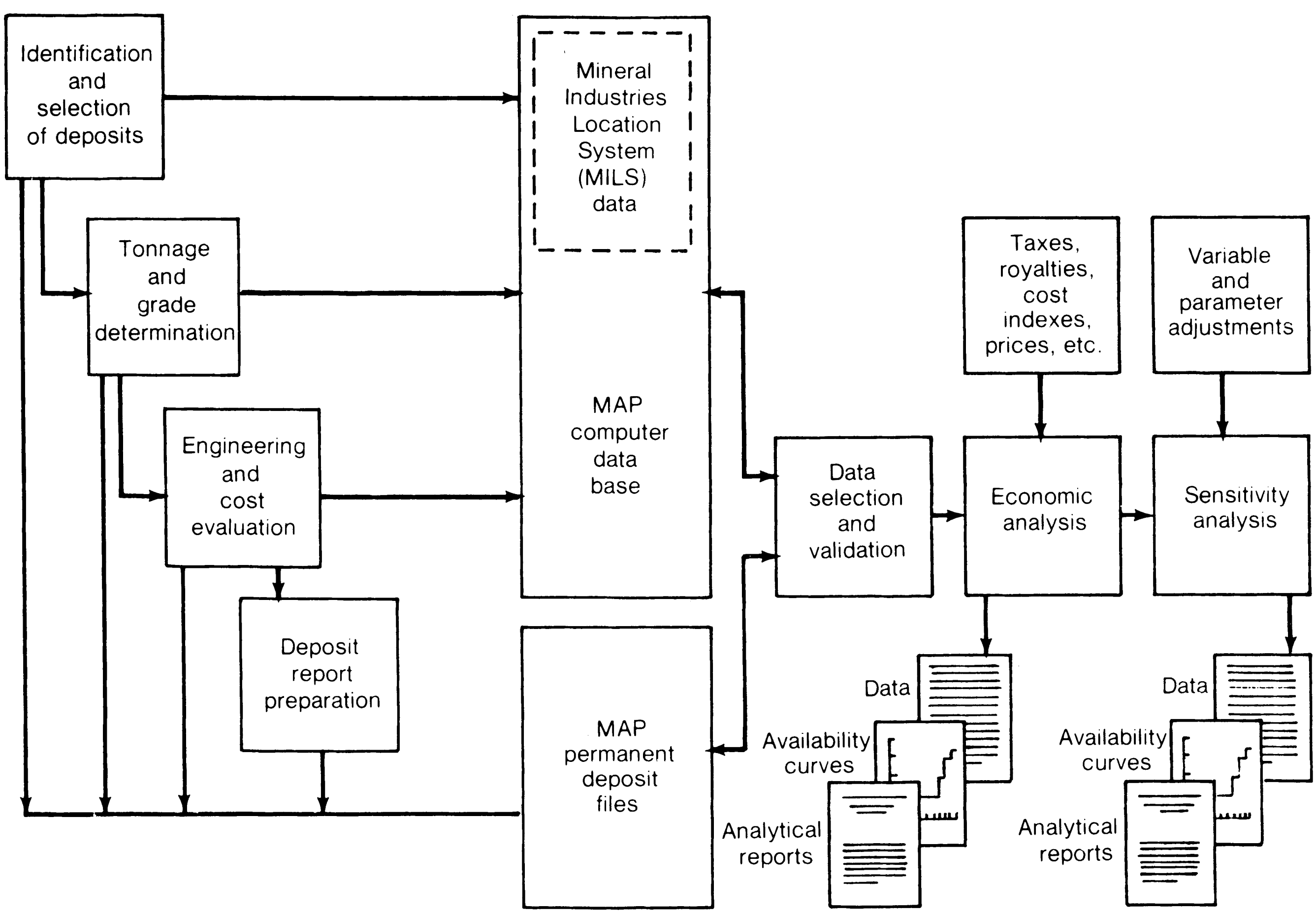

Figure 4.—MAP deposit evaluation procedure.

Identification and Selection of Commodities and Deposits

Commodities were selected for availability analysis if they presented production problems to the United States with respect to import dependency, national security, environmental sensitivity, economic competitiveness of the domestic industry, perceived scarcity, etc. The priorities of the studies were originally influenced by a special report (*19*), *Critical Imported Materials*, issued by the Council on International Economic Policy. To date, commodities are not studied if domestic resources are large and do not pose national policy concerns (e.g., sand and gravel), if they are recovered solely as byproducts in the smelting and refining stages (e.g., germanium from zinc ores), or if they fall under the jurisdiction of other agencies (e.g., uranium).

Uniform deposit selection criteria were established for all commodities to assure comparable and representative coverage. In theory, producing mines are ranked within each MEC in descending size of capacity until the cumulative capacity accounts for 85 to 90 pct of each country's capacity. All mines so identified are candidates for cost evaluations and inclusion in the availability study. Known, but nonproducing mines and mineral deposits, with resources similar to those of the producing mines, are also evaluated to indicate long-term supply potential and to facilitate analysis of resources in response to price fluctuations. In practice, deposit coverage is priority driven; it is a function of program budgets, data access, and the number and average size of deposits. Coverage of deposits tends to favor the United States because required information is more readily available and cheaper to obtain than for other MEC's. Table 1 indicates the number of deposits included in the availability studies, by commodity. A total of nearly 2,000 major mines and deposits, and approximately 900 mineral processing plants, were evaluated for these studies. Some are counted under more than one commodity because of coproduct or byproduct production.

Centrally planned economy country (CPEC)[2] supply sources are poorly represented in this first series of availability analyses because needed information is generally costly to obtain or unobtainable. Also, meaningful, all-inclusive availability analysis is less likely because CPEC mineral production decisions are more responsive to national policies of self-sufficiency than to open market economic criteria. However, the CPEC's dominant resource-producer position for some minerals, and their need for mineral export earnings, make them an important factor in world supply.

Tonnage and Grade Determination

In the availability appraisals, total resource availability is a function of the deposits selected for evaluation based on the aforementioned criteria. Demonstrated resources, which imply sufficient geological assurance to determine the quality and quantity of a resource for feasibility analysis, are depicted under the availability curves. This tends to yield conservative estimates of physical resources at many mine sites because it ignores the likelihood that inferred resources exist. In each commodity chapter of this report, availability appraisal resources are therefore compared with resources in the broader categories of reserve base and identified world resources. Estimates of resources in these categories by the Bureau and the U.S. Geological Survey are presented (*20*).

Undiscovered resources are beyond the scope of the current analyses. However, new deposits (or confirmation of inferred resources) that are revealed through exploration are evaluated as priorities and program resources allow.

Site-specific resource data are obtained from various sources, including U.S. corporate reports to the Securities and Exchange Commission, from site and company visits, and publications. The data may include proprietary information, and are frequently supplemented with data provided by other Government minerals specialists. Professional judgment is used in the evaluation and utilization of all resource data to characterize the resource according to the best information available with respect to the tonnage, grade, and distribution of the ore minerals within the deposit. All valuable minerals (primary, coproduct, and byproduct constituents) of a deposit are described to assess total value. In addition, knowledge of other important mineral deposit characteristics such as shape, dimensions, attitude (flat lying, vertical, tilted), thickness, and depth is needed to project future mining costs of producers and to design best fit mining methods for undeveloped deposits.

Table 1.—Mines and deposits included in availability appraisals, by commodity[1]

Commodify[2]	Producing mines	Deposits	Total
Aluminum	50	34	84
Antimony	10	6	16
Asbestos	28	10	38
Barite	42	22	64
Beryllium	6	7	13
Chromium	77	5	82
Cobalt	35	29	64
Columbium	3	16	19
Copper	110	131	241
Fluorspar	46	8	54
Gold	100	10	110
Graphite	19	10	29
Iron	67	82	149
Lead and zinc[3]	155	95	250
Lithium	7	9	16
Magnesium	32	6	38
Manganese	15	10	25
Mercury	6	9	15
Molybdenum	29	63	92
Nickel	45	69	114
Phosphate rock	82	124	206
Platinum	4	4	8
Potash	41	27	68
Rare earths and thorium	21	17	38
Silver	319	128	447
Sulfur	27	7	34
Tin and tantalum	129	11	140
Titanium, hafnium, and zirconium	23	38	61
Tungsten	33	22	55

[1] Some properties occur in multiple commodity chapters because of coproduct and byproduct relationships.
[2] Commodities are grouped as they are in the availability studies.
[3] Of the total number of mines, 26 are primary lead (18 producers), 186 are primary zinc (111 producers), 16 are primary copper, and 22 are primary silver.

[2] The CPEC's are Albania, Bulgaria, China, Cuba, Czechoslovakia, German Democratic Republic, Hungary, Kampuchea, Laos, Mongolia, North Korea, Poland, Romania, U.S.S.R., and Vietnam.

Engineering and Cost Evaluation

Engineering data are required to derive costs and estimate materials recovery factors and rates for each mine and deposit. Necessary data include mine and mill capacities, including announced expansions and development plans, and materials balance data, or estimates, for each recovered product. Typical smelting and refining toll charges and pay-for schedules in each of the countries or regions producing refined mineral products are also determined and assigned to appropriate mines and deposits.

For each mine and deposit included in the evaluation, capital expenditures (when applicable) are estimated for site exploration and acquisition, for stationary and mobile equipment, for development construction and engineering fees, for infrastructure, and for working capital. Infrastructure expenditures include the cost of access and haulage facilities, ports, water facilities, power supply, and personnel accommodations borne by the mine.

Direct and indirect operating costs for each mine and deposit are also developed. Direct operating costs include charges for materials, utilities, production and maintenance labor, and payroll overhead. Indirect operating costs include charges for technical and clerical labor, administrative activities, maintenance of facilities, supplies, and research. Fixed charges for local taxes, insurance, depreciation, deferred expenses, and any interest payments are also included in the analysis. Operating cost estimates do not include allowances for capital recovery (depreciation), taxes, or royalties; these costs are calculated and entered into the analyses separately.

For producing and developing mines, detailed costs are obtained from operators and/or owners, derived from asset and labor lists, or estimated. Assumptions about operating schedules, productivity, and efficiency, power and supplies consumption, and maintenance and replacement schedules for each type of equipment are incorporated. Site-specific information gathered during visits, operating experience as described in Government and public files and publications, equipment performance manuals, and data on local labor conditions all provide the basis for these assumptions.

For undeveloped deposits and for operating properties where only limited data are available, cost estimates are derived using operating parameters at mines working similar deposits in conjunction with the CES, which comprises engineering equations relating the cost of component processes (clearing, hoisting, grinding, etc.) to material throughput rates. Figure 5 illustrates the type of calculation done with the CES handbook. The curves shown relate the operating costs of drilling and blasting at a surface mine to the metric tons of ore mined per day. For instance, an operation using percussion drilling to mine ore at a rate of 1,000 mt/d (X) would incur estimated labor costs (Y_L) of \$931.74/d (or \$0.93/mt). The coefficients are averages of cost parameters found at existing operations in the United States and Canada.

Results of these calculations are modified when information on particular sites indicates the need for and size of such adjustments. Costs estimated in this way are believed to be within ±25 pct of what actual costs at that site will ultimately prove to be.

Operating cost estimates for the purpose of availability analyses are longrun averages. Operating costs are estimated to reflect the cost of mining an entire ore body(ies) and include such factors as increasing haulage distance, increasing mining depth, declining ore grade, increasing stripping ratios, etc. Because the mining costs are long run in nature, they are generally higher than current costs, reflecting an average cost over the mine life rather than a current cash cost, which may reflect the selective mining of ore in response to changing prices. To the extent that these average mining costs impact on milling costs or other downstream processing costs (due, for example, to an increase in some deleterious element) these costs reflect this longrun impact as well.

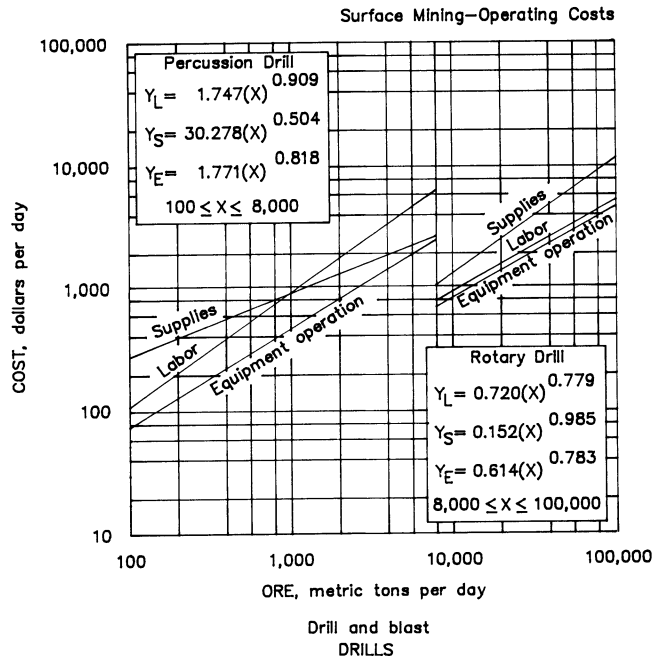

Figure 5.—Example process cost plot from CES.

Economic Analysis

Cash flow analysis is performed to aggregate capital, operating, and other costs into a total cost for each deposit. The engineering costs are reformatted into a spreadsheet (called an economic time diagram or ETD) that distributes capital and operating cost estimates over the mine's projected life. Table 2 depicts a simplified ETD.

The initial capital costs are depreciated starting in the actual investment year with the costs denominated in the dollars of the year in which the investment occurred (i.e. 1978 dollars, 1982 dollars, etc.). For purposes of financial evaluation the undepreciated balances are treated as a capital investment in the base year of the evaluation (for this report the base year is 1985). Capital reinvestments will vary according to capacity, production life, and age of the facilities. For expenses that have not yet occurred, costs are updated to constant 1985 U.S. dollars according to local currency factors and individual country inflation indexes, weighted proportionately by the impacts of such factors as labor, energy, and capital in each of the mineral industries on an individual country basis.

Table 3 illustrates the basic cash-flow computation made for each year of the mine's life. Capital, operating, and tax expenditures are subtracted from revenues derived from production of the primary and byproduct commodities. Certain deductions considered for tax computation purposes

Table 2.—Example economic time diagram (ETD), thousand dollars unless otherwise indicated

Year	Ore production, 10^3 mt	Acquisition costs	Exploration costs	Development costs	Mine-mill costs: Equipment	Mine-mill costs: Plant	Working capital	Infrastructure costs	Operating cost, $/mt ore: Mine	Operating cost, $/mt ore: Mill
1977	0	35,100	14,900							
1978	0		14,900							
1979	0		14,900							
1980	0		7,400							
1981	0			21,500		6,700		20,800		
1982	0			21,800	18,900	45,500		50,100		
1983	0			43,900	39,500	60,900		33,000		
1984	0			65,500	75,500	61,800		29,400		
1985	1,100				76,800	18,700	19,300		40.00	30.00
1986	8,300				28,100		2,100		6.00	4.00
1987	11,600				91,700		8,700		6.00	4.00
1988	13,200				192,400		15,800		8.00	5.00
1989	16,500						3,900		7.00	5.00
1990	16,500								7.00	5.00
1991	16,500				18,900				7.00	5.00
1992	16,500				39,500				7.00	5.00
1993	16,500				75,500				7.00	5.00
1994	16,500				76,800				7.00	5.00
1995	16,500				28,100				7.00	5.00
1996	16,500				91,700				7.00	5.00
1997	16,500				192,400				7.00	5.00
1998	16,500								7.00	5.00
1999	16,500								7.00	5.00
2000	16,500								7.00	5.00

Table 3.—Annual cash-flow calculation

Income	*Adjustments*
Revenues	Minus operating costs. Minus depreciation. Minus royalties. Minus property taxes.
Before tax income	Minus depletion. Minus severance taxes. Minus tax loss carryforwards.
Taxable income	Minus Federal and State income tax. Plus tax adjustments.
Net income	Plus depreciation. Plus depletion. Plus deferred deductions. Minus equity investment.

(depreciation, losses in earlier years, etc.) are added back to net income to determine annual cash-flow because they are not actual charges against current revenue.

The computed cash-flow value is used in the following formula to solve for the incentive price or the DCFROR. The formula computes the net present value (NPV) of a mining operation by discounting, at the assumed rate of return (r), and summing the annual cash-flow (C) over the currently projected life of the mine (L).

$$NPV = \sum_{n=0}^{L} C_n(e^r-1/re^{rn})$$

where C = each year's net aftertax cash-flow,
L = life of operation, in years,
n = sequential number of year being discounted,
r = annual discount rate and/or rate of return (as a decimal),
and e = 2.718281828....

After completion of the engineering cost evaluation, three unknowns remain in the formula—the price (of the principal commodity) assumed for generating revenue, the DCFROR or discount rate, and the NPV. Fixing two of the unknowns allows solving for the other.

Availability analysis is based on determining the revenue requirements to cover all expenditures made to produce the product. Technically, NPV is set equal to 0 so that the discounted sum of total revenues from the sale of the product(s) equals the discounted sum of all costs of production over the life of the operation. Availability curves are then constructed using the price determination methodology for r equal to 0 and 15 pct.

Because a maximum lifetime production quantity has been established by the mine plan adopted, finding the revenue that provides 0 NPV allows finding the average longrun price required by this mine to break even at the desired DCFROR. The MINSIM model determines this price by an iterative convergence process, rather than by inverting the NPV equation, because some of the costs (taxes and royalties) netted from revenue are a function of revenue; that is, they must be solved for simultaneously with price.

Alternatively, the discounted cash-flow or internal rate of return (r) can be found by fixing price, setting NPV equal to 0, and solving for r. In this case, the DCFROR is the rate of return that makes the present worth of cash-flows from an investment equal to the present worth of all aftertax investments, including a return of r on those investments, and can be interpreted as what the investment in this operation will yield over its projected life.

Certain assumptions are inherent in the DCFROR evaluations. Each deposit is treated as a separate corporation for purposes of tax computation so that losses generated in the early years of an operation, when depreciation is high and perhaps output is being built up to design capacity, are not treated as deductions from other sources of income. Rather, they are carried forward to later years, when positive income from this property would generate positive taxable income. Under the influence of discounting, these postponed deductions result in "smaller" (more discounted) tax costs than if they were used earlier (less discounted periods) in the mine's life. The incentive price level consequently is overestimated for those properties that in reality may be consolidated with other income producing assets for tax purposes.

Also, the DCFROR evaluations are done assuming a 0-pct rate of inflation over the life of the project to avoid the complexity of forecasting an inflation rate. The value of some components of annual cash-flow adjust with inflation (revenue and perhaps operating costs) while others (depreciation of the original cost of capital equipment) do not. Thus, constant dollar analysis, which adjusts all components of cash-flow equally for inflation each year, is an assumption that no inflation occurs over the life of the project. When positive inflation does occur over the life of an operation, actual capital deductions and tax losses and credits carried forward are relatively smaller than represented in a cash-flow calculation at a 0-pct inflation. Consequently, incentive price levels are underestimated, especially for properties that must recover large capital investments during the evaluation period.

To handle multiple product operations, price files are maintained in the SAM for all commodities that are or will be relevant to the availability analyses. All commodities recovered in the analyses are considered to be marketable. For those operations recovering byproduct commodities, market prices were established, and any resulting revenues (credits) were deducted from the total revenues required to cover all costs at the prespecified DCFROR. The revenues remaining after removal of byproduct credits are the total primary or coproduct revenues. For DCFROR analyses, these byproduct credits are added to total revenues to determine the DCFROR and NPV. The byproduct commodity prices used in this summary report are as of January 1985 and are shown in table 4.

When a price determination analysis is performed for a property designated as a coproduct deposit, the total

Table 4.—Prices of commodities evaluated in this study, including significant byproducts, January 1985 U.S. dollars

Commodity	Unit	Unit price
Aluminum	lb	0.81
Antimony	lb	2.01
Asbestos:		
2	mt	2,604.00
3	mt	1,597.00
4	mt	1,0421.00
5	mt	591.00
6	mt	418.00
7	mt	190.00
8	mt	75.00
Barite:		
Concentrate	mt	60.00
Chemical grade	mt	95.00
Mud grade	mt	63.00
Ground	mt	169.50
Beryl ore	mt	1,058.40
Beryllium	lb	313.00
Chromium:		
Concentrate	mt	40.00
Ferrochromium:		
High grade	lb	.54
Low grade	lb	.89
Metallurgical grade	mt	11.00
Refined grade	mt	60.00
Chemical grade	mt	50.00
Cobalt	lb	11.70
Columbite ore	lb	3.50
Columbium	lb	5.66
Copper	lb	.631
Fluorspar:		
Acid grade	mt	108.00
Ceramic grade	mt	103.00
Metallurgical grade	mt	80.00
Gold	tr oz	302.79
Graphite:		
Concentrate	mt	250.00
A	mt	630.00
B	mt	490.00
Lump	mt	550.00
Ilmenite:		
Domestic	mt	41.00
Foreign	mt	34.00
Iron:		
Ore	lt	54.41
Lump	lt	19.52
Pellet	lt	29.24
Pellet feed	lt	20.10
Sinter	lt	18.16
Lead	lb	0.183
Leucoxene	mt	253.00
Lithium:		
Metal	lb	22.70
Carbonate	lb	1.54
Concentrate	mt	160.00
Magnesium	lb	1.48
Manganese	lt	1.45
Mercury	flask	318.00
Molybdenum concentrate	lb	2.60
Nickel	lb	3.20
Palladium	tr oz	124.00
Phosphate	mt	20.00
Platinum	tr oz	274.00
Potash:		
KCl	mt	64.50
K_2SO_4	mt	192.90
Pyrite	mt	43.00
Rare-earth oxides	mt	400.00
Rutile:		
Concentrate	mt	364.00
Synthetic	mt	350.00
Scheelite	lb	3.68
Silver	tr oz	6.10
Sulfur	mt	142.75
Tantalum:		
Tantalite concentrate	lb	30.00
Slag	mt	6,790.00
Thorium	kg	35.58
Tin	lb	5.74
Titanium	lb	5.55
Tungsten:		
Metal	lb	13.10
APT	lb	5.55
Concentrate	lb	1.94
Yttrium concentrate	mt	385.00
Zinc	lb	.43
Zircon		
Domestic	mt	182.00
Foreign	mt	94.00

Note.—Local ore and concentrate values may vary depending on location, quality, and terms of contracts.

revenues determined are apportioned between the coproducts according to the market price differential for each product. For modeling purposes and comparison between operations, this evaluation assumes that the same relationship exists between market prices of the products and the share of average total cost of production assigned to each product.

The price proportions allow revenues to be divided between coproducts according to their relative market value, rather than assigning a price for one product and determining a price for the other. In the United States, for example, unground mud-grade barite was assigned the factor 0.60 (corresponding to the price of $60/mt), while unground chemical-filler grades were given a factor of 0.95 (price of $95/mt). This would result in revenues being split in such a fashion that the mud-grade revenues determined were approximately 63 pct (0.60/0.95) of the chemical-filler grade revenues.

The comparative production cost tables included in each summary chapter present mining, milling, and (where applicable) smelting and refining operating costs on a weighted-average country basis. Thus, one can make cross-country comparisons of these data to derive an indication of longrun "production cost" competitiveness between different MEC producers of the same mineral commodity taken to the same final product form. Also, to indicate the future resource replenishment costs, similar estimates derived for currently known but nondeveloped deposits are included for some commodities.

Availability Presentation

After the operating parameters and cost estimates are developed for each deposit, they are carefully reviewed for consistency, revised where necessary, and entered into the SAM. The SAM is an interactive computer system for analyzing the effects of various parameters on the economic evaluation of many mineral deposits. It is an integrated system composed of the MINSIM price determination and DCFROR determination algorithims, tax and country cost index files for 95 MEC's and the United States, and price files of all marketable commodities.

For each availability analysis, SAM makes the conforming transformations in the individual deposit files and simultaneously performs a DCFROR evaluation on each property. The result is a uniform estimate of the price for each property at which the primary commodity must be sold (f.o.b. the mill, smelter, or refinery) to recover all costs of production including a prespecified DCFROR on all investments.

To generate comparable values for many mines and deposits, the SAM uses indexes to adjust the cost estimates for all the mines and deposits for each commodity to a common denominator currency (the U.S. dollar) and base year (January 1985). The country cost factors adjust for country differences in cost and productivity of factor inputs. The international mining cost index system (IMCI) developed by the Bureau allows data of different time periods within a country to be updated to a common base year for analytical purposes. The IMCI is composed of 12 factor-cost indexes and an exchange rate series for the 95 MEC's and the United States denominated in local and U.S. currencies. The IMCI for each country contains 12 time-series indexes that measure the average annual rate of change in the cost of factor inputs to the mining industry within that country. Mining industry factor-cost indexes include wages, plant and equipment, construction and mining materials, electric power, transportation, and other measures that are considered to comprise the majority of capital and operating costs.

The same two DCFROR's are used for each property in the availability determination. A 0-pct DCFROR is used to estimate a minimum longrun break-even cost of established operations, it is probably low for properties that still must commit to substantial investments. A 15-pct DCFROR is generally selected as the return necessary to compensate for foregone earnings on alternative investments and for the risk inherent in developing a mineral property. This real rate of return is generally accepted as reasonable for commercial mining ventures in developed countries. It may be high for publicly owned mining projects in less developed countries where part of the return is in terms of the local employment multiplier, training and foreign exchange earning value of such projects. It may be low for commercial development in high-risk areas.

The availability of mineral commodities from a single property is a function of the average total cost of production associated with those commodities. Availability curves are constructed as aggregations of the total amount of commodities potentially available from each of the evaluated operations, ordered from the deposits having the lowest average total cost per unit of production to those having the highest. Three types of curves are usually shown, total quantity available from all resources evaluated, over the estimated life of each deposit (total availability), the capacity output available in 1 yr (1985) from producing operations (annual capacity), and the potential annual capacity available over the next 10 yr (1985-95) from producing operations.

Total availability curves depict the cumulative total amount of mineral product available at increasing average total costs of production. They can be interpreted by comparing the 1985 market price to the total cost of production, and relating this to the cumulative amount of mineral product available. Long-term resource availability positions and comparative cost levels between countries can be ascertained by studying the availability curves of each summary chapter.

Figure 3 is an example of a total availability curve. The vertical axis depicts cost (equal to long-term constant dollar price) estimates per unit of product and the horizontal axis relates these cost or price points to the cumulative total amount available. Because the total availability curves do not include all known demonstrated resources, or inferred resources, the reader should not interpret the curves as representing a precise boundary of resource availability, but rather as a boundary drawn with a degree of confidence that is driven by available data on known demonstrated resources and their costs of production. Clearly the availability curves "shade out" into an area of increasing resource availability *but* at decreasing levels of confidence as to quantity and cost. The example curve is "shaded" to convey this qualitative assessment or additional resource availability. In the commodity chapters, curves are constructed for producing operations and/or, where applicable, for nondeveloped deposits and nonproducing mines.

Annual capacity curves (fig. 6) illustrate the aggregated potential annual production that could be provided in 1985 from the evaluated deposits. Because the cash-flow procedure from which the site cost estimates are derived is a longrun feasibility methodology, the DCFROR annual curves may overstate the cost necessary to make the depicted quantities available in the current year (i.e., supply). Consequently, annual availability is depicted with a band of costs for each quantity. The upper line in figure 6 connects the 0-pct DCFROR value (price necessary for recovery of capital, operating costs, and taxes) for each property and the lower line connects the operating costs for each property arranged in ascending order.

Operating cost does not accurately reflect this concept when temporary cost reduction activities such as selective mining are practiced, nondiscretionary (take or pay) contracts exist, and when other costs such as equipment replacement are unavoidable. The operating cost approximates the amount that could be saved by shutting down or the minimum revenue the property must generate to continue operating. Because some of the accounting costs included in the 0-pct DCFROR value can be postponed (capital recovery, development, some interest payments), the minimum price the property must receive to operate for the period covered is below the upper line. Hence, the annual supply potential of the commodity represented, from the properties evaluated, is between these upper and lower bounds. For instance, at $0.60/lb between 2.5 to 3.5 million mt could be made available in 1985.

Whereas the total availability curves are constructed to depict a stock of available resource, the annual availability curve depicts a potential full capacity "flow" of mineral product that is available over a range of years (1985-95) from known and evaluated demonstrated resources at average total costs of production based upon current technology. These curves depict total availability of mineral product for each year from 1985 to 1995, based on the assumed production schedules for producing operations.

Certain assumptions are inherent in both types of annual curves. First, all deposits produce at the estimated or proposed capacity used in the summary analyses throughout the life of the deposit. Second, each operation will be able to sell all of its primary or coproduct output at the determined average total cost and obtain at least the minimum specified DCFROR. Third, byproducts are considered to be sold at the prices listed in table 3.

Figure 7 is an example of an annual availability curve. It depicts aggregated annual production potential (capacity) from producing operations at selected average total costs of production over the period 1985 through 1995. The rate at which the curves decline and the total reduction in available mineral product from evaluated demonstrated resources provides a quantitative assessment of resource replenishment requirements. Any resource depletion indicated creates the incentive for extending demonstrated resources, developing new capacity, discovering new resources, installing improved recovery technology at existing mines, and/or bringing new deposits into production. Declines over time are best viewed as indicators of how much and how rapidly resource and capacity augmentation efforts are required or where they may most likely by profitable.

In summary, the total resource availability curves presented in this Bulletin are intended to convey the amount and cost (as of January 1985) of current demonstrated, extractable resources from significant MEC producing and developing mines and, as yet undeveloped, deposits. Annual capacity curves for 1985 indicate the same information for 1 yr rather than for the life of the evaluated deposits. Annual capacity curves, 1985 to 1995, provide a quantitative assessment of the adequacy of current demonstrated resources and annual production potential. The commodity chapters provide an overview of MEC production and consumption statistics with which to compare the resource availability data.

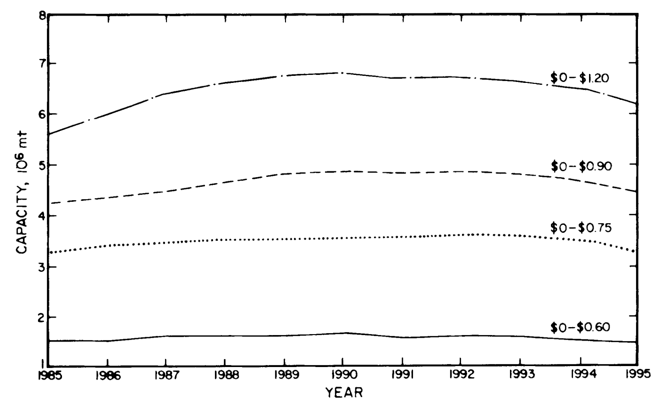

Figure 6.—Annual resource availability curve.

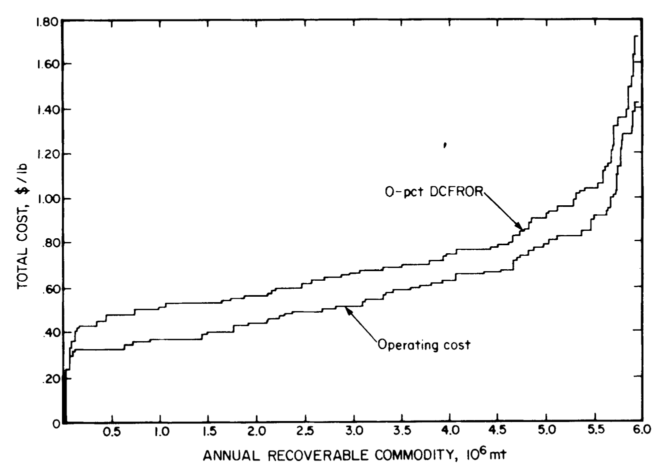

Figure 7.—Potential annual production from producing mines.

BASIC OUTLINE OF COMMODITY CHAPTERS

Commodity chapters are structured to follow the same general outline, with additions or deletions as may be necessary for unique circumstances. The general chapter outline is as follows:

Introduction
Geology and resources
  Geology
  Resources
U.S. and world historical production
Extraction and processing technology
  Mining
  Processing
Production costs
Availability
  Total recoverable
  Annual capacity
Conclusions
References

REFERENCES

1. Kingston, G. A., F. V. Carrillo, J. J. Gray, and P. McIlroy. Availability of U.S. Primary Nickel Resources. BuMines IC 8469, 1970, 57 pp.
2. Bennett, H. J., L. Moore, L. E. Welborn, and J. E. Toland. An Economic Appraisal of the Supply of Copper From Primary Domestic Sources. BuMines IC 8598, 1973, 156 pp.
3. Larson, L. P., R. L. Lowrie, and G. R. Leland. Availability of Tungsten at Various Prices From Resources in the United States. BuMines IC 8500, 1971, 65 pp.
4. McIlroy, P., J. S. Coffman, W. L. Rice and M. H. Hibpshman. Availability of U.S. Primary Lead Resources. BuMines IC 8646, 1974, 60 pp.
5. U.S. Bureau of Mines. The Bureau of Mines Minerals Availability System and Resource Classification Manual. IC 8654, 1974, 199 pp.
6. Kingston, G. Reserve Classification of Identified Nonfuel Mineral Resources by the Bureau of Mines Minerals Availability System. Mathematical Geol., v. 9, No. 3, 1977, pp. 273-279.
7. Bennett, H. J., J. G. Thompson, H. J. Quiring, and J. E. Toland. Financial Evaluation of Mineral Deposits Using Sensitivity and Probabilistic Analysis Methods. BuMines IC 8495, 1970, 82 pp.
8. Berg, A. W., and F. V. Carrillo. MILS: The Mineral Industry Location System of the Federal Bureau of Mines. BuMines IC 8815, 1980, 24 pp.
9. U.S. Government Accounting Office. The Department of the Interior's Minerals Availability System. EMD-78-16, July 17, 1978, 57 pp.
10. National Research Council. A Review of the Statistical Program of the Bureau of Mines. NAS, 1982, 209 pp.
11. U.S. Department of the Interior. Mineral Information Systems Review. An Internal Report by the Office of Minerals Policy and Research Analysis. Jan. 2, 1982, 14 pp.
12. U.S. Bureau of Mines and U.S. Geological Survey. Principles of a Resource/Reserve Classification for Minerals. U.S. Geol. Surv. Circ. 831, 1980, 5 pp.
13. Babitzke, H. R., A. F. Barsotti, J. S. Coffman, J. G. Thompson, and H. J. Bennett. The Bureau of Mines Minerals Availability System: An Update of Information Circular 8654. BuMines IC 8887, 1982, 54 pp.
14. U.S. Bureau of Mines. Bureau of Mines Cost Estimating System Handbook. (In Two Parts.) 1. Surface and Underground Mining. IC 9142, 1987, 631 pp.
15. ______. Bureau of Mines Cost Estimating System Handbook. (In Two Parts.) 2. Mineral Processing. IC 9143, 1987, 566 pp.
16. Davidoff, R. L. Supply Analysis Model (SAM): A Minerals Availability System Methodology. BuMines IC 8820, 1980, 45 pp.
17. Davidoff, R. L., and R. J. Hurdelbrink. Taxation and the Profitability of Mineral Operations in Seven Mountain States and Wisconsin: A Hypothetical Study. BuMines Mineral Issues, 1983, 32 pp.
18. U.S. Department of Commerce, International Trade Administration, and U.S. Department of Interior, Bureau of Mines. A Cost Comparison of Selected U.S. and Colombian Coal Mines. 1986, 62, pp.
19. Council on International Economic Policy (CIEP). Special Report, Critical Imported Materials. 1974, 110 pp.
20. U.S. Geological Survey. United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, 722 pp.

ALUMINUM

INTRODUCTION

After silicon, aluminum is the most abundant metal in the Earth's crust, averaging more than 8 pct. Despite its abundance, aluminum is a relatively new metal that has been produced in commercial quantities for less than 100 yr. It weighs about one-third as much as steel or copper, has excellent corrosion resistance, is highly malleable and ductile, and is easily machined and cast. Aluminum is important in virtually all segments of the world economy and its use, measured in either quantity or value, now exceeds that of any other metal except iron.

The aluminum industry consumes nearly 90 pct of the bauxite mined; the remainder is used as abrasives, chemicals, refractories, and miscellaneous minor products. Packaging accounted for an estimated 29 pct of domestic consumption in 1985; building, 22 pct; transportation, 21 pct; electrical, 10 pct; consumer durables, 7 pct; machinery and equipment, 6 pct; and other uses, 5 pct (*1*).[1]

In its natural state, aluminum is highly reactive and occurs only in compound forms such as hydrated oxides and silicates. From the time bauxite was first mined in the mid-1800's, it has been the predominant raw material for the production of alumina which, in turn, may be reduced into aluminum. This study analyzes the availability of aluminum from the mining of bauxite to the production of primary aluminum.

Most of the information presented in this chapter is updated from Bureau of Mines Information Circular 8917 "Aluminum Availability—Market Economy Countries, A Minerals Availability Program Appraisal" (*2*). Demonstrated resources, production status, and production costs for each operation were updated to January 1985 values. Production data and information on changes in the industry were extracted from the aluminum, and bauxite and alumina chapters of recent editions of the Bureau's Minerals Yearbooks.

Information on the potential availability of alumina from domestic resources, including alumina from alunites, clays, and ferruginous bauxites, is available elsewhere (*3*).

GEOLOGY AND RESOURCES

GEOLOGY

Bauxite is composed of aluminum hydroxide minerals with impurities of free silica, clay, silt, and iron hydroxides (*4*). The types of bauxite are (1) trihydrate, consisting mainly of gibbsite, $Al_2O_3 \cdot H_2O$; (2) monohydrate, consisting mainly of boehmite, AlO(OH), or diaspore, $Al_2O_3 \cdot H_2O$; and (3) mixed bauxite, consisting of both gibbsite and boehmite. Bauxite is believed to be formed as a residual soil in humid, tropical, or subtropical regions with good drainage. Extreme weathering conditions common to tropical climates decompose the iron and aluminum silicates and water percolating downward leaches the silica and other elements. Bauxite deposits typically assay 28 to 55 pct Al_2O_3. Deposits of bauxite are widespread globally, although the major deposits are confined to a belt extending 20° north and south of the equator.

RESOURCES

For the bauxite deposits analyzed in this report, tonnage estimates were made at the demonstrated resource level according to the mineral resource classification system developed jointly by the U.S. Geological Survey and the Bureau (*5*). The demonstrated resource category includes measured plus indicated tonnages.

The world bauxite reserve base amounts to 23.2 billion mt of bauxite ore, of which about 22 billion mt is in market economy countries (MEC's). The latest reserve base estimate is 950 million mt greater than the reserve base estimates over the past 4 yr owing to increased resource estimates for Brazil (600 million mt increase) and other MEC's (350 million mt increase). In addition, total world bauxite resources (identified plus subeconomic and undiscovered resources) amount to 55 to 70 billion mt (*6*). The International Bauxite Association (IBA) has reported a more optimistic total world resource figure of 103.4 billion mt (*7*). Regardless of the source of data, worldwide bauxite resources are extensive and will likely increase in the future as a result of exploration activities. Additional bauxite resource information is available in a recent U.S. Geological Survey publication (*8*).

This study is based on the availability of aluminum from 84 bauxite properties in 20 countries, which have demonstrated bauxite resources of 19.7 billion mt, accounting for almost 90 pct of the reserve base for MEC's. Eighteen mines and deposits in Yugoslavia were not included in this study because cost data are not sufficiently reliable to be included with properties from the other MEC's.

[1]Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

Estimates of total bauxite resources in MEC's that were evaluated for this study and their subsequent recoverable aluminum metal tonnages are shown in table 1, and information on the individual properties included in the analysis is presented in table 2.

Estimates of world resources, the bauxite reserve base, and the demonstrated resources used in this study are illustrated in figure 1.

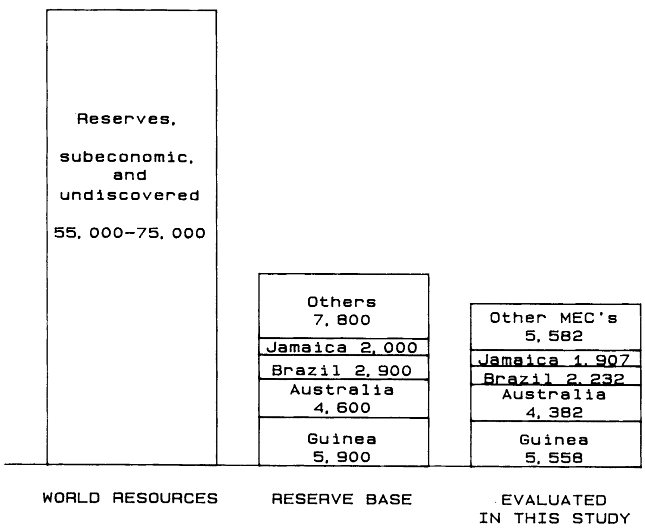

Figure 1.—Estimates of world bauxite resources (million metric tons).

Table 1.—Summary of MEC demonstrated bauxite resources evaluated for this study, as of January 1985

Country	Number of deposits	In situ resource, 10^3 mt	In situ resource, Percent of total	Recoverable aluminum metal, 10^3 mt	Recoverable aluminum metal, Percent of total
Caribbean: Jamaica	15	1,907,121	9.70	374,300	10.58
United States	1	15,975	.08	3,597	.10
South and Central America:					
Brazil	7	2,232,107	11.35	332,776	9.40
Costa Rica	1	78,500	.40	10,425	.29
French Guiana	1	42,000	.21	9,333	.26
Guyana	11	493,101	2.51	98,827	2.79
Suriname	4	533,206	2.71	120,851	3.41
Venezuela	1	236,000	1.20	58,153	1.64
Total	25	3,614,914	18.38	630,365	17.79
Africa:					
Cameroon	1	800,000	4.07	145,449	4.10
Ghana	3	556,443	2.83	118,736	3.35
Guinea	6	5,558,442	28.27	1,052,109	29.73
Malawi	1	28,760	.15	5,668	.16
Sierra Leone	2	155,752	.79	32,576	.92
Total	13	7,099,397	36.11	1,354,538	38.27
Europe:					
France	3	25,375	.13	1,805	.05
Greece	4	579,849	2.95	130,973	3.70
Turkey	1	13,150	.07	3,398	.10
Total	8	618,374	3.15	136,176	3.85
Asia:					
India	8	1,172,899	5.97	197,726	5.59
Indonesia	2	799,500	4.07	142,077	4.01
Total	10	1,972,399	10.04	339,803	9.60
Oceania:					
Australia	10	4,382,151	22.28	690,488	19.51
Solomon Islands	2	50,300	.26	9,969	.28
Total	12	4,432,451	22.54	700.457	19.79
Grand total	84	19,660,631	100.00	3,539,236	100.00

NOTE.—Resource figures, by country, have been rounded to the nearest thousand metric ton.

Table 2.—List of MEC bauxite properties included in this study

Country and property name	Ownership	Current status[1]	Type[2]	Estimated stripping ratio (waste to ore)	Type of ore
Australia:					
Aurukun	Billiton-Pechiney	N	S	0.1:1-1:1	Mixed, 53 pct Al_2O_3, 6 to 8 pct silica.
Cape Bougainville	Comalco-Alcoa-Billiton	N	S	0.13:1	Gibbsite, some boehmite, 36 pct Al_2O_3, 1.9 pct silica.
Chittering (Muchea)	Comalco	N	S	0.33:1	Gibbsite, some boehmite, 30 to 35 pct Al_2O_3, 1 to 2 pct reactive silica.
Gove	Swiss Aluminum Australia Ltd., Gove Alumina Ltd.	P	S	0.29:1	Gibbsite, some boehmite, 50.4 pct Al_2O_3, 2 to 6.5 pct silica.
Huntly-Del Park	Alcoa of Australia	P	S	0.13:1	Gibbsite, some boehmite, 32.5 pct Al_2O_3, 15 to 22 pct silica, 1 to 2 pct reactive silica.
Jarrahdale	..do	P	S	0.13:1	Gibbsite, some boehmite, 32.5 pct Al_2O_3, 15 to 22 pct silica, 2 to 3 pct reactive silica.
Mitchell Plateau	Comalco-Alcoa-Billiton	N	S	0.13:1	Gibbsite, some boehmite, 47 pct Al_2O_3, 2 to 3 pct silica.
Willowdale	Alcoa of Australia	P	S	0.13:1	Gibbsite, some boehmite, 32.5 pct Al_2O_3, 15 to 22 pct silica, 1 to 2 pct reactive silica.
Weipa-Andoom	Comalco	P	S	0.31:1	Mixed, mainly gibbsite, 48 to 56 pct Al_2O_3, 4.5 to 9 pct silica.
Mount Saddleback	Reynolds-Shell-BHP Ltd-Kobe Alumina.	P	S	0.33:1	Pistolitic gibbsite, 32.2 pct Al_2O_3, 15 to 22 pct silica, 1 to 2 pct reactive silica.

See explanatory notes at end of table.

Table 2.—List of MEC bauxite properties included in this study—Continued

Country and property name	Ownership	Current status[1]	Type[2]	Estimated stripping ratio (waste to ore)	Type of ore
Brazil:					
Almeirim, Jutai	Companhia Vale do Rio Doce	N	S	3:1	Gibbsite, 51.6 to 57.5 pct Al_2O_3, low silica.
Ouro Preto	Alcan	P	S	0.2:1	Gibbsite, 37 pct Al_2O_3 in situ, 47 pct washed, 2.23 pct reactive silica.
Paragominas	Mineracao Vera Cruz S.A.	N	S	5:1-7:1	Gibbsite, 53 to 58 pct Al_2O_3, 3 to 7 pct silica.
Poco de Caldas-Alcominas.	Alcoa-Hanna-State Minas Gerais.	P	S	0.1:1	Gibbsite, 46 to 54 pct Al_2O_3, 5 to 6 pct silica.
Pocos de Caldas	Companhia Brasileira do Alum	P	S	0.1:1	Gibbsite, 46 to 54 pct Al_2O_3, 5 to 6 pct silica.
Trombetas	Mineracao Rio do Norte	P	S	1:1	Gibbsite, 50 pct Al_2O_3, 4 pct reactive silica.
Trombetas, Alcoa	Alcoa-Billiton	N	S	1:1	Gibbsite, 50 pct Al_2O_3, 4 pct reactive silica.
Solomon Islands:					
Rennell Island	Mitsui Mining and Smelting Co.	N	S	0.4:1	Gibbsite, 48 pct Al_2O_3, low silica.
Wagina Island	Pacific Aluminum-CRA Exploration.	N	S	0.1:1	Gibbsite, 47.1 pct Al_2O_3, low silica.
Cameroon: Minim Martap.	Cameroon Government-Pechiney	N	S	0.33:1-1:1	Gibbsite, 42 pct Al_2O_3, 3 pct silica.
Costa Rica: San Isidro del General.	Costa Rica Government	N	S	0.2:1	Gibbsite, some boehmite, 33.8 pct available Al_2O_3, 7.5 pct reactive silica.
France:					
Blanquette-Combecave	Bauxites et Alumines de Provence.	P	U	NA	Boehmite and diaspore, 54 pct Al_2O_3.
Canonnettes	Aluminum Pechiney	P	U	NA	Boehmite and diaspore, 50 pct Al_2O_3, 6.8 pct silica.
Peygros (Var)	..do	P	U	NA	Boehmite and diaspore, 50 pct Al_2O_3, 7 pct silica.
French Guiana: Kaw Mountains.	Cie. Miniere de Guyane	N	S	0.1:1	Gibbsite, 42 pct Al_2O_3, 2 pct silica.
Ghana:					
Awaso	Ghana Government-British Alcan	P	S	1:1	Gibbsite, some boehmite, 48.7 pct Al_2O_3, 1.7 pct silica.
Kibi (Atewa)	Ghana Government	N	S	0.63:1	Gibbsite, 44 pct Al_2O_3, 3 pct silica.
Nyinahin	..do	N	S	0.1:1	Gibbsite and boehmite (5:1), 40 pct Al_2O_3, 3.2 pct silica.
Greece:					
Eleusis	Eleusis Bauxite Mining Co.	P	U	NA	Boehmite, 51 pct Al_2O_3, 0.7 to 2 pct silica.
Helikon	Greek Helikon Bauxites Co.	P	U	NA	Boehmite and diaspore, 60 pct Al_2O_3, 4 pct silica.
Itea	Eleusis Bauxite-Skalistiri	P	U	NA	Boehmite and diaspore, 60 pct Al_2O_3, 4 pct silica.
Parnasse	Bauxite Parnasse Mining Co.	P	S	3:1	Boehmite and diaspore, 57.5 pct Al_2O_3, 2 pct silica.
Guinea:					
Aye-Koye	Guinea Government	N	S	0:1	Gibbsite, 49 pct Al_2O_3.
Dabola	Guinea Government-Bauxite de Dabola.	N	S	0:1	Gibbsite and boehmite (9:1), 45 pct Al_2O_3, 1.33 pct silica.
Fria	Frialco-Guinea Government	P	S	0:1	Gibbsite, 48 pct Al_2O_3, 2.5 pct silica.
Kindia	Guinea Government	P	S	0:1	Gibbsite, some boehmite, 48 pct Al_2O_3, 1 to 3 pct silica.
Sangaredi	Halco Inc.-Guinea Government	P	S	0:1	Gibbsite, some boehmite, 48 to 60 pct Al_2O_3, about 4 pct silica.
Tougue	Guinea Government-Alusuisse	N	S	0:1	Gibbsite, 43 pct Al_2O_3, 3.8 pct silica.
Guyana:					
Arrowcane East	Guyana Government	P	S	9:1	Gibbsite, some boehmite, 59 pct Al_2O_3, 1 to 10 pct silica.
Arrowcane South	..do	P	S	9:1	Gibbsite, some boehmite, 59 pct Al_2O_3, 1 to 10 pct silica.
Coomacka	..do	P	S	9:1	Gibbsite, some boehmite, 59 pct Al_2O_3, 1 to 10 pct silica.
East Mombaka	..do	P	S	9:1	Gibbsite, some boehmite, 59 pct Al_2O_3, 8.3 pct silica.
East Montgomery	..do	P	S	9:1	Gibbsite, some boehmite, 59 pct Al_2O_3, 1 to 10 pct silica.
Ituni District	..do	P	S	9:1	Gibbsite, some boehmite (20:1), 50 to 63 pct Al_2O_3, 8.3 pct silica.
Kara-Kara	..do	P	S	9:1	Gibbsite, some boehmite, 59 pct Al_2O_3, 1 to 10 pct silica.
Manaka	..do	P	S	9:1	Gibbsite, some boehmite, 59 pct Al_2O_3, 8.3 pct silica.
West Bank #3	..do	P	S	9:1	Gibbsite, some boehmite, 59 pct Al_2O_3, 1 to 10 pct silica.
West Mombaka	..do	P	S	9:1	Gibbsite, some boehmite, 59 pct Al_2O_3, 8.3 pct silica.
Yararibo	..do	P	S	9:1	Gibbsite, some boehmite, 59 pct Al_2O_3, 1 to 10 pct silica.

See explanatory notes at end of table.

Table 2.—List of MEC bauxite properties included in this study—Continued

Country and property name	Ownership	Current status[1]	Type[2]	Estimated stripping ratio (waste to ore)	Type of ore
India:					
Amarkantak	Bharat Aluminum Co. Ltd.	P	S	4:1	Mixed, 37 to 54 pct Al_2O_3, 2 to 7 pct silica.
Anantagiri	..do	N	S	0:1	Gibbsite and hematite, 48 pct Al_2O_3, 2.5 pct silica.
Bagru Hills	Indalco	P	S	1.1:1	Gibbsite, 53.2 pct Al_2O_3, 3.3 pct silica.
Chandgad	..do	P	S	1.5:1	Gibbsite and hematite, 48.6 pct Al_2O_3, 3.2 pct silica.
Chintaplee	Bharat Aluminum Co. Ltd.	N	S	0.11:1	Gibbsite, 46 pct Al_2O_3, 2 pct silica:
Gandhamardan	..do	N	S	NA	Gibbsite, 47 pct Al_2O_3, 2.6 pct silica.
Lohardaga (Hindalco)	Hindustan Aluminum Co	P	S	3:1	Gibbsite, some boehmite, 45 pct Al_2O_3, 3 pct silica.
Panchpatmali-Orissa	Bharat Aluminum Co. Ltd.	N	S	0.4:1	Gibbsite, 42 to 56 pct Al_2O_3, 1 to 3 pct silica.
Indonesia:					
Bintan Island	P.T. Aneka Tambang	P	S	0.1:1	Gibbsite, 45 to 50 pct Al_2O_3, 8 to 13 pct silica.
Singakawang	..do	N	S	0.1:1	Gibbsite, 54 pct Al_2O_3, 5 pct silica (washed).
Jamaica:					
Alpart	Kaiser-Reynolds	P	S	0.125:1	Gibbsite, some boehmite, 42 to 50 pct Al_2O_3, 1 to 20 pct silica.
Breadnut Valley	Alcoa-Jamaica Government	P	S	0:1	Gibbsite, some boehmite, 43.5 pct Al_2O_3, 1.5 pct silica.
Cambridge	Jamaica Government	N	S	0.125:1	Gibbsite, some boehmite, 43 pct Al_2O_3.
Ewarton	Alcan-Jamaica Government	P	S	1:1	Gibbsite, some boehmite, 43 pct Al_2O_3, 1.8 pct silica.
Kirkvine	..do	P	S	0:1	Gibbsite, some boehmite, 45 pct Al_2O_3, 1 pct silica.
Lydford	Reynolds-Jamaica Government	T	S	0:1	Gibbsite, some boehmite, 42.5 pct Al_2O_3, 1 pct silica.
Maggotty	Jamaica Government	P	S	0.1:1	Gibbsite, some boehmite, 43 pct Al_2O_3.
New Market East	..do	N	S	0.1:1	Gibbsite, some boehmite, 42 pct Al_2O_3.
Samalco	..do	N	S	0.4:1	Gibbsite, some boehmite, 44 pct Al_2O_3.
Schwallenburgh West	..do	N	S	0:1	Gibbsite, some boehmite, 42 pct Al_2O_3.
Spanish Town West	..do	N	S	0:1	Gibbsite, some boehmite, 42 pct Al_2O_3.
Trelawny	..do	N	S	0:1	Gibbsite, some boehmite, 42 pct Al_2O_3, less than 3 pct silica.
Trelawny Central	..do	N	S	0.1:1	Gibbsite, some boehmite, 43 pct Al_2O_3.
Water Valley	Kaiser Jamaica Bauxite Co.	P	S	0.1:1	Gibbsite, some boehmite, 45 pct Al_2O_3.
Williamsfield East	Jamaica Government	N	S	0:1	Gibbsite, some boehmite, 43 pct Al_2O_3, less than 3 pct silica.
Malawi: Mulanje Mountain	Malawi Government	N	S	0:1	Gibbsite, 43.9 pct Al_2O_3, 2.2 pct reactive silica.
Sierra Leone:					
Moyamba	Sierra Leone Government—Alusuisse.	P	S	0:1	Gibbsite, 47 pct Al_2O_3, 4.5 pct silica.
Port Loko	..do	N	S	0.5:1	Gibbsite, 49 pc Al_2O_3, 4 pct silica.
Suriname:					
Bakhuis Mountains	N.V. Grassalco (Government)	N	S	0.2:1	Gibbsite, 48.4 pct Al_2O_3, 3.1 pct silica.
Moengo	Alcoa	P	S	1:1	Gibbsite, 54 pct Al_2O_3.
Onverdacht	Billiton-Alcoa	P	S	1.7:1	Gibbsite, 52 pct Al_2O_3, 4.5 pct reactive silica.
Suralco (Lelydorp)	..do	T	S	NA	NA.
Turkey: Seydisehir	Etibank	P	S	1:1-3:1	Boehmite, 56 to 59 pct Al_2O_3, 7 to 8 pct silica.
United States: Alcoa Bauxite	Alcoa	P	S	6.1:1	Gibbsite, some boehmite, 50 pct Al_2O_3, 13 pct silica.
Venezuela: Los Pijiguaos	Industria Venezolana de Aluminio.	N	S	0.05:1	Gibbsite, 49 pct Al_2O_3, 9.3 pct silica (2.2 pct reactive silica).

NA Not available. [1]P, producing; N, nonproducing; T, temporarily shut down. [2]S, surface; U, underground.

U.S. AND WORLD HISTORICAL PRODUCTION

Bauxite production in 1985 was estimated to be 85.1 million mt from 23 countries (*6*). Australia was the dominant bauxite producing country with 1985 production of 32.4 million mt, accounting for 38 pct of total world production. The United States produced only 0.565 million mt of bauxite in 1985, most of which went to nonmetal uses.

Primary aluminum production in 1985 was estimated to be 15.3 million mt from 43 countries (*1*). Approximately 80 pct of this production was from MEC's and 20 pct was from centrally planned economy countries (CPEC's). The United States remained the largest producer with production of 3.5 million mt, although 1985 production was 0.6 million mt less than production of 4.1 million mt in 1984. By the end of 1985, U.S. producers were only operating at 63 pct of capacity. As a result of high operating costs and increasing foreign competition, a larger percentage of U.S. aluminum metal will come from imports and recovery of metal from scrap. Bauxite, alumina, and aluminum production data, from 1950 through 1985, are shown graphically in figures 2 through 4.

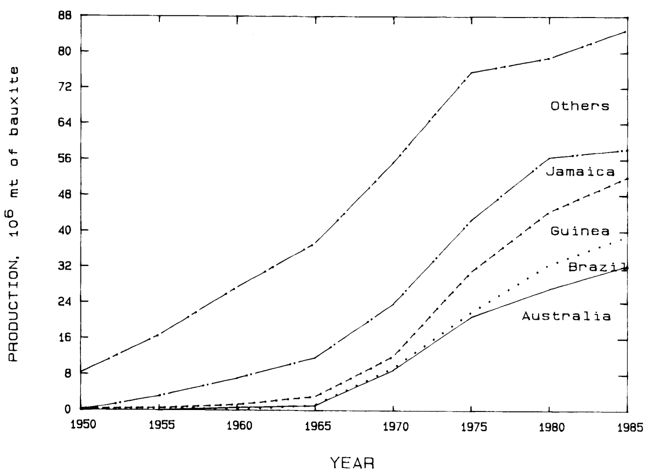

Figure 2.—World bauxite production, 1950-1985.

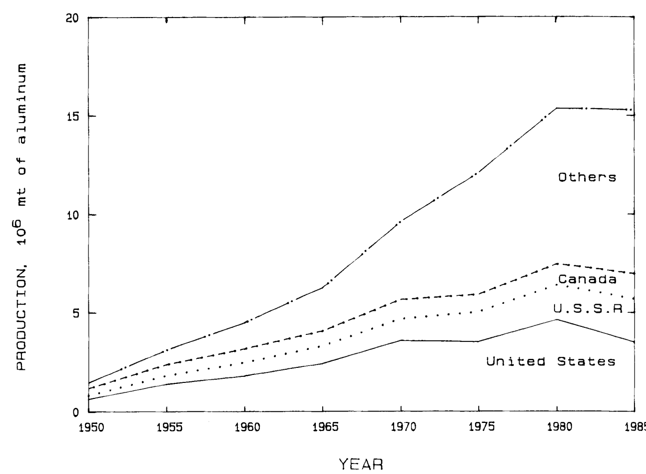

Figure 4.—World primary aluminum production, 1950-85.

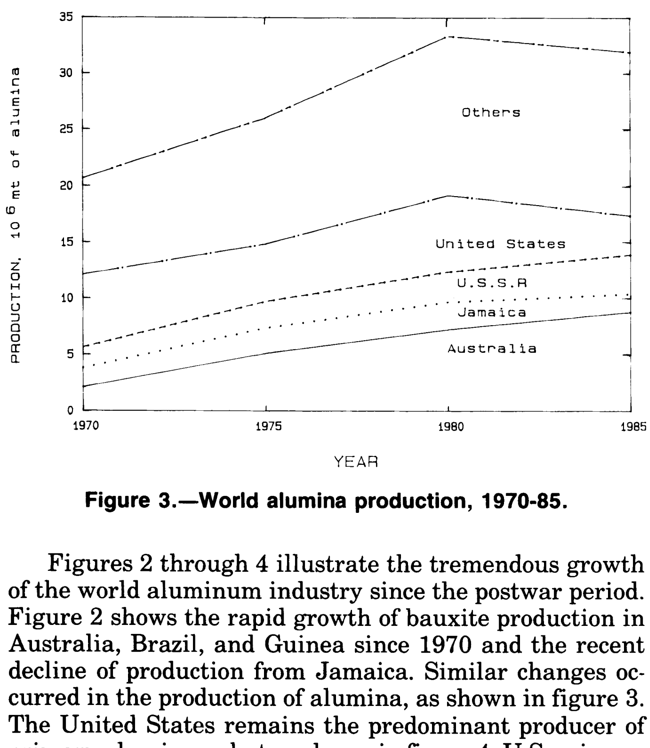

Figure 3.—World alumina production, 1970-85.

Figures 2 through 4 illustrate the tremendous growth of the world aluminum industry since the postwar period. Figure 2 shows the rapid growth of bauxite production in Australia, Brazil, and Guinea since 1970 and the recent decline of production from Jamaica. Similar changes occurred in the production of alumina, as shown in figure 3. The United States remains the predominant producer of primary aluminum, but as shown in figure 4, U.S. primary aluminum production is decreasing in the face of foreign competition.

A number of U.S. Bayer alumina refineries shut down permanently during the early 1980's: Alcoa's Mobile, AL, plant shut down in 1982; Kaiser permanently shut down its operations at Hurricane Creek, AR, along with all bauxite mines and other facilities during the last quarter of 1984. The Martin Marietta refinery in St. Croix, Virgin Islands, was closed in mid-1985 owing to high production costs and is currently offered for sale along with the smelter at The Dalles, OR, which closed in 1984.

Several other aluminum smelters in the United States also shut down permanently during the early 1980's: Alcoa's Point Comfort, TX, smelter closed permanently in 1982; the experimental Alcoa smelter in Palestine, TX, shut down indefinitely in mid-1982 pending technical evaluation of the chloride bath process; Conalco's Lake Charles, LA, smelter closed in January 1983; Kaiser Aluminum closed its Chalmette, LA, smelter in January 1983; Revere Copper & Brass closed its smelter in Scottsboro, AL, in mid-1982; and Reynolds permanently closed its smelter at San Patricio, TX, in early 1984.

In recent years, there has been a continuing movement of smelting capacity away from aluminum's traditional fabrication centers and increasing competition among aluminum producers at a time of slower growth rates in consumption. The United States and Japan have suffered the most from the poor price performance of aluminum during the 1980's. Besides the U.S. plant closures mentioned, the Japanese have closed 12 smelters since 1981, leaving only five active aluminum smelters. Japanese production capacity reached a peak of 1.6 million mt in 1978 with only 354,000 mt of capacity remaining in 1986.

To replace domestic smelting capacity, Japanese companies have invested heavily overseas; e.g., investments in Indonesia's Asahan smelter, the Venalum smelter in Venezuela, the Boyne smelter in Australia, and the Albras smelter in Brazil (*9*).

Annual smelter capacity in MEC's has been reduced by some 1.5 million mt since 1984 as marginal producers have shut down. This readjustment of the industry has provided the benefit of increased capacity utilization rates of the surviving smelters, although recent cutbacks in U.S. capacity have been matched by new capacity from new producers in other countries.

In Australia, two new bauxite mines (Mt. Saddleback and Willowdale) began production in 1983, and two refineries (Wagerup and Worsley) started up in 1984. In Brazil, the Aluminio do Maranhao S.A. alumina-aluminum complex started production at Sao Luis, Maranhao, in 1984. The 100,000 mt/yr smelter is planned to be expanded to 245,000 mt/yr by late 1986.

In Venezuela, the Interalumina refinery at Puerto Ordaz started operation in 1983, using bauxite supplied from Brazil, Sierra Leone, and Suriname. The refinery is scheduled to begin receiving bauxite from the Los Pijiguaos bauxite mine by the end of 1986.

Mining operations in both the Dominican Republic and Haiti were shut down in 1982, and the Lydford Mine in Jamaica shut down in 1984. In Guyana, the Linden refinery was shut down in 1982 for maintenance work and modifications and has yet to reopen.

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

Although many technological improvements have been made in producing aluminum from bauxite, the basic processes are unchanged since the metal first became available in commercial quantities almost 100 yr ago. These processes consist mainly of surface mining of bauxite, followed by hydrometallurgical processing to produce alumina (99.5 pct Al_2O_3), and then reducing the alumina to aluminum metal by fused-salt electrolysis in a molten bath of fluoride salts (*10*, p. 1). Generally, for each 4.5 mt of bauxite fed into the alumina refiner, 2 mt of alumina is extracted, from which about 1 mt of aluminum is produced.

Bauxite ore is mined mostly by open pit methods. This entails stripping the overburden, which can subsequently be used to restore the land surface after mining, followed by actual mining of the ore. In Arkansas, where as much as 13 m of overburden must be removed for every meter of bauxite ore exposed, draglines, scrapers, shovels, and trucks are used in the stripping operations. In Jamaica, because the bauxite deposits lie very close to the surface, the overburden of vegetation and topsoil is easily removed. The ore is mined with shovels, draglines, and scrapers; blasting is usually not necessary. The ore is then transported by truck, rail, or aerial tram to the alumina refinery or port. Deposits in Australia and Guinea also lie very close to the surface, with very little stripping of overburden required; however, the bauxite in some of these deposits requires blasting to loosen the ore. Underground mining, by room-and-pillar methods, accounts for most of the bauxite production from deposits in France and other European countries.

In their crude state, bauxites may contain from 10 to 25 pct free moisture. If the bauxite is to be transported over a long distance to the alumina refinery, most of this moisture is usually removed by drying at the mill, usually located at the mine site or at the port. Apart from crushing, drying is often the only processing applied to a metallurgical-grade bauxite before it reaches the alumina plant. Ores of different composition are often blended before milling to ensure as uniform a feed as possible and thus enhance alumina recovery from the refinery.

PROCESSING

Bayer Process for Alumina Production From Bauxite

The Bayer process is the only commercial-scale method of converting metallurgical-grade bauxite to alumina. In the classic Bayer process, aluminum and other elements in the bauxite are dissolved at elevated temperatures and pressures in a strong alkali solution, generally NaOH, to form sodium aluminate ($NaAlO_2$). After separation from the "red mud," the sodium aluminate solution is cooled and seeded to precipitate aluminum trihydrate. The trihydrate is filtered, washed, and calcined to the anhydrous crystalline form, alumina.

Depending on the mineral content of the ore, variations occur in the digestion temperature, pressure, and caustic concentration. In addition, higher silica ores (greater than 8 pct SiO_2) require additional steps, known as the lime-soda-sinter or combination process, to recover alumina and soda lost by combination with silica. Processing trihydrate ore is known as the American Bayer process, and processing monohydrate ore is known as the European Bayer process. Trihydrate ores containing up to 25 pct monohydrate recently have been processed by a method known as the modified Bayer process.

Typical recoveries of aluminum in producing alumina from bauxite ores are 90 to 96 pct for the American Bayer process and 80 to 85 pct for the modified Bayer process or the combination process. Loss of aluminum is due principally to washing steps and silica-alumina reactions. The percent of alumina lost to silica is roughly equal to the weight-percent of reactive silica present in the ore. Miscellaneous losses account for 3 to 5 pct.

For high-silica ores (above 8 pct silica content), enough alumina is lost to justify the addition of the lime-soda-sinter process. In this process, the red mud tails from the Bayer process are combined with soda ash and limestone and sintered at 1,150° to 1,260° C. This sinter is then ground in the presence of water, which leaches the alumina from the sinter. The leach solution is separated from the remaining sinter (brown mud) and returned to the digestor of the Bayer process. The brown mud is discarded.

Hall-Heroult Process (11)

Primary aluminum is produced by the electrolysis of alumina in a molten bath of natural or synthetic cryolite (Na_3AlF_6), which serves as an electrolyte and a solvent for the alumina. The reduction cells or pots containing the bath are typically 3 to 4.6 m wide, 6 to 12 m long, and about 1 to 1.2 m deep, lined with carbon, and connected in electrical series of 100 to 240 cells to form a potline. From 800 to 3,000 lb of aluminum is produced per day in each pot. The carbon lining that serves as the cathode usually must be replaced every 3 or 4 yr. Compounds of aluminum, fluorine, and sodium absorbed in the used pot linings are recovered and recycled by some plants, as is the carbon.

Cryolite and aluminum fluoride are added to the electrolyte to maintain the desirable ratio of sodium and aluminum fluoride and to replace fluorine lost from the cell in pot linings or through volatilization. The melting point of the bath is lowered by the addition of small quantities of fluorspar or, in some instances, lithium compounds. The carbon anode, which is consumed during the operation, is replenished by the Soderberg continuous method or replaced by the prebaked method.

The molten bath or electrolyte may be as deep as 36 cm, but the anode is usually only 5 cm from the pad of molten aluminum. The resistance of the bath is sufficient to maintain an optimum operating temperature of 950° to 980° C. At this temperature, the maximum alumina content of the bath ranges from 3 to 10 pct.

Every 1 or 2 days the molten aluminum is removed from the bottom of the cell by a vacuum-siphon technique. Thermally insulated cast iron pots with airtight lids and downward-sloping spouts are used to withdraw the molten metal. The pots are evacuated, and the molten metal is sucked into the cast iron pot. The molten metal is blended in a holding furnace with other batches, alloyed and cast into various solid forms, or transported molten to fabricating plants as far as 300 miles away.

PRODUCTION COSTS

Estimated average bauxite mining and milling (drying) costs are presented, by country, in table 3. Weighted-average mining and milling costs range from $4.58/mt of bauxite in Jamaica to $30.89/mt of bauxite in Guyana. The estimated weighted-average mining and milling cost for all of the bauxite properties evaluated for this study is $8.54/mt of bauxite. With the exception of the underground bauxite mines in Europe, differences in bauxite mining costs are primarily the result of different stripping ratios. Given the variances in mine and mill operating costs around the world, these differences have little effect on variations of the total cost of aluminum ingot. In some cases, such as Jamaica, taxes and transportation are much more important cost components of bauxite production than mining and drying.

Estimated total aluminum production costs for producing operations and for nonproducers, on a cents per pound aluminum basis, are presented in table 4, and production costs from selected countries are illustrated in figure 5. Weighted-average mining and drying costs in MEC's are only $0.20/lb of aluminum. The balance of the production cost is determined by transportation, refining, smelting, taxes, and recovery of capital. The salient statistic from table 4 is the percentage of total operating costs accounted

Table 3.—Bauxite production costs[1] for producing mines in selected MEC's

(All costs are in January 1985 U.S. dollars per metric ton of bauxite on a weighted-average basis)

Country	Mine and mill operating cost	Transportation to port or refinery	Levy or severance tax	Total cost at port or local refinery
Australia . . .	4.82	5.41	0.89	11.11
Brazil	8.62	9.17	1.00	18.79
France	8.15	5.49	.00	13.64
Greece	7.65	1.27	.00	8.92
Guinea	10.09	14.82	9.13	34.05
Guyana . . .	30.89	7.03	.07	37.98
India	7.51	9.23	.42	17.15
Jamaica . . .	4.58	1.59	14.50	20.68

[1]Weighted average of estimated operating costs for all the producing mines in selected countries.

Table 4.—Aluminum production costs for producing and nonproducing mines in selected MEC's

(All costs are in January 1985 U.S. cents per pound of aluminum on a weighted-average basis)

Bauxite origin	Operating cost			Transportation to refinery or smelter	Byproduct credit[1]	Net operating cost[2]	Recovery of capital[3]	0-pct DCFROR		15-pct DCFROR		
	Mine and mill	Refinery	Smelter					Taxes and royalties[4]	Total cost[5]	Taxes and royalties[6]	Return on investment[7]	Total cost[8]
PRODUCING MINES												
Caribbean: Jamaica	1.1	14.8	44.5	1.0	0.0	61.4	0.5	3.4	65.3	3.7	0.5	66.1
Latin America:												
Brazil	2.4	14.5	32.9	2.9	.0	52.7	.6	2.3	55.6	2.9	1.0	57.2
Guyana	6.8	14.8	46.1	1.8	5.2	64.2	.5	.1	64.7	.7	.7	66.0
Suriname	3.2	14.4	44.1	2.0	.0	67.0	.3	3.6	65.3	3.7	1.0	66.4
Europe:												
France	1.6	19.9	50.1	1.4	.0	73.0	.1	.2	73.2	.1	.2	73.5
Greece	1.4	13.1	25.2	.5	.0	40.2	.5	.5	41.3	2.3	2.9	46.0
Asia: India	1.6	17.9	30.0	2.6	.0	52.1	2.1	1.0	55.2	4.0	2.1	60.3
Oceania: Australia	1.3	13.2	42.2	2.2	.1	58.9	.7	.5	60.0	1.4	1.5	62.4
Africa:												
Guinea	2.3	15.4	46.5	5.7	2.9	67.0	2.1	2.7	71.7	4.4	4.4	77.8
Others[9]	1.4	17.4	36.2	2.6	.0	57.6	1.0	.5	59.1	1.5	1.2	61.3
Other countries[10]	2.4	22.3	52.0	.6	.0	77.3	4.1	1.9	83.3	4.5	4.7	90.6
Average	2.0	14.3	41.0	2.5	.9	59.1	.9	1.6	61.5	2.5	1.6	64.0
NONPRODUCING MINES												
Caribbean: Jamaica	1.1	14.0	43.7	1.4	0.0	60.2	1.0	3.6	64.7	4.4	1.3	66.9
Latin America:												
Brazil	2.0	12.8	40.1	2.7	.1	57.5	.7	2.7	60.8	3.7	2.7	64.5
Others[11]	1.3	18.0	39.8	1.1	.0	60.3	2.7	2.1	65.1	6.9	5.4	75.3
Asia: India	1.1	18.0	31.1	3.2	.0	53.5	3.4	3.1	59.9	11.1	6.3	74.2
Oceania: Australia	1.2	8.7	48.9	1.3	.0	60.1	3.1	1.5	64.8	9.5	10.5	83.3
Africa:												
Guinea	3.0	15.3	45.3	11.0	.0	74.6	5.6	3.1	83.5	6.6	8.1	95.1
Others[12]	.7	12.1	39.4	3.0	.0	55.1	2.6	3.6	61.3	7.8	6.1	71.6
Other countries[13]	2.6	14.7	58.3	2.4	.0	78.0	7.4	3.4	88.9	15.1	17.4	118.0
Average	1.5	13.5	42.9	3.4	.0	61.3	2.8	2.9	67.0	6.9	5.9	76.9

[1]Calculated by multiplying the recoverable quantity of calcined bauxite by its market price.
[2]Includes all operating costs plus transportation costs minus the credit for byproduct revenues.
[3]Sum of remaining undepreciated investments and reinvestments required over the life of the operation.
[4]Includes all property, local, national, and severance taxes plus any royalty or levy, at a 0-pct DCFROR.
[5]Recovery of all costs of production including recovery of capital without discounting.
[6]Includes all property, local, national, and severance taxes plus any royalty or levy, at a 15-pct DCFROR.
[7]Additional revenue per metric ton required to obtain a 15-pct DCFROR.
[8]Includes a 15-pct DCFROR on all investments over the life of the operation.
[9]Includes Ghana and Sierra Leone.
[10]Includes Indonesia, Turkey, and the United States.
[11]Includes Costa Rica, French Guiana, Suriname, and Venezuela.
[12]Includes Cameroon, Ghana, Malawi, and Sierra Leone.
[13]Includes Indonesia and the Solomon Islands.

NOTE—Assumed locations of refineries and smelters for this study are shown in appendix A of reference 2.

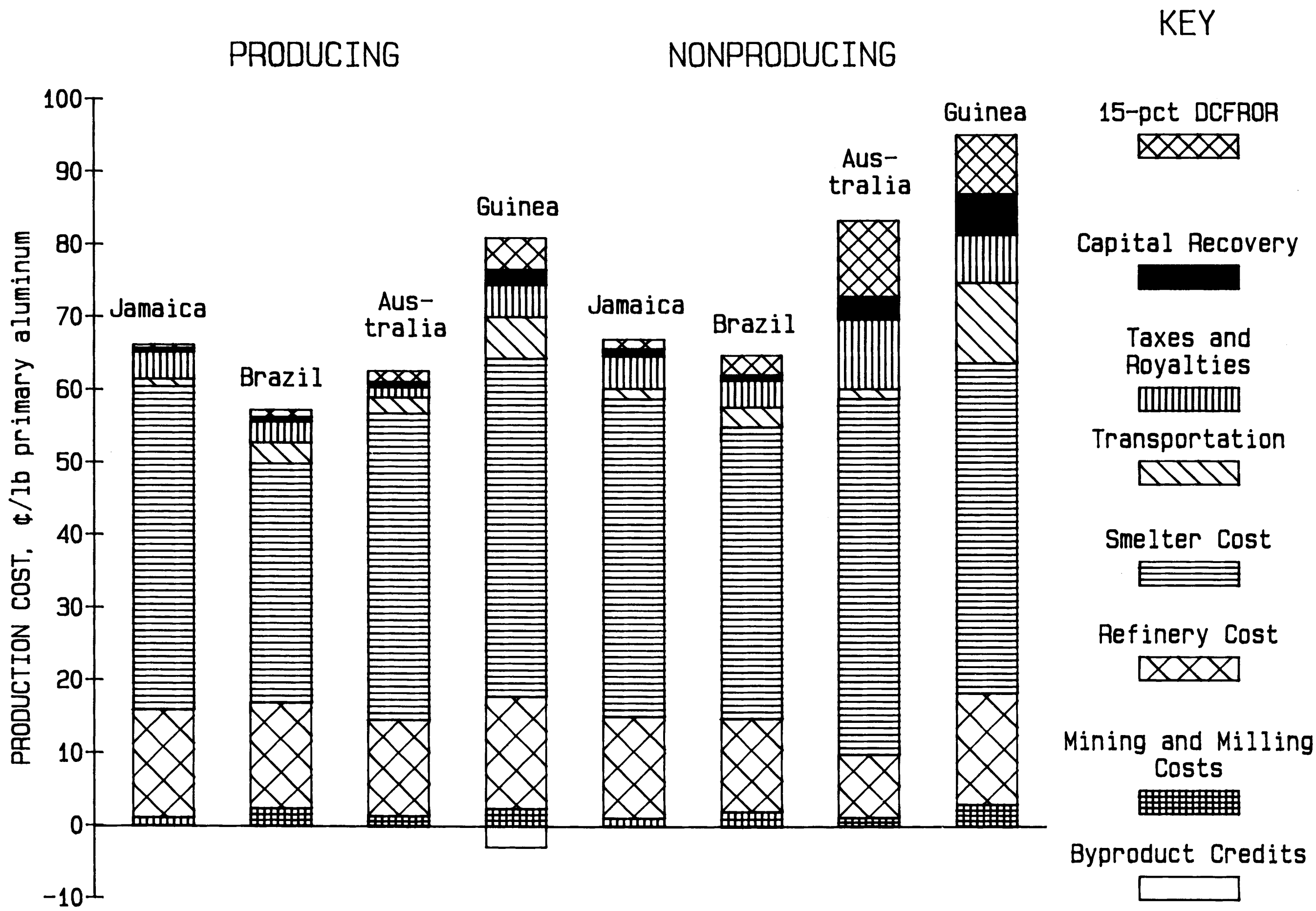

Figure 5.—Aluminum production costs for producing and nonproducing operations in selected MEC's (January 1985 U.S. dollars).

for by refining and smelting. For both producers and potential producers, refining and smelting account for over 90 pct of total operating costs. Costs for aluminum smelting alone account for almost 70 pct of the total operating costs of the operations analyzed in this study.

Cost variations in alumina production are largely determined by the type of bauxite (which determines the type of Bayer process employed to produce alumina, as well as soda losses per metric ton of alumina and the cost of removing organics); and the bauxite-alumina ratio, which is a function of the recoverable Al_2O_3 grade of the bauxite.

Often accounting for up to 60 pct of the total operating cost of aluminum ingot production, electricity is the critical cost factor in aluminum smelting. Electric power costs charged to aluminum smelters vary widely throughout the world and are the principal reason for the variation in aluminum ingot production costs between producing nations. Assuming average electricity consumption of 14,200 kW·h/mt of aluminum produced in a modern plant, a price change of 1 mill/kW·h can result in a variation in operating costs of $14.20/mt of aluminum. In 1983, electric power rates paid by aluminum smelters ranged from 3 to 60 mills/kW·h (*12*, p. 31). The weighted-average power rate for primary aluminum producers in MEC's was estimated to be 19 mills/kW·h in 1983, compared to the U.S. average of 23 mills/kW·h. The MEC average excluding the United States, was 17 mills/kW·h, and eliminating high energy cost producers in Asia reduced the average to 12 mills/kW·h. Some smelters located in Canada, Brazil, Venezuela, Norway, Ghana, Australia, and the Gulf States in the Middle East have energy costs at, or below, 10 mills/kW·h (*13*).

Power rates in the United States have risen dramatically since the second energy crisis in 1979. In the Pacific Northwest, the Bonneville Power Administration (BPA) rate to the aluminum smelters in its service area increased from 2.97 mills/kW·h in 1979 to 22.5 mills/kW·h in 1983; a 658-pct increase (*13*). In late 1985, BPA temporarily reduced the rate to 18.8 mills/kW·h with future prices tied to the realized price for ingot. The Tennessee Valley Authority (TVA) rate remains at about 36 mills/kW·h, and rates charged to other smelters generally range between the BPA and TVA rates.

AVAILABILITY

TOTAL RECOVERABLE

This study analyzed potential aluminum production based upon the demonstrated resources of 84 bauxite mines and deposits in 20 MEC's. Of this resource, representing an in situ tonnage of 19.7 billion mt, some 18.3 billion mt is considered to be minable, providing a production potential of 3.5 billion mt of primary aluminum. The resource tonnage includes refractory-grade bauxite contained in 15 of the properties studied, which have a production potential of 597 million mt of calcined refractory bauxite. In the analysis, the revenues generated from the production of calcined refractory bauxite are credited against the cost of producing primary aluminum. As of January 1985, 50 of the mines evaluated were assumed to be in production and 34 nonproducing deposits were evaluated as potential operations. This total includes deposits that are currently developing or have specific development plans or have been explored. Aluminum operations in Yugoslavia and CPEC's were not evaluated owing to a lack of reliable cost and resource information.

The potential production of aluminum based on the average total cost of production for each operation is illustrated in figure 6. The figure includes three curves: the solid lower curve reflects potential availability of aluminum from producing operations at a 0-pct DCFROR, the dashed curve reflects the potential availability of aluminum from producing operations at a 15-pct DCFROR, and the solid upper curve reflects the potential availability of aluminum from potential (nonproducing) operations at a 15-pct DCFROR. A total of 3.54 billion mt of aluminum is potentially recoverable from the demonstrated bauxite resources of the MEC operations evaluated. This total is 231 times 1985 production of 15.3 million mt of primary aluminum. The estimated total costs for producing and potential operations are listed in table 5. Potential aluminum from the 50 bauxite mines evaluated as producing operations amounts to 1.51 billion mt (43 pct of the total), and potential aluminum from the 34 nonproducers amounts to 2.03 billion mt (57 pct).

As illustrated in figure 6, the individual costs for producing aluminum operations range from about \$0.41/lb to \$0.93/lb at a 0-pct DCFROR, and from \$0.42/lb to \$1.04/lb at a 15-pct DCFROR. For nonproducing operations, estimated costs range from \$0.53/lb to \$1.25/lb at a 15-pct DCFROR. The Metals Week U.S. market price for primary aluminum metal was 55.25¢/lb in January 1985. European prices were slightly higher in the European market, ranging from 57.45¢/lb in the United Kingdom to 62.85¢/lb in Italy. At the January 1985 market price, 359.1 million mt of aluminum (24 pct of the total) from producing operations could be produced and return at least a 0-pct DCFROR. A total of 304.3 million mt (20 pct of the total) could be produced and return at least a 15-pct DCFROR at a constant-dollar price of 57.45¢/lb. For nonproducers, 248.2 million mt of aluminum (12.2 pct of the total) could be produced and return at least a 0-pct DCFROR at the January 1985 market price, and 48.0 million mt (2.4 pct of the total) could be produced and return at least a 15-pct DCFROR.

As shown in table 5, the weighted-average total cost of aluminum production for producing operations in MEC's was estimated to be 61.5¢/lb of aluminum in January 1985 dollars. This suggests that if ingot prices remain below 61.5¢/lb in constant January 1985 dollars, that the majority of aluminum producers will lose money on ingot production. This explains the closures of a number of high-cost refineries and smelters in the United States and Japan over the past several years. The cost figures also suggest that profits in the aluminum industry accrue from the manufacture and

Table 5.—Potential aluminum production and average total cost for producing and nonproducing operations

Bauxite origin	Producing operations: Potential production, 10^3 mt	Producing operations: Total cost,[1] ¢/lb, 0-pct DCFROR	Producing operations: Total cost,[1] ¢/lb, 15-pct DCFROR	Nonproducing operations: Potential production, 10^3 mt	Nonproducing operations: Total cost,[1] ¢/lb, 0-pct DCFROR	Nonproducing operations: Total cost,[1] ¢/lb, 15-pct DCFROR
Caribbean: Jamaica	128,777	65.3	66.1	245,522	64.7	66.9
Latin America:						
Brazil	150,152	55.6	57.2	182,624	60.8	64.5
Guyana	98,827	64.7	66.0	NAp	NAp	NAp
Others[2]	101,173	65.3	66.4	97,589	65.1	75.3
Total	350,152	NAp	NAp	280,213	NAp	NAp
Europe:						
France	1,805	73.2	73.5	NAp	NAp	NAp
Greece	130,973	41.3	46.0	NAp	NAp	NAp
Total	132,778	NAp	NAp	NAp	NAp	NAp
Asia: India	16,319	55.2	60.3	181,407	59.9	74.2
Oceania: Australia	406,353	60.0	62.4	284,135	64.8	83.3
Africa:						
Guinea	442,466	71.7	77.8	609,643	83.5	95.1
Others[3]	14,258	59.1	61.3	288,171	61.3	71.6
Total	456,724	NAp	NAp	897,814	NAp	NAp
Other countries:[4]	17,226	83.3	90.6	141,815	88.9	118.0
Grand total	1,508,329	61.5	64.0	2,030,906	67.0	76.9

[1]Weighted average.
[2]Includes Costa Rica, French Guiana, Suriname, and Venezuela.
[3]Includes Cameroon, Ghana, Malawi, and Sierra Leone.
[4]Includes Indonesia, Solomon Islands, Turkey, and the United States.

sale of fabricated products rather than ingot, giving the large integrated firms and nonintegrated fabricators the advantage over nonintegrated producers of ingot.

The data presented in table 5 highlight the importance of Australia, Guinea, and Brazil as the dominant sources of bauxite for current and future aluminum production in MEC's. Based on this analysis, fully two-thirds of the potentially recoverable aluminum from producing operations in MEC's will be derived from bauxite from Australia, Guinea, or Brazil, and 53 pct of the aluminum potentially recoverable from nonproducing deposits is from these three countries.

In terms of production costs, countries such as Australia and Brazil, which are developing integrated industries, appear to have the advantage over the other producers. These countries have cheap sources of electric power, the alumina refineries are located relatively close to the bauxite mines, and the transportation costs to the smelters are relatively low. The Jamaicans, on the other hand, are losing their traditional U.S. market as a number of gulf coast refineries have shut down, and alumina produced in Jamaica is not competitive with Australian alumina in the U.S. market. Virtually all of the alumina imported into smelters in Oregon and Washington now comes from Australia. In U.S. dollar terms, production of aluminum from Guinean bauxite appears high, largely because of an overvalued currency and the fact that much of Guinea's exports go to high-cost refineries in Europe. In reality, the mining and drying costs for Guinea that appear in table 5 would be much lower if Guinea would devalue its currency against the U.S. dollar. Conversely, production costs in Greece are currently quite low in U.S. dollar terms as devaluations of the drachma have exceeded the rate of Greek inflation.

ANNUAL CAPACITY

Potential annual capacity of aluminum from the demonstrated resources of producing bauxite operations in MEC's is shown in figure 7. The curves reflect potential annual aluminum production from producing bauxite mines, including planned expansions when known. The cost ranges shown in figure 7 include a 15-pct DCFROR on the assumption that, over the long run, all costs of production, including a 15-pct DCFROR, must be recovered.

The curves indicate a potential production of 19.7 million mt of aluminum in 1985 from the existing capacity of producing bauxite mines in MEC's. This potential production capacity was 7.4 million mt higher than the actual 1985 production of 12.3 million mt from MEC's. The curves increase slightly during the late 1980's and early 1990's and reach a peak of 21.8 million mt of aluminum in 1994. A real price of over $0.80/lb (in January 1985 dollars) would be required in order for all of this capacity to be available, however. The assumption that capacities remain constant (with a few exceptions) throughout the life of the mine makes output appear more constant through 1995 than may be the case. If needed, additional bauxite resources from undeveloped deposits with an aluminum production potential of 14 million mt/yr could be developed by 1990, although production capacities of existing bauxite mines can generally be expanded at a much lower cost than that for developing new bauxite operations. However, demand for bauxite (and alumina) is derived from that for aluminum. New bauxite capacity, either from expansion or the development of new mines, would correspond with the expansion or development of new refining and smelting capacity in order to process the bauxite into aluminum. As mentioned previously, most of the costs associated with aluminum production are incurred in refining and smelting, not bauxite production.

The figures suggest that little new bauxite capacity will have to be added during the rest of this decade, and as a result, any new mines will likely be low-cost mines that will replace higher cost mines that are currently producing. This scenario would be a continuance of what happened in the late 1970's and early 1980's when new bauxite production in Australia replaced production from Jamaica.

The Bureau has forecast that the probable world average annual growth rate for primary aluminum demand will be about 3.7 pct up to the year 2000 (*14*). This would require world primary aluminum production of 22.6 million mt in 1990 and 30.8 million mt in 2000. Additional capacity, planned or under construction, in Australia, Brazil, Canada, India, and Venezuela should meet increases in demand (and serve to depress prices) through the balance of this decade. However, if prices remain low and the construction of new capacity is uneconomic, the aluminum market could tighten in the 1990's.

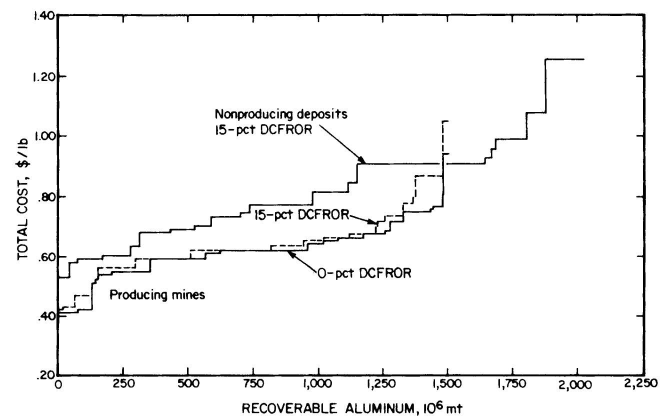

Figure 6.—Potential total aluminum available from MEC's (January 1985 U.S. dollars).

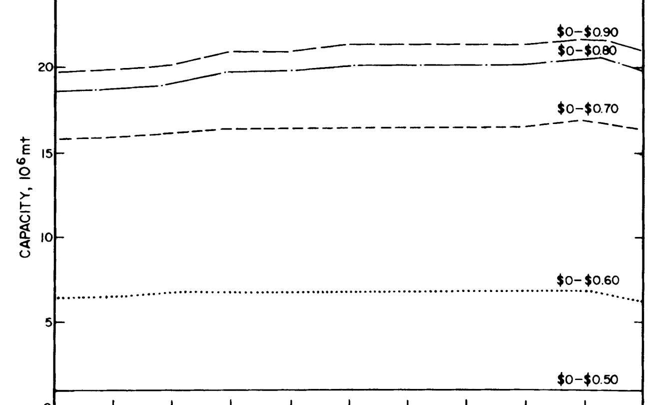

Figure 7.—Potential annual aluminum production from producing mines in MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

CONCLUSIONS

The demonstrated resources of bauxite ore contained in the 84 deposits analyzed in this study amount to approximately 19.7 billion mt, from which an estimated 3.54 billion mt of aluminum is recoverable.

Demonstrated resources of bauxite are more than sufficient to serve as the source of primary aluminum well into the next century. Furthermore, as new bauxite deposits are discovered, and as bauxite deposits with tonnage estimates at the identified resource level are further explored, they will be included as demonstrated resources.

For producing operations, bauxite mining and milling account for 3.4 pct of the total operating cost, transportation accounts for 4.1 pct, alumina refining accounts for 23.3 pct, aluminum smelting accounts for 66.7 pct, and levies and taxes account for 2.5 pct.

A larger percentage of alumina and aluminum will be produced in the major bauxite-producing countries in the future, especially Australia and Brazil. Guinea is a growing supplier of the export bauxite market, whereas Jamaican production is declining as its traditional market in the United States continues to import higher quantities of bauxite and alumina from Australia and Brazil. Japan has ceased to be a major factor in the aluminum industry except as a provider of equity capital and to serve as a major export market for such countries as Australia, Brazil, Indonesia, and Venezuela. The U.S. aluminum producers will continue to face increasing competition from overseas and, as a result, a larger percentage of U.S. aluminum supplies will come from imports and the recovery of metal from scrap.

REFERENCES

1. McCawley, F.X. Aluminum. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 8-9.

2. Peterson, G.R., and S.J. Arbelbide. Aluminum Availability—Market Economy Countries. A Minerals Availability Program Appraisal. BuMines IC 8917, 1983, 31 pp.

3. Peterson, G.R., R.L. Davidoff, D.I. Bleiwas, and R.J. Fantel. Alumina Availability—Domestic. A Minerals Availability System Appraisal. BuMines IC 8861, 1981, 23 pp.

4. U.S. Bureau of Mines. A Dictionary oı Mining, Mineral, and Related Terms. 1968, p. 89.

5. U.S. Bureau of Mines and U.S. Geological Survey. Principles of a Resource/Reserve Classification for Minerals. U.S. Geol. Surv. Circ. 831, 1980, 5 pp.

6. Baumgardner, L.H. Bauxite. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 18-19.

7. International Bauxite Association. Quarterly Review. V. 7, No. 2, Oct.-Dec. 1981, 46 pp.

8. Patterson, S.H., H.F. Kurtz, J.C. Olson, and C.L. Neely. World Bauxite Resources. U.S. Geol. Surv. Prof. Paper 1076-B, 1986, 151 pp.

9. Australian Mineral Economics Pty. Ltd. (Sydney, Australia). The World Aluminum Industry Statistical Update 1985. 1986, 139 pp.

10. Stamper, J.W., and H.F. Kurtz. Aluminum. BuMines Mineral Commodity Profile, 1978, 29 pp.

11. Kurtz, H.F., and L.H. Baumgardner. Aluminum. Ch. in Mineral Facts and Problems, 1980 Edition. BuMines B 671, 1981, pp. 9-34.

12. International Bauxite Association. Quarterly Review. V. 10, No. 4, Apr.-June 1985, 35 pp.

13. Kennedy, J.S. Energy and the Primary Aluminum Industry. U.S. Dep. Commerce, International Trade Administration, 1985, 54 pp.

14. McCawley, F.X., and L.H. Baumgardner. Aluminum. Ch. in Mineral Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 9-31.

ANTIMONY

INTRODUCTION

In industry, antimony is used in either metallic or nonmetallic forms. As a metal, antimony is used as an alloying element to increase strength and inhibit chemical corrosion, such as in antimonial lead used in lead-acid storage batteries. The consumption of nonmetallic antimony, the larger portion of the total primary consumption, is mainly in flame retardants. Antimony is also used as an opacifier for enamel and pigment. In most cases, other elements can be substituted for antimony uses.

The United States, with no significant antimony resource, is expected to remain dependent on imported ores, oxides, and metal.

Antimony is recovered as a primary product, or as a byproduct of other metal, mostly at the smelter. Secondary antimony metal is recovered from recycling of scrap metal (mostly battery scrap) where antimony is an alloy. However, in this study, only those mines and deposits from which antimony is or can be produced as primary antimony have been analyzed. For this study, the final product is refined antimony metal.

This chapter analyzes the availability of antimony metal. Most of the information presented here is an updated summary of Bureau of Mines Information Circular 9098 "Antimony Availability—Market Economy Countries. A Minerals Availability Appraisal" (*1*).[1] Additional information on the world antimony industry is available from other Bureau of Mines publications (*2-4*).

GEOLOGY AND RESOURCES

GEOLOGY

Antimony distribution ranges from 0.2 to 0.5 ppm in the continental crust. In igneous rocks, the concentration ranges from 0.1 to 1.0 ppm, with higher concentration noted in basaltic than in granitic rocks (*5*). Because of its strong affinity for sulfur and the metallic elements such as lead, copper, and silver, antimony is rarely found as native metal.

Normally associated with igneous activity, antimony deposits are genetically related to such intrusives as granites, diorites, and monzonites. Antimony ores are commonly found in quartzose veins, pegmatites, and as replacements in limestone. Typical antimony deposits are small, irregular, and discontinous bodies with grade sharply decreasing at depth.

Antimony occurrences are classified into simple and complex deposits (*5*). Simple deposits consist principally of stibnite (Sb_2S_3), or on rare occasions, native antimony in a siliceous gangue with some sulfide metals. In some cases, the antimony is associated with small amounts of gold. The simple deposits generally occur at shallow depth, suggesting the minerals are formed at low temperature of formation. The Wadley deposit in Mexico and the Turhal-Tokat deposit in Turkey are believed to be of this type. Complex deposits consist of Sb_2S_3 in association with scheelite, or antimony sulfosalts. The complex deposits also may contain varying amounts of copper, lead, gold, and silver as well as the common sulfide of these metals. Ore of complex deposits generally is mined primarily for lead, gold, silver, zinc, or tungsten. Most of the antimony produced in the United States is from complex deposits.

RESOURCES

Primary Antimony

The in situ demonstrated resources under study accounted for a total of 460,000 mt of contained antimony (table 1). An additional 974,000 mt antimony is available from inferred resources of the deposits studied, although these resources were not included in the economic analysis. Based on the demonstrated resources, Africa and Latin America account for about 63 pct of the total contained antimony. On the inferred level combined resources from Latin America and other countries (Italy and Thailand) account for 80 pct of the total contained antimony. The mines in Italy and Thailand, although containing smaller in situ inferred resource tonnages as compared to Latin America and Africa, account for about 49 pct of the total contained antimony because they are high-grade deposits.

Estimates of total world antimony resources are conflicting and vary from one source to another. These sources are, however, in consensus that more than half of the world antimony resources are located in China. Figure 1 indicates estimated world antimony resources. Antimony resources located in the Soviet Union, China, and other countries within the centrally planned economy countries (CPEC's) were not analyzed in this study. Production cost estimates could not be supported because of the difficulty in collecting quantitative resource information.

[1]Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

Table 2 shows the antimony properties selected for evaluation.

Byproduct Antimony

In additon to the primary antimony resource, antimony is also recovered as a byproduct from the smelting and refining of lead and silver ores. The recovery of byproduct antimony in most cases is incidental to the recovery of the primary metals. A major domestic source of byproduct antimony is the Sunshine Mine in Idaho, which recovers antimony because it is deleterious to silver refining. Antimony is also contained in lead ores. However, most often it is not paid for by the smelters because of its negative effect in processing the primary lead. As a result, the antimony resource base from this source is hard to define.

Secondary Antimony

Secondary (scrap) antimony provides a large portion of the total antimony supply in most industrialized countries. In general, the recovery of secondary antimony is incidental to the recovery of the principal metals, mainly lead. The

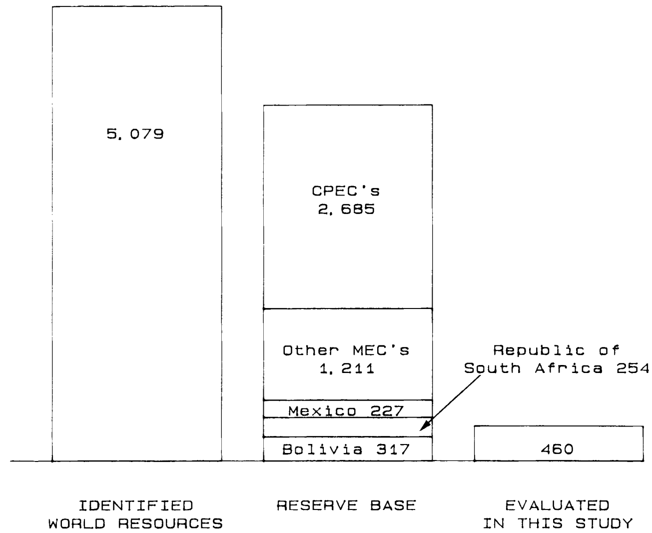

Figure 1.—Estimates of world antimony resources (thousand metric tons of contained antimony).

Table 1.—Summary of MEC demonstrated antimony resources evaluated for this study, as of January 1985, thousand metric tons

Location	In situ resources		Contained antimony		Recoverable antimony metal[1]
	Demonstrated	Inferred	Demonstrated	Inferred	
Africa	7,366	4,467	212	129	142
Asia, Australia	1,777	1,598	72	58	50
Latin America	1,404	6,662	80	304	61
North America	2,373	193	50	7	31
Others[2]	779	2,815	46	476	0
Total	13,699	15,735	460	974	284

[1]Estimated recovery from demonstrated resources.
[2]Includes mines in Italy and Thailand that do not produce refined antimony as an end product and are not included in the availability analysis.

Table 2.—MEC antimony properties included in this study

Country and property name	Ownership	Current status[1]	Mining method	Annual ore capacity, 10^3 mt
Australia:				
Hillgrove	New England Antimony Mines NL	P	U	21.0
Wild Cattle Creed	Antimony Australia N.L.	N	U	70.0
Bolivia:				
Candelaria	Empresa Minera S. Juan Ltd.	P	U	60.0
Caracota	EMUSA	P	U	33.7
Chilcobija	Empresa Minral Unificada SA (EMUSA)	P	U	95.0
Churquini	Churquini Enterprises, Inc.	P	U	18.0
Espiritu Santo	EMUSA	T	U	48.0
la Salvadora	Churquini Enterprises, Inc.	T	U	12.0
Rosa de Oro	Empresa Minera Bernal llnos	P	U	15.0
Canada: Lake George	Consolidated Durham Mines and Resources Ltd.	T	U	100.0
Mexico: Wadley	Cia Minera y Refinadora Mexicana SA	P	U	31.2
Morocco:				
Timerhdoudine	Societe des Travaux et de Recherches Miniere	P	U	40.0
Tourtit	..do	T	U	60.0
South Africa, Republic of:				
Consolidated Murchison Ltd.	Consolidated Murchison Ltd.	P	U	500.0
Turkey: Turhal-Tokat	Ozdemir Antimony Mine Ltd.	P	U	35.1
United States: Yellow Pine	Ranchers Exploration and Development Co.	T	S	[3]350.4

[1]N, nonproducing; P, producing; T, temporarily shut down.
[2]S, surface; U, underground. For deposits not producing, mining method is proposed.
[3]Proposed mining capacity.

largest single source of secondary antimony is the antimonial lead battery scraps. Other sources are bearing metal, babbitt metal, and lead dross. Depending upon the degree of purity of the scrap metal, the plants either resmelt the scrap or produce a specification material through the addition of lead, tin, or antimony (*6*). Antimony derived from this source is normally consumed in secondary alloy production.

U.S. AND WORLD HISTORICAL PRODUCTION

World antimony production in 1985 (not including the United States) was estimated at 53,000 mt with the market economy countries (MEC's) accounting for a total of 27,000 mt or 51 pct of the total production. U.S. production was withheld to avoid disclosing company proprietary data. World historical production trends, in 5-yr intervals, are shown in figure 2. The curves illustrate how production from Africa and Latin America have dominated antimony markets for many years.

Because of its high degree of industrial activity, the United States, with no significant antimony production capacity, will continue to rely on imports. Currently, the United States imports mostly from Bolivia, Mexico, Republic of South Africa, and China. Normally the United States imports and consumes 30 to 50 pct of the MEC total annual mine production.

Secondary antimony has always been an important part of the U.S. antimony supply. Historically, domestic production from secondary antimony contributed between 30 and 60 pct of the total annual antimony supply. (*6*).

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

Antimony deposits are small, irregular, and discontinous ore bodies that often present problems in ore extraction, hence efficient exploitation by large-scale mining operations is limited. In some cases, such as in Thailand, mining of antimony ore is accomplished by unsystematic hand operations. Where the deposit or vein structure warrants the use of large-scale mining operations, mining methods such as open pit, cut and fill, sublevel stoping, shrinkage, and open stoping are used. Table 2 shows the antimony properties selected for evaluation.

PROCESSING

Beneficiation

Run-of-mine ores, except direct shipping grades (5 to 25 pct Sb), are generaly upgraded to marketable products through crushing, grinding, and classifying processes. Beneficiation of antimony sulfide ores is normally accomplished by gravity and flotation processes. Antimony from simple ores, those that contain only stibnite and siliceous materials, are easily concentrated to about 65 pct Sb with the use of flotation reagents such as copper sulfate or lead nitrate for an activator and xanthate for a collector (*6*). Beneficiation of complex antimony ores generally employs a flotation process along with gravity concentration to recover the byproducts.

Antimony oxides have not been successfully floated. In Mexico, oxide ores are normally upgraded by either hand sorting or by hand jigging.

Smelting

Extraction of antimony from ore or concentrate is accomplished by either pyrometallurgical or electrometallurgical methods. Because of the ease of volatilizing oxide ores as well as reducing both sulfide and oxide ores to metal, pyrometallurgical extraction methods have been used more often than electrometallurgical. Antimony metal and antimony trioxide can be produced in the same pyrometallurgical plant by merely changing the quantities and types of fluxing agents. Where oxide ores are volatilized, a special type of roaster is adopted in the process. The electrometallurgical technique is not discussed in this report because it is not used in general commercial application.

Pyrometallurgical smelting of antimony ores and concentrates to recover antimony dominates the industry, owing to the relative ease of the extraction technique. The important characteristics of antimony that favor smelting include the low melting and boiling points of the metal, high

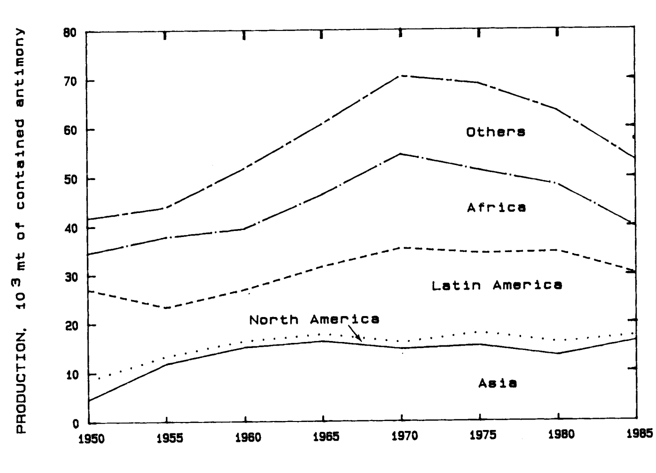

Figure 2.—World antimony production, 1950-85.

vapor pressure, thermal disassociation, oxidation-reduction reactions, rate of chemical reactions, and the equilibrium established during the process (7).

Antimony is smelted in blast furnaces in two distinct temperature reaction zones. At the initial stage of smelting, stibnite melts and trickles down the charges. As the molten sulfide reaches the temperature zone of 1,332° F, a portion of the molten sulfide is vaporized and carried upward by the blast. Upon reaching a temperature of 2,012° F, thermal dissociation to elemental antimony occurs. Simultaneously, oxidation of the vapors to SO_2 and Sb_2O_3 takes place. Metallic antimony not vaporized or oxidized is collected periodically in the forehearth. Vaporized antimony passes through the furnace into the flue where it is cooled. Upon completion of cooling, the condensate is collected in the baghouse. The collection of a large volume of fumes and dusts is necessary to realize high recovery. The entire process recovers about 85 pct of the antimony content as Sb_2O_3 and 10 pct as metallic antimony.

Several variations are available in the antimony smelting process. One of them is the precipitation technique. The process takes advantage of the greater affinity of sulfur for iron than for antimony. The smelting is accomplished by mixing fine iron scrap in the furnace charge. The iron reduces the stibnite to metallic antimony with an iron sulfide matte.

Products from smelters are impure antimony oxides and metal. To attain a commercial grade, the intermediate products are refined to desired products.

Refining

Refining is accomplished either by a pyrometallurgy process or by an electrorefining process. Basically, pyrometallurgy could refine the impure metal or the antimony oxide to the desired final product, i.e., in time of high metal demand, antimony oxide is readily converted to pure metal and vice versa. In electrorefining, the process starts with impure metal and produces pure metal as the final product.

Pyrometallurgical refining operations in most cases are carried on in a small reverberatory furnace. Refining begins with the charging and melting of the impure antimony metal. Upon melting of the metal, a mixture of soda and coke dust is added to produce thick slag. In about 3 h, the slag is skimmed off. The impurities of iron and sulfur are then removed by adding chemical reagents such as oxysulfide of antimony and potash. The final slag, referred to as star slag, which is principally made up of antimony glass, contains 20 to 60 pct Sb. After about 15 min, the antimony metal, called regulus, is ladled into molds where a starlike pattern forms on the metal surface. The starred regulus usually contains over 99.6 pct Sb (7).

Electrolytic refining starts by casting the impure metal into anodes. Once the anodes are submerged into the aqueous bath, the antimony is dissolved into the solution. By means of electrolysis, the antimony is redeposited on the cathode. The impurities, generally composed of sulfur, iron, copper, arsenic, gold, and silver, are collected in the cell slimes. When present in appreciable quantities, arsenic and copper codeposit with antimony in the cathode. Copper is normally removed from the electrolyte by cementation on powdered metallic antimony, but arsenic tends to concentrate in the electrolyte. Arsenic is partially removed from the electrolyte by distillation. As a result, antimony cathodes always contain a small amount of arsenic. Complete arsenic removal is possible by resmelting the cathode with an oxidizing slag composed of caustic, sodium nitrate, and soda ash, which removes arsenic as sodium arsenate. The starred regulus metal produced normally contains over 99.9 pct Sb (7).

PRODUCTION COSTS

Table 3 and figure 3 illustrate the production costs for selected world antimony mines and deposits. The mine production costs from producing mines vary from an average of \$0.19/lb to \$0.57/lb Sb. The difference in cost is mainly because of the mining method used in the operations. The mines included in the "other" category use expensive underground timbered open stoping or cut-and-fill methods, whereas Latin American mines utilize open cut, open stope, and cut and fill. In addition, labor costs in Latin America are much lower compared with labor costs in Africa and Australia.

In milling operations, "other" category mines also show the highest operating costs. The high production cost is mainly because of the added cost incurred in the recovery of gold as a byproduct. Latin America shows the lowest milling cost because of the low labor cost in these countries. In addition, one mine produces a direct shipping ore, thus incurring no milling cost.

Smelting and refining costs are also expensive in "other" category countries because of the high labor cost and mineral complexity of the ore. Latin America shows the least expensive operation, because most of the ores are high grade and there is availability of cheap labor.

Transportation cost reflects the total cost of transporting the mill concentrate to the smelter. These costs may include one or a combination of trucking, railroad transportation, ocean freight, insurance, and handling costs. As shown in table 3, operations in the "other" category reflect the highest transportation cost. This is because this category mainly represents mines in Africa and a large portion of the concentrates from these mines are shipped to European smelters. Hence, concentrates shipped from African mines incur the cost of land and ocean transportation including additional handling and insurance costs. Latin America shows the least expensive transportation costs, because large amounts of the concentrates are smelted within relatively short trucking and/or railroad transportation distances.

When byproduct credits are removed from the total production cost, mines of "other" category countries show the most expensive operation. The high cost is mostly influenced by expensive mining operation.

Most of the mines in Latin America have been in operation for quite a long time; therefore, most capital investments have been fully depreciated. As shown in the table, Latin America has the lowest capital recovery cost.

Of the nonproducers, the North American mines reflect the most expensive total operating costs, primarily because of high labor costs and low ore grades. On the other hand, Latin American nonproducers have the least expensive costs, primarily because of the availability of cheap labor, short transportation distances, and high ore grades.

Table 3.—Antimony production costs for producing and nonproducing operations in selected MEC's

(All costs are in January 1985 U.S. dollars per pound antimony metal on a weighted-average basis)

	Operating Cost			Transportation cost	Byproduct credit[1]	Net operating cost	Recovery of capital[2]	0-pct DCFROR		15-pct DCFROR		
	Mine	Mill	Smelter-refinery					Taxes and royalties[3]	Total cost[4]	Taxes and royalties[5]	Return on investment	Total cost[7]
Producing:												
Latin America	0.19	0.08	0.05	0.07	0.00	0.39	0.05	0.04	0.48	0.08	0.07	0.59
Other[8]	.57	.19	.19	.14	.18	.91	.07	.01	.99	.07	.08	1.13
Nonproducing[9]	.33	.27	.53	.08	.18	1.03	.38	.03	1.44	.23	.24	1.88

[1]Includes all byproduct revenue credits for the operation.
[2]Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure, and reinvestments required over the life of the operation.
[3]Includes property, State, Federal, and severance taxes, and royalties, where applicable, calculated at a 0-pct DCFROR.
[4]Equal to the sum of net operating costs, taxation, and capital recovery determined at a 0-pct DCFROR.
[5]Includes property, State, Federal, and severance taxes, and royalties, where applicable, calculated at a 15-pct DCFROR.
[6]Revenue increase per metric ton necessary to obtain a 15-pct DCFROR.
[7]Equal to the sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery *plus* the per metric ton increase necessary to provide a 15-pct DCFROR after taxation.
[8]Includes 1 property each in Turkey, Republic of South Africa, Australia, and Morocco.
[9]Includes 1 property in Australia, 2 in Bolivia, and 1 each in Canada, Morocco, and the United States.

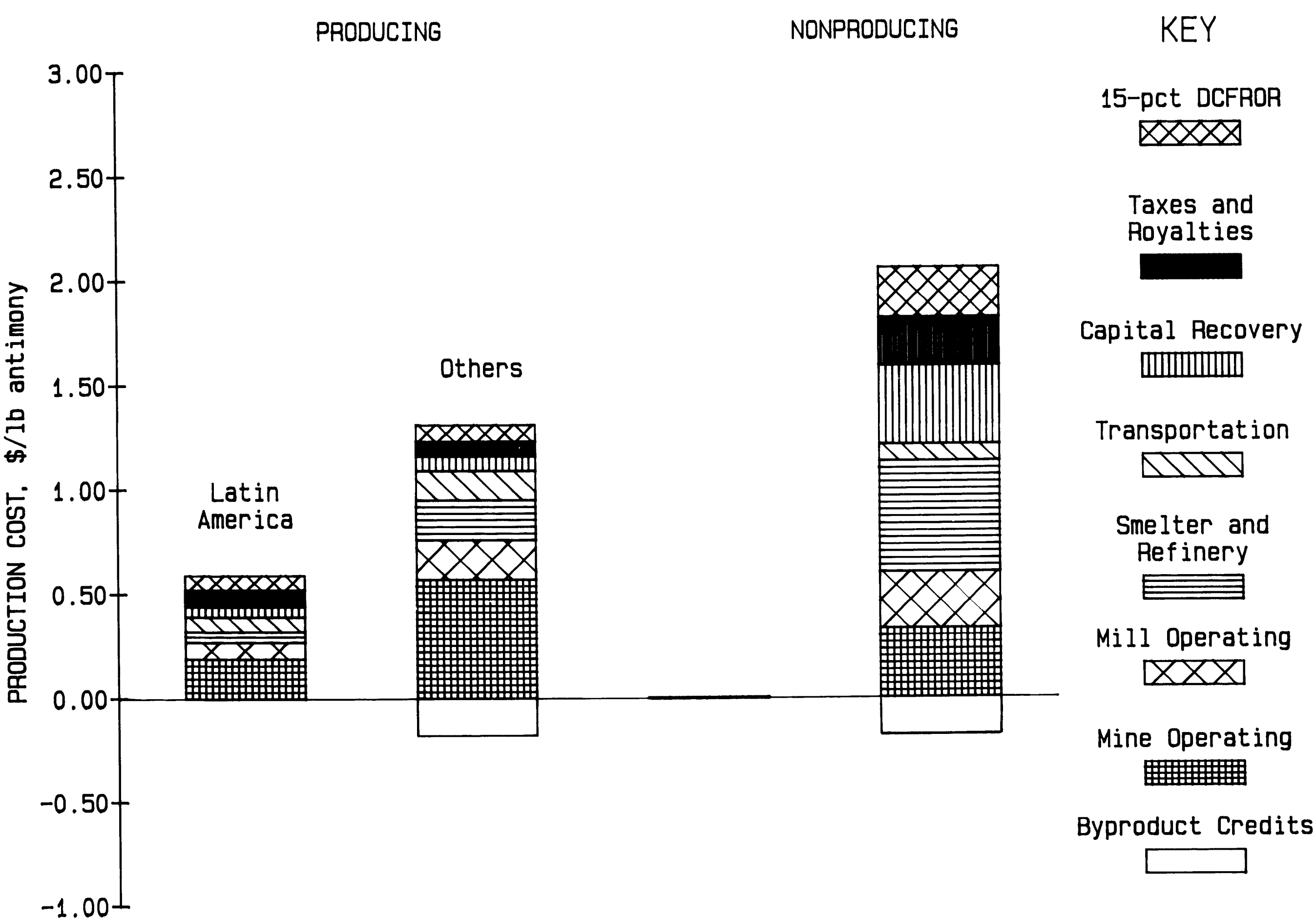

Figure 3.—Antimony production costs for selected MEC's (January 1985 U.S. dollars).

AVAILABILITY

TOTAL RECOVERABLE

At the demonstrated resource level, potential total recoverable antimony from MEC's was estimated at 284,000 mt Sb from producing and nonproducing properties. Figure 4 illustrates only 273,900 mt since approximately 10,000 mt is at costs greater than \$2/mt (off of the graph). As of January 1985, 10 properties were evaluated as producing mines and 6 as nonproducing mines. The nonproducing properties include those mines that are temporarily closed because of weak market conditions. Production costs for properties in CPEC's were not estimated because of the difficulty of gathering information from these countries.

Of the total 235,000 mt Sb that would be available at a production cost, including a 15-pct discounted-cash-flow rate of return (DCFROR), of less than the January 1985 price for antimony metal (*4*) of approximately \$1.30/lb,[2] 80 pct is from producing mines.

ANNUAL CAPACITY

Potential 1985 production (at full capacity) from producing mines is shown in figure 5. As shown, the curves reflect 1985 potential production of producing mines at different production cost levels. The upper curve represents production cost at 0-pct DCFROR and the lower curve represents the operating cost. All 21,400 mt of capacity could be produced at less than the January 1985 market price of antimony of approximately \$1.30/lb. This compares to 1985 MEC production of approximately 28,000 mt (*4*).

[2]New York dealer price for 99.5 to 99.6 pct metal, c.i.f. U.S. ports.

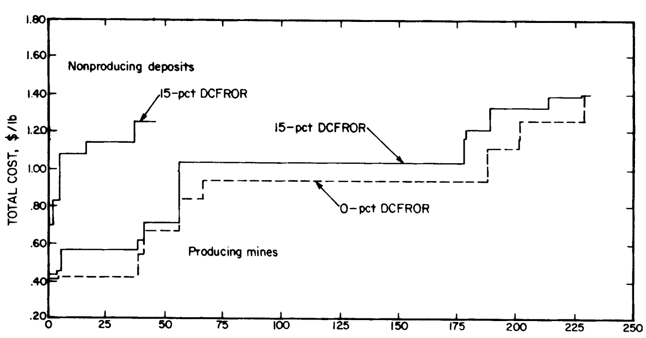

Figure 4.—Potential total antimony available from MEC's (January 1985 U.S. dollars).

Potential annual production from 1985 to 1995 for the 10 producing properties analyzed is shown on figure 6. The curves represent annual capacity in metric tons of antimony over the 10-yr period. The 1985 potential capacity at a production cost of \$1.15/lb Sb or below (including a 15-pct DCFROR) was estimated at 19,000 mt Sb, increasing to 21,400 mt Sb at a production cost of \$2/lb Sb. The curves on figure 6 show a decline in capacity in 1986 and 1993 because of depletion of certain resources. After 1995, the decline is more rapid. Some of the estimated decline could be counteracted by an expansion of production capacities of the remaining producers, which have large demonstrated resources, although such an expansion would effectively shorten their producing lives. In addition, some of the mines may be able to produce from inferred resources.

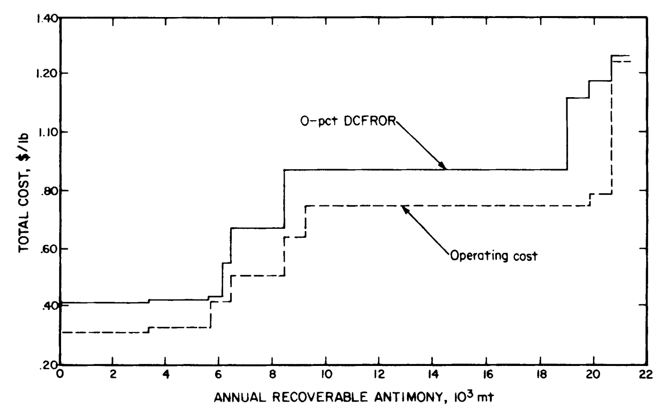

Figure 5.—1985 antimony capacity from producing mines in MEC's (January 1985 U.S. dollars; 0-pct DCFROR included).

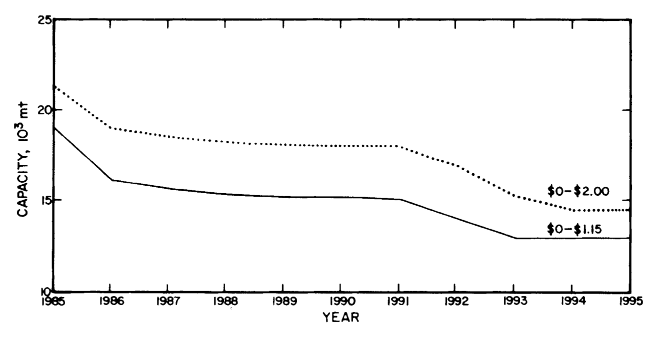

Figure 6.—Potential annual antimony production from producing mines in MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

CONCLUSIONS

Total demonstrated resources included in this study contain 460,000 mt of contained antimony, of which 63 pct are in Latin America and Africa. Of this total, 284,000 mt is recoverable as antimony metal. Based on all the properties, at a production cost of $1.30/lb Sb or below, a total of 235,000 mt Sb is available. In 1985, the 10 producers could have produced 21,400 mt at less than $1.30/lb (compared to 28,000 mt produced in that year).

The United States, with its high degree of industrial activity, normally imports and consumes 30 to 50 pct of the total annual mine production from the MEC's. With such a small percentage of the MEC's resources located in the United States, the United States will probably continue to rely on foreign antimony to satisfy domestic industrial requirements. The most probable sources of U.S. antimony supply would be Mexico, Bolivia, Republic of South Africa, and China.

Even though byproduct and secondary antimony supply a major part of the antimony production, these sources are not analyzed because there is no definable resource base. To gain a better understanding of the total availability, additional information on byproduct and secondary antimony is needed.

REFERENCES

1. Palencia, C.M., and C.P. Mishra. Antimony Availability—Market Economy Countries. A Minerals Availability Appraisal. BuMines IC 9098, 1986, 20 pp.

2. Plunkert, P.A. Antimony. Ch. in Minerals Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 33-42.

3. ______. Antimony. Ch. in BuMines Minerals Yearbook 1984, v. 1, pp. 109-118.

4. ______. Antimony. Sec. in BuMines Mineral Commodity Summaries, 1986, pp. 10-11.

5. Miller, M.H. Antimony. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 45-49.

6. National Materials Advisory Board. Trends and Usage of Antimony. NAS, NMAB-274, Dec. 1970, 113 pp.

7. U.S. Bureau of Mines (With the cooperation of the Geological Survey). Antimony. MSI, 1951, pp. I-9, IV-29.

ASBESTOS

INTRODUCTION

Asbestos is a term applied to six naturally occurring minerals that are exploited commercially for their desirable physical properties, which are in part derived from their asbestiform habit. The characteristics that render this group of minerals commercially important include their fibrous structure, tensile strength, fiber length, and resistance to high temperature and chemical attack.

The main use of asbestos is in the manufacture of construction materials, primarily in the form of asbestos cement pipe and sheet, flooring products, and roofing products. Other major end uses are in friction materials, paper, coatings and compounds, packing and gaskets, textiles, and plastics.

For purposes of product grading, the major industry standard was developed by the Quebec chrysotile asbestos industry, whereby asbestos is classified and valued according to fiber length, from the longest (No. 1) to the shortest (No. 9). Major uses by fiber grade are shown on table 1.

This study addresses the availability of chrysotile, crocidolite, and amosite asbestos of grades 3 through 7. Although crocidolite and amosite are not classified according to the chrysotile standard, they will be referred to in chrysotile grade equivalents. None of the properties evaluated produces grades 1 or 9; only one property produces grade 2 and one property produces grade 8, so no discussion of these grades is provided. Availability of grades 3 through 7 is presented on total and annual curves in terms of dollars per metric ton of fiber. A price proportioning routine was used in the analysis.

Prices of asbestos fiber are normally negotiated, and the details are seldom available publicly. Absolute prices of a particular grade fiber may vary widely, depending on the market forces relative to supply and demand. The f.o.b. mine, Quebec, prices established by a major mining company acting as price leader, historically generally set the pattern for world asbestos prices. The prices used in this analysis, based largely on these producer (or posted) prices, have not changed appreciably for several years.

Most of the information presented in this chapter is an updated summary of Bureau of Mines Information Circular 9036 "Availability of Asbestos—Market Economy Countries, A Minerals Availability Program Appraisal" (*1*).[1] Additional information on the domestic and foreign asbestos industry is available from other Bureau of Mines publications (*2-4*).

Table 1.—Quebec standard grades and uses of asbestos (*2*)

Grade	Length specifications, mm	Uses
Long fiber:		
No. 1 crude	19	Textiles.
No. 2 crude	9.5-19	Textiles, insulation.
No. 3	6 - 9.5	Textiles, packaging, brake linings, clutch facings, electrical, high-pressure and marine insulation.
Medium fiber:		
No. 4	3 - 6	Asbestos-cement pipe.
No. 5	3 - 6	Asbestos-cement pipe and sheets, molded products, paper products, brake linings and gaskets.
No. 6	3	Asbestos-cement products, brake linings and gaskets, plaster, backing for vinyl sheets.
Short fiber:		
No. 7	3	Molded brake linings and clutch facings, plastics, filler in vinyl and asphalt floor tiles, asphalt compounds, and caulking compounds, paints and drilling mud additive.
No. 8	([1])	Do.
Very short fiber: No. 9	([1])	Rock ballast, asphalt paving aggregate, landfill.

[1]Weight specifications.

GEOLOGY AND RESOURCES

GEOLOGY

Most chrysotile deposits, including the majority of those evaluated for this study, are located in serpentinized ultramafic (peridotite) bodies. In the important producing Thetford mines area of eastern Quebec, Canada, the fibrous form of serpentine is found in veinlets, mostly less than 3 cm wide, in dark green massive serpentine. The veinlets and wall rock have the same chemical composition and vary only in physical character.

Among the amphibole asbestos deposits, only those in the northern Cape and northern Transvaal Provinces of the Republic of South Africa constitute an important resource. The Republic of South Africa deposits occur with banded ironstone of the Transvaal Supergroup. Both the crocidolite deposits, located in the northern Cape, and amosite deposits (northern Transvaal) are situated within the banded ironstone, although separated by over 600 km. This lateral persistence is evident within the banded ironstone itself, in which individual bands, even those a fraction of a millimeter in thickness, can be traced for several kilometers

[1]Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

without perceptible change. Additional geologic information is available from U.S. Geological Survey Professional Paper 820 (*5*).

RESOURCES

Figure 1 shows estimates of world identified and hypothetical asbestos resources, reserve base, and demonstrated resources evaluated for this study. The Republic of South Africa's evaluated resources are substantially lower than the reserve base because only demonstrated resources in the important producing properties or past producers were evaluated. The northern Cape crocidolite field has a large number of relatively insignificant deposits that were not evaluated for this study.

Table 2 summarizes resource information for evaluated properties. Among producers, Canada has the largest demonstrated resource, more than 28 million mt, all chrysotile. Of the total for South African producers, 55 pct is crocidolite, 30 pct amosite, and 15 pct chrysotile. The unusually high grade of U.S. producers is due to the resource of the New Idria intrusive which, locally, grades more than 50 pct fiber, all grade 7. The resource potential on the New Idria intrusive is virtually unlimited, and demonstrated resources included in this evaluation apply only to the legal boundaries of the evaluated properties.

Pertinent information concerning evaluated market economy country (MEC) properties is presented in table 3.

Table 2.—Summary of world demonstrated asbestos resources evaluated for this study, as of January 1985

Country	Total ore treated, 10^3 mt	In situ grade, pct	Total recoverable fiber, 10^3 mt
Canada:			
Producers	554,699	5.12	28,387
Nonproducers	226,288	3.97	9,297
South Africa, Rep. of:			
Producers	22,826	10.80	2,219
Nonproducers	4,211	4.70	198
United States:			
Producers	32,688	18.51	3,488
Nonproducers	60,897	8.64	3,975
Zimbabwe: Producers	91,543	5.46	4,829
Other:			
Producers[1]	351,348	3.68	12,843
Nonproducers[2]	37,581	W	2,185
Total	NAp	NAp	67,421

W Withheld to avoid disclosing company confidential information.
NAp Not applicable.
[1]Includes Australia, Brazil, Colombia, Cyprus, Greece, Italy, and Swaziland.
[2]Includes Mexico.

Table 3.—MEC properties included in this study

Country and property name	Ownership	Current status[1]	Mining method[2]	Fiber grades	Fiber type[3]
Australia: Woodsreef	Woodsreef Mines Ltd.	P	S	4	Ch
Brazil: Cana Brava	S.A. Mineracao de Amianto	P	S	4-6	Ch
Canada:					
Abitibi	Abitibi Asbestos & Brinco Ltd.	N	U	4-7	Ch
Baie Verte	Baie Verte Mines Inc.	P	S	4-6	Ch
Bell	La Soc. Nat. de l'Amiante	P	U	3-7	Ch
Black Lake	Lac d'Amiante du Quebec Lte	P	S	3-7	Ch
British Canadian	La Soc. Nat. de l'Amiante	P	S	3-7	Ch
Cassiar	Cassiar Mining Corp	P	C	3-6	Ch
Jeffrey	Johns Manville Asbestos Inc.	P	C	4-7	Ch
King-Beaver	La Soc. Nat. de l'Amiante	P	C	3-7	Ch
Midlothian	United Asbestos Inc.	PP	S	4-7	Ch
Penhale	La Soc. Nat. de l'Amiante	N	U	3-7	Ch
Roberge Lake	McAdam Mining Corp. Ltd.	N	S	5-7	Ch
Colombia: Las Brisas	Minera Las Brisas S.A.	P	S	4,6	Ch
Cyprus: Amiandos	Cyprus Asbestos Mines Ltd.	P	S	3-4	Ch
Greece: Zidani	Asbestos Mines of No. Greece	P	S	4-6	Ch
Italy: Balangero	Amiantifera di Balangero SpA	P	S	4-8	Ch
Mexico: Pegaso	Cia. Minera Pegaso S.A.	N	S	5-7	Ch
South Africa, Republic of:					
Danielskuil	General Mining Union Corp	PP	U	3-4	Cr
Elcor	..do	P	U	3-4	Cr
Emmarentia	Lonhro Ltd.	P	U	3-4	Cr
Klipfontein	General Mining Union Corp.	P	U	3-4	Cr
Msauli	..do	P	U	4-7	Cr
Penge	..do	P	U	3-4	A
Pomfret	..do	P	U	3-4, 6	Cr
Riries	..do	PP	U	3-4	Cr
Senekal	..do	PP	U	5-7	Ch
Wandrag	Lonhro Ltd.	P	U	3-4	Cr
Whitebank	General Mining Union Corp.	P	U	3-4	Cr
Swaziland: Havelock	Turner & Newall PLC and the Swazi nation	P	U	4-5	Ch
United States:					
Alaska: Slate Creek	Tanana Asbestos Corp. and GCO Minerals	N	S	4	Ch
California:					
Calaveras	Calaveras Asbestos Corp.	P	S	4-6	Ch
Christie	Tenneco Oil Co.	PP	S	7	Ch
Santa Rita	KCAC Inc.	P	S	7	Ch
Vermont: Lowell	Vermont Asbestos Co, Inc.	P	S	3-7	Ch
Zimbabwe:					
Gath's	Turner & Newall PLC	P	C	4-5	Ch
King	..do	P	U	4-5	Ch
Shabanie	..do	P	U	2-6	Ch

[1]P, producing; PP, past producer; N, nonproducing.
[2]U, underground; S, surface; C, combined methods.
[3]A, amosite; Ch, chrysotile; Cr, crocidolite.

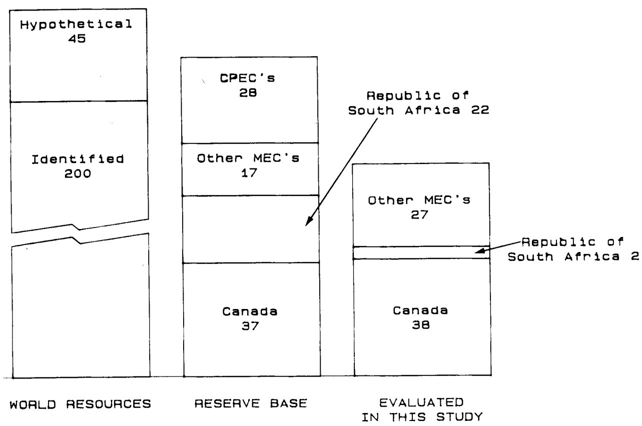

Figure 1.—Estimates of world asbestos resources (million metric tons of fiber).

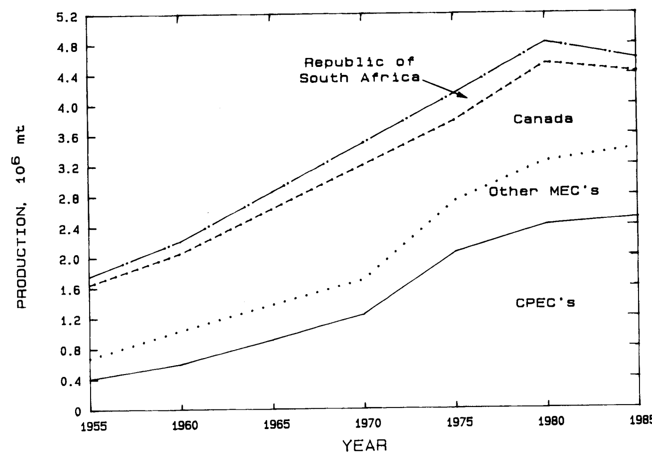

Figure 2.—World asbestos production, 1955-85.

U.S. AND WORLD HISTORICAL PRODUCTION

Figure 2 shows world production trends in 5-yr intervals from 1955 through 1985. Production from the important MEC producers, Canada and the Republic of South Africa, are included separately. Total world production has increased from 1.8 million mt in 1955 to 4.6 million mt in 1985. The major growth in production has occurred within centrally planned economy countries (CPEC's), primarily the Soviet Union and China.

The United States is a minor asbestos producer, accounting for only about 1 pct of world production in 1985. While U.S. production has declined by a factor of nearly 2 since 1970, for the years shown, the U.S. has never accounted for more than 4 pct of total world production.

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

The methods used in asbestos mining are largely dictated by the geological occurrence, type, strength, and other physical characteristics of the host rock. Selection of mining method must recognize ore body value and fiber length distribution, extent and value of dilution zone, rock mass classification, contaminants, and fiber degradation. Indiscriminate selection of mining method can result in high ore loss, high dilution, and expensive support maintenance costs.

Canadian asbestos occurs in irregular veins or veinlets in massive serpentine deposits that are laterally and vertically extensive, so that large-scale, highly mechanized open pit methods have been widely employed. Crocidolite operations in the northern Cape of the Republic of South Africa have typically consisted of a number of small mines using highly labor intensive methods to provide hand-sorted ore to feed centrally located mills. The three major mining methods used in the Cape operations are room-and-pillar, semishrinkage stoping, and post-pillar with hydraulic fill (cut-and-fill).

PROCESSING

The concentration process for asbestos is unique in that it involves separation of a fibrous mineral from a massive form of the same mineral. Thus, neither chemical composition nor specific gravity can be used as the basis for separation, which is done strictly by mechanical means.

A most important principle governing asbestos milling is separation of fiber from host rock with a minimum of fiber breakage. This is because the value depends largely on the fiber length. Modern mills are designed to remove the fiber by aspiration (air suction) after each crushing stage so that destruction of fiber is minimized.

Essentially, all dry milling processes are very similar, relying exclusively on mechanical processes to separate and recover fiber. Asbestos milling consists essentially of coarse crushing, drying, and recrushing in stages, each step followed by screening and air separation of fiber from rock.

NEW TECHNOLOGY

A recent development in the treatment of asbestos is introduction of wet milling process. Besides improving recovery of fiber from certain deposits, an especially promising aspect of a wet mill processing is that it lowers fiber dust levels in the workplace. This is a particularly important concern in the mining and processing of asbestos. Woodsreef Mines Ltd. has operated a pilot-plant wet-process mill. The Santa Rita operation in California also utilizes a wet process in part of its milling plant, but the particular details of that process are not widely available.

PRODUCTION COSTS

Weighted-average operating costs for evaluated properties are shown in table 4 and figure 3. Because nearly all operations do or would produce several fiber grades, the only meaningful cost comparison is on the basis of cost per metric ton of recovered fiber.

The anomalously low mine cost for the United States producers is because of the very high grade at Santa Rita (locally, up to 50 pct fiber). Republic of South Africa producers, although all underground operations, have a relatively low mine and mill cost because of high grades also (average 10.8 pct fiber). All but one of the properties in the "other" producer grouping are surface operations, which results in low mine cost but mill costs that are more comparable to that of other countries, notably Canada and the United States.

Among nonproducers, the relatively low mine cost figure for the United States is because of the Christie deposit, which, like Santa Rita, is located on the New Idria intrusive and has a very high grade.

AVAILABILITY

The amount of asbestos fiber potentially available from evaluated properties for grades 3 through 7 is shown on graphs in this section. Because only one property contains grade 2 and one property contains grade 8, no curves or availability discussion are included for these grades. Crocidolite and amosite availability are discussed where appropriate.

TOTAL RECOVERABLE

Grade 3

There is a total of 2.03 million mt of grade 3 fiber potentially available from evaluated producers (fig.4). There is an additional 145,000 mt contained in nonproducing properties, of which 61,000 mt is Republic of South Africa crocidolite. Canadian producing properties contain about 548,000 mt of chrysotile fiber of grade 3. About 38 pct of the total of 2.03 million mt is contained in Republic of South Africa producing properties, all of which, except for Penge, produce crocidolite. Republic of South Africa crocidolite and amosite properties account for 75 pct of the total amount of grade 3 fiber potentially available at a cost of less than $660/mt. About 87 pct of the total amount of grade 3 fiber contained in all producing properties has a cost lower than the January 1985 price of $1,600/mt.

Grade 4

There is a total of 15.02 million mt of grade 4 fiber potentially available from evaluated producers, of which 1.08 million mt is Republic of South Africa crocidolite and amosite (fig. 5). About 9.30 million mt of chrysotile is contained in producing properties in Canada. There is an additional 4.30 million mt contained in nonproducing properties, of which only 49,000 mt is crocidolite. Nearly 63 pct of the total of 15.02 million mt is available at a cost lower than or equal to the January 1985 price of about $1,040/mt.

Grade 5

There is a total of 10.80 million mt of grade 5 fiber potentially available from producing operations, of which more than 90 pct has a cost equal to or lower than the January 1985 price of about $590/mt (fig.6). All of the grade 5 fiber is chrysotile. Approximately 4.80 million mt is in Canadian producing properties, and 1.40 million mt in Zimbabwean producing properties. There is an additional 4.20 million mt contained in nonproducing properties.

Table 4.—Asbestos production costs for producing and nonproducing operations in selected MEC's

(All costs are in January 1985 U.S. dollars per metric ton fiber)

Country	Operating cost: Mine	Operating cost: Mill	Transportation cost	Net operating cost	Recovery of capital[1]	0-pct DCFROR: Taxes and royalties[2]	0-pct DCFROR: Total cost[3]	15-pct DCFROR: Taxes and royalties[4]	15-pct DCFROR: Return on investment[5]	15-pct DCFROR: Total cost[6]
PRODUCERS										
Canada	178	154	117	449	45	7	501	32	32	558
South Africa, Republic of	110	96	107	313	32	6	351	17	16	378
United States	30	125	0	155	24	1	180	5	10	194
Zimbabwe	196	94	102	392	71	0	463	0	19	482
Other[7]	74	134	47	255	29	1	285	14	21	319
NONPRODUCERS										
Canada	164	169	160	493	100	13	606	117	115	825
South Africa, Republic of	168	182	77	427	56	2	485	48	66	597
United States	102	231	139	472	100	12	584	125	252	949

[1]Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure as of January 1985, and investments required over the life of the operation.
[2]Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 0-pct DCFROR.
[3]Equal to the sum of net operating costs, taxation, and capital recovery determined at a 0-pct DCFROR.
[4]Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 15-pct DCFROR.
[5]The revenue increase necessary per metric ton to obtain a 15-pct DCFROR.
[6]Equal to the sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery *plus* the per metric ton increase necessary to provide a 15-pct DCFROR after taxation.
[7]Includes Australia, Brazil, Colombia, Cyprus, Greece, Italy, and Swaziland.

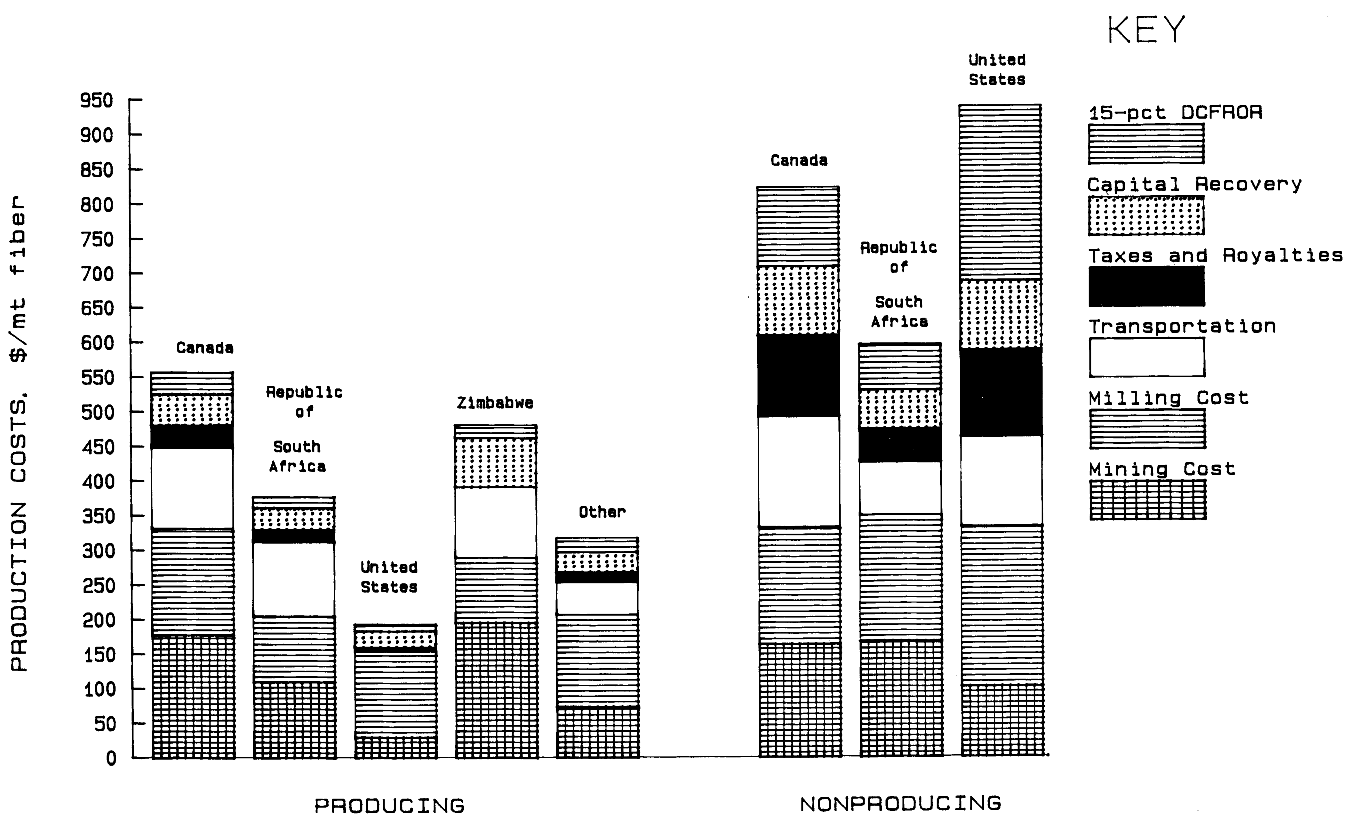

Figure 3.—Asbestos production costs for selected MEC's (January 1985 U.S. dollars).

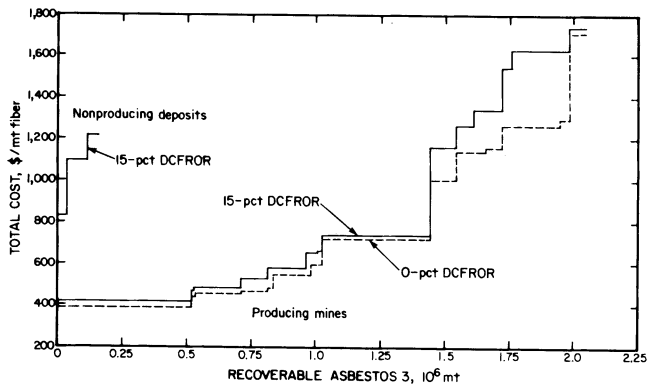

Figure 4.—Potential total asbestos 3 available from MEC's (January 1985 U.S. dollars).

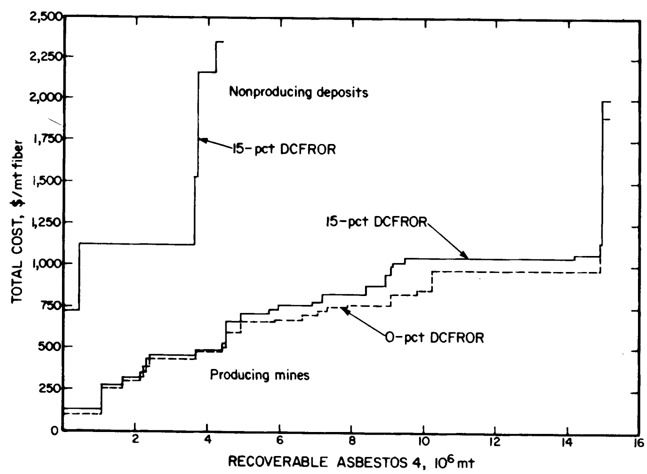

Figure 5.—Potential total asbestos 4 available from MEC's (January 1985 U.S. dollars).

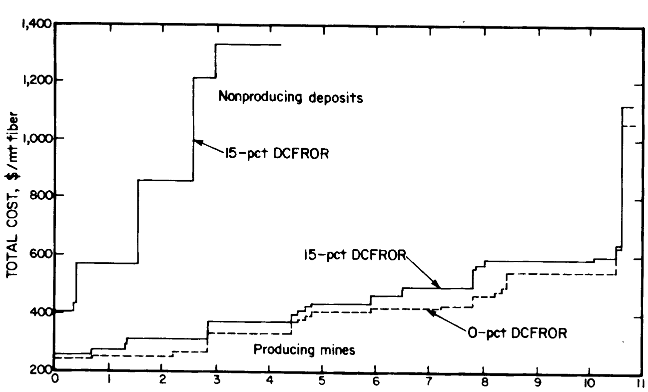

Figure 6.—Potential total asbestos 5 available from MEC's (January 1985 U.S. dollars).

Grade 6

There is a total of 9.72 million mt of grade 6 fiber potentially available from producing operations, all chrysotile, of which 55 pct is in Canadian producing properties (fig. 7). There is an additional 3.50 million mt of chrysotile in nonproducing properties. Approximately 60 pct of the total amount in producing properties is potentially available at a cost lower than the January 1985 price of about $420/mt.

Grade 7

There is a total of 12.80 million mt of grade 6 fiber, all chrysotile, in producing properties, of which more than 95 pct is potentially available at a cost lower than the January 1985 price of $190/mt (fig.8). Sixty-five percent is in Canadian properties. There is an additional 3.50 million mt of chrysotile in nonproducing properties, of which two-thirds is from Canada.

ANNUAL CAPACITY

The following graphs show the amount of fiber potentially available annually from 1985 through 1995 from producers for the various fiber grades. Except for properties for which adequate information was available regarding likely future production levels, the amount of fiber available on an annual basis represents the capacity production level. Unfortunately, individual fiber grade production statistics are not available for most countries, so a comparison of actual production with data resulting from this evaluation is not possible.

This evaluation included only currently demonstrated resources. Several of the properties have demonstrated resources that are not sufficient to last to the end of the time period shown (1995). However, based on geology of the deposits and familiarity with the operations, it is likely that many of the deposits have additional resources at lower levels of probability that will replace currently demonstrated resources as they are depleted. Therefore, it is unlikely that potential annual availability of any fiber grade will decline substantially within the time period shown.

Grade 3

In 1985, a total of about 157,000 mt of grade 3 fiber was available from producers, of which 91 pct was in properties with a cost lower than the January 1985 price of $1,600/mt (fig. 9*A*). There was an additional 18,000 mt potentially available from nonproducers. Annual production potential from producers reaches a peak of 165,000 mt in 1986, but declines to 102,000 mt by 1995. All Republic of South Africa fiber in the grade 3 range is amosite or crocidolite, which together account for 54 pct of the total grade 3 fiber available.

Grade 4

There was 675,000 mt of grade 4 fiber available in 1985, of which 82 pct was in properties with a cost equal to or lower than the January 1985 price of about $1,040/mt (fig. 9*B*). There was an additional 177,000 mt potentially available annually in nonproducing properties. Thirty-six percent of the amount of grade 4 fiber available from producers in 1985 was in the Republic of South Africa crocidolite and amosite producing properties. Annual availability peaks at 698,000 mt in 1986 and declines to 611,000 mt in 1995.

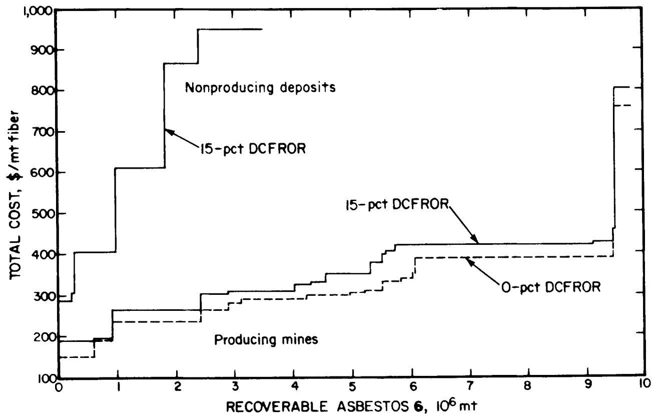

Figure 7.—Potential total asbestos 6 available from MEC's (January 1985 U.S. dollars).

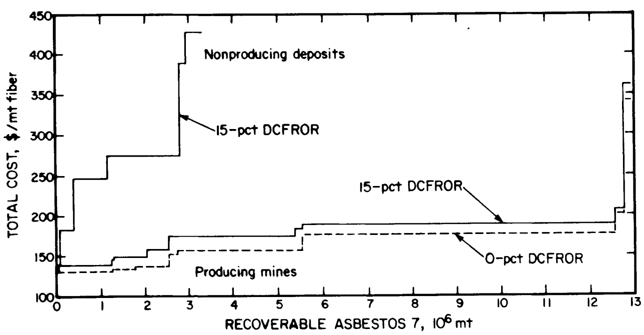

Figure 8.—Potential total asbestos 7 available from MEC's (January 1985 U.S. dollars).

Grade 5

There was a total of 456,000 mt of grade 5 fiber potentially available from producers in 1985, of which 87 pct was contained in properties with a cost lower than the January 1985 price of about $590/mt (fig. 9*C*). All of the fiber of grade 5 is chrysotile. Canadian properties account for 37 pct of the total amount of grade 5 fiber potentially available in 1985. There was an additional 180,000 mt potentially available in nonproducing properties, of which 77 pct was contained in Canadian properties.

Grade 6

There was a total of 391,000 mt of grade 6 fiber (all chrysotile) potentially available in 1985 from producers, and an additional 155,000 mt from nonproducing properties, of which 83 pct is in Canadian properties (fig. 9*D*). Among producers, 72 pct was contained in properties having a cost lower than the January 1985 price of about $420/mt. Annual availability peaks at 399,000 mt in 1986 and declines to 287,000 mt by 1995.

Grade 7

There was about 277,000 mt of grade 7 fiber, all chrysotile, potentially available in 1985 from producers, of which 92 pct was in properties having a cost lower than the January 1985 price of $190/mt (fig. 9*E*). There was an additional 142,000 mt in nonproducing properties, of which 76 pct is in Canadian properties.

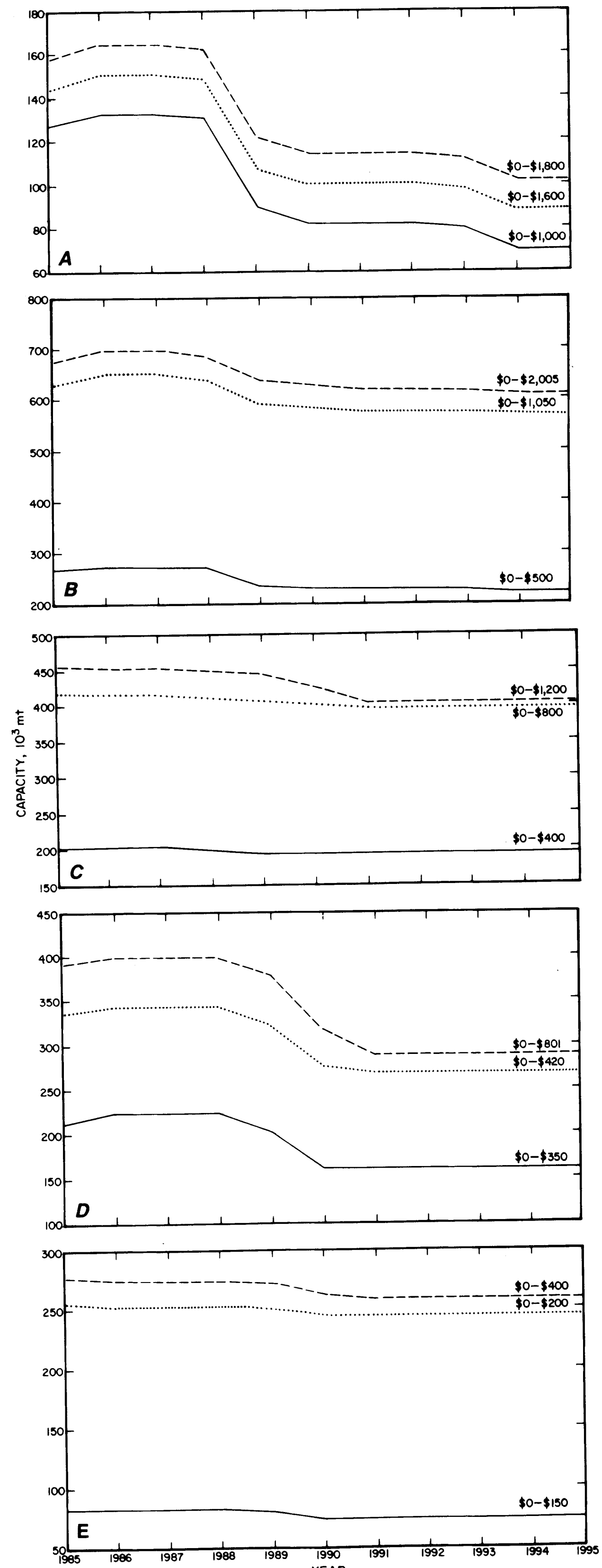

Figure 9.—Potential annual asbestos production from producing mines in MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars). *A*, asbestos 3; *B*, asbestos 4; *C*, asbestos 5; *D*, asbestos 6, E, asbestos 7.

CONCLUSIONS

The results of this analysis suggest that, in terms of total availability, demonstrated recoverable fiber in evaluated properties is adequate to sustain current production levels for the foreseeable future. In general, the amounts of grade 3 through 7 fiber potentially available on an annual basis are projected to decline over the next 10 yr owing to the exhaustion of currently demonstrated resources. However, most of the properties evaluated have substantial additional resources at lower levels of probability, and long-term adequacy of all fiber types is reasonably assured.

In view of the generally depressed state of the asbestos industry during the past several years, in part resulting from the growing concern relating to potential health hazards associated with the exploitation and use of asbestos fiber, it is virtually certain that factors other than geologic availability and potential of fiber will have the greatest impact on the long-term viability of the industry.

REFERENCES

1. Anstett, T. F., and K. E. Porter. Asbestos Availability—Market Economy Countries. A Minerals Availability Program Appraisal. BuMines IC 9036, 1985, 21 pp.

2. Clifton, R. A. Ch. in Mineral Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 53-64.

3. ______. Asbestos. Ch in BuMines Minerals Yearbook 1984, v. 1, pp. 119-127.

4. ______. Asbestos. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 14-15.

5. Shride, A. F. Asbestos. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 63-73.

BARITE

INTRODUCTION

By far, the largest use of barite ($BaSO_4$) is as a weighting material in oil and gas well drilling fluids (drilling mud). In 1983, over 96 pct of the barite consumed in the United States and 85 to 90 pct of that consumed in Europe was as drilling mud.

The chemical, filler, and ceramic industries utilize both precipitated salts and natural ore in a variety of applications; however, these industries use only a small amount of the total production. This type of barite generally must have a minimum grade of 95 pct $BaSO_4$.

On a worldwide basis, barite is distributed by a relatively few service companies. These companies market barite as a component of their drilling engineering services.

Barite is a soft, chemically inert mineral of the composition of $BaSO_4$ having a specific gravity of 4.5. The commercially exploitable occurrences of barite are relatively widespread; in the 1979-85 period, 45 countries recorded barite production. Barite is generally easy to concentrate by simple gravity methods; relatively small deposits can be developed and mined with comparatively small preproduction costs.

Market specifications of barite concentrates for drilling mud are established by the American Petroleum Institute (API). The principal specification is that the material have a specific gravity of at least 4.2 (about 91 pct $BaSO_4$). Lower grades of less than 4.2 specific gravity are marketed, but they must be blended with higher grades to maintain the minimum specific gravity. Chemical- and filler-grades generally must contain at least 95 pct $BaSO_4$ and be free of iron oxide red discoloration. The availability analysis in this chapter is in terms of mud-, chemical-, and filler-grade barite concentrates.

In 1984, the Bureau of Mines investigated the availability of barite ($BaSO_4$) from 34 (combined to 31 operations) U.S. and 39 non-U.S. mines and deposits (36 operations) in 17 market economy countries (MEC's) (*1*).[1] The evaluation assessed the availability of drilling-mud-grade barite and concentrates consumed by the chemical and filler industries. About 90 pct of the U.S. barite is produced in Nevada. This summary is an update of the 1984 study; the changes affecting the industry since 1984 are discussed below.

Relative to the 1984 study, three of the U.S. mines that were shut down in 1982 started up in late 1984 on a limited scale, and the mine in Greece was placed on standby.

A small operation began operating in Gabon (about 30,000 mt/yr); in Mexico, Pemex began production from a new mine in Michoacan; and a "large" resource discovery was reported in Indonesia. Because of a lack of available data, these are not included in the update.

Three Illinois fluorspar operations producing barite as a byproduct were depleted and closed by the end of 1985 and are not included in this update, leaving a total of 64 operations.

The U.S. industry still experienced increasing pressure from imports from China. These imports are increasingly in the form of fine ground barite, which is resulting in a cutback in production or closure of the U.S. grinding plants. Gulf coast shipments from Nevada in 1985 were largely the result of fulfilling previous contracts. It is thus expected that in the immediate future (at least 1986-87) there will be very little Nevada barite shipped to the gulf coast, thereby drastically reducing the Nevada (and U.S.) output. More information on the barite industry can be found in various other publications (*2-5*).

GEOLOGY AND RESOURCES

GEOLOGY

Barite occurs principally in bedded, vein, and detrital (residual) deposits. There are also relatively minor occurrences associated with metallic sulfide and fluorite deposits. In terms of world barite resources, bedded deposits are by far the most important. They generally occur as stratiform beds, lenses, or discontinous horizons that are conformable with the enclosing rocks (*5*).

The bedded deposits often involve groups of several lenses or horizons. They usually exhibit a gradation from higher grade material near the center of the lens, to lower grades toward the extremities. This gradation occurs stratigraphically as well; that is, the barite content generally decreases nearer the enclosing rock contact. Vein deposits are generally hydrothermal in origin and smaller than bedded deposits.

Detrital (residual) deposits of barite are formed in a clay-bearing or clay residium that results from surficial weathering. These are normally low grade (6 to 10 pct $BaSO_4$) and have been exploited in Georgia and Missouri mainly for chemical- and filler-grade barite.

[1]Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

RESOURCES

In many cases, resources are not well defined, since advance exploration normally defines only a few years of production. Extreme fluctuations in drilling activities, on which barite consumption depends, make it difficult for a company to justify the expense of establishing a long-range resource. In addition, the development of a barite mine normally requires relatively little initial capital expenditure, and thus, a large resource is not needed to recover initial invested capital.

Demonstrated resource estimates were updated to January 1985 for this summary. Total evaluated U.S. resources amount to 53 million mt of contained $BaSO_4$, and foreign properties total 74 million mt. The total world reserve base amounts to 454 million mt of contained $BaSO_4$ (139 million mt in the evaluated countries) (2). Evaluated demonstrated resources, reserve base, and world resource estimates (2) are shown in figure 1. Evaluated resources, by country, are shown in table 1. Barite mine and deposit data are shown in table 2.

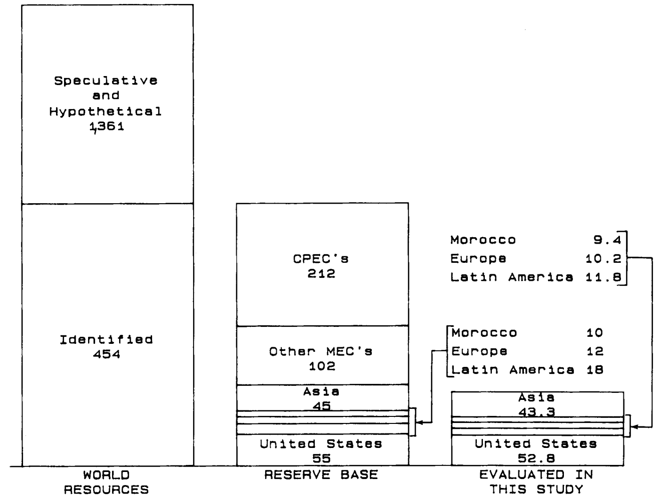

Figure 1.—Estimates of world barite resources (million metric tons of contained barite).

Table 1.—Summary of MEC demonstrated barite resources evaluated for this study, as of January 1985, thousand metric tons

Country	Demonstrated: In situ material	Demonstrated: Contained $BaSO_4$	Demonstrated: Recoverable product	Identified: In situ material	Identified: Contained $BaSO_4$
Belgium	498	349	305	823	576
Brazil	644	605	565	644	605
Chile	80	69	62	4,000	3,320
France	5,628	2,124	1,551	9,000	3,490
Germany, Federal Rep. of	1,884	899	599	3,845	1,634
Greece	573	395	332	1,000	690
India	34,307	31,562	30,876	71,400	60,732
Ireland	1,514	1,347	1,514	2,400	2,256
Italy	9,285	1,890	1,091	23,150	4,323
Mexico	9,982	7,910	6,370	20,000	16,000
Morocco	10,308	9,445	8,519	23,210	17,633
Pakistan	2,369	2,109	2,000	5,000	4,250
Peru	3,556	3,201	3,225	8,030	7,227
Spain	5,080	1,167	909	7,475	1,536
Thailand	14,247	9,665	8,069	22,200	14,616
United Kingdom	4,000	2,000	2,050	5,998	3,840
United States:					
Arkansas[2]	11,208	5,104	4,090	18,853	8,524
Georgia	24,002	2,138	1,639	25,800	2,358
Missouri	90,020	6,820	3,453	91,920	7,011
Nevada	56,077	38,780	33,933	84,131	52,305
Total	181,307	52,842	43,115	220,704	70,198
Grand total	285,262	127,579	111,152	428,879	212,926

[1]Resources estimated for the purpose of analysis.
[2]Includes Illinois and Washington.

Table 2.—MEC barite properties included in this study

Country and property name	Current status	Mining method	Beneficiation mdethod	End use products	Annual capacity,[1] mt
Belgium: Fleurus	Producing	Open pit	Flotation, gravity[2]	Drilling	40,000
Brazil: Camamu	..do	..do	Gravity	..do	51,900
Chile: Baritex/Resguardo	..do	Open stope	Gravity, hand sorting.	..do	95,600
France:					
Barytine de Chaillac	..do	Open pit	Flotation	Chemical[3]	101,600
Rossignol	..do	Shrinkage	Heavy media	Drilling, CaF_2	9,000
Total					110,600
Germany, Federal Republic of:					
Clara	..do	Sublevel stoping	Flotation	Drilling, chemical, CaF_2	W
Dreislar	..do	Filled stoping	..do	Chemical	W
Total					63,600
Greece: Mykonos Mykobar	Shut down	Shrinkage	Gravity	Drilling	42,200
India:					
Mangampet	Producing	Open pit	Hand sorting	..do	360,000
Tiffin	..do	..do	..do	..do	24,000
Total					384,000
Ireland: Ballynoe	..do	..do	Crushing	..do	250,000
Italy:					
Barego	..do	Sublevel stoping	Gravity	..do	35,100
Mineraria Silius	..do	..do	Flotation	Drilling, chemical, CaF_2, Pb conc.	25,800
Montego	..do	Overhand shrinkage	..do	Drilling, chemical, Pb and Ag conc.	49,500
Total					110,400

See footnotes at end of table.

Table 2.—MEC barite properties included in this study—Continued

Country and property name	Current status	Mining method	Beneficiation mdethod	End use products	Annual capacity,[1] mt
Mexico:					
Barita de Santa Rosa	..do	Room and pillar	Hand sorting, flotation.	Drilling	79,100
Barita de Sonora	..do	Open pit	..do	..do	106,700
Cobachi	Developing	..do	..do	..do	202,200
Total					388,000
Morocco:					
Jbel Ihroud	Producing	..do	Gravity	..do	36,900
Seksaoua	..do	Shrinkage	Hand sorting	..do	111,400
Tessaout	..do	Room and pillar	..do	..do	17,100
Zelmu	..do	Open pit	Crushing	..do	235,200
Total					400,600
Pakistan: Bolan Barytes	..do	..do	Hand sorting	..do	26,200
Peru:					
Chagla (Tingo Maria)	Explored	..do	Crushing	..do	125,000
Graciela-Juanita-Minera Barmine	Producing	Open pit-cut and fill	Hand sorting, flotation.	Drilling, Pb and Zn conc.	254,000
Total					379,000
Spain:					
Guillermin-San Fernando	..do	Sublevel stope	Flotation	Drilling, Pb conc	35,500
La Carolina	..do	Open pit	Gravity	Drilling	32,400
Total					67,900
Thailand:					
American Thai (Attbar, Thung Wa)	..do	..do[4]	Washing	..do	68,000
Ban Hin Khao	..do	..do	..do	..do	57,800
Khao Mai Phai	..do	..do	..do	..do	40,000
Nidhi	..do	..do	Hand sorting	..do	60,000
Oriental Gold	..do	..do	..do	..do	30,000
Siam Barite	..do	..do	Washing	..do	23,600
STA	..do	..do	Hand sorting	..do	45,000
Tip	..do	..do	Washing	..do	40,300
Total					364,700
United Kingdom:					
Aberfeldy	Explored	Shrinkage	Crushing	..do	150,000
Derbyshire	Producing	Open pit, sublevel stope	Flotation	Drilling, CaF_2, Pb conc.	11,800
Total					161,800
United States:					
Arkansas: Fancy Hill, McKnight	Shut down	Open pit	Flotation	Drilling	178,000
Georgia:					
New Riverside	Producing	..do	..do	Chemical	28,000
Paga	..do	..do	..do	..do	32,800
Total					60,800
Illinois: Denton	..do	Room and pillar	..do	Chemical Pb and Zn conc, CaF_2.	3,400
Missouri:					
Apex, Mineral Point	Shutdown	Open pit	Jig, flotation	Drilling	73,600
Cadet	Producing	..do	Jig	..do	22,800
Dresser No. 4	Shut down	..do	Jig	..do	13,600
Dresser No. 10	..do	..do	Jig	..do	18,100
Kingston	Producing	..do	Jig	Drilling, chemcial	48,400
Old Mines	Shut down	..do	Jig	Chemical	8,800
Richwoods	Producing	..do	Jig	Drilling, chemical	45,900
Stone Spring	Explored	..do	Jig	Drilling	35,400
Sun	Shut down	..do	Jig	Chemical	17,600
Total					284,200
Nevada:					
Ann	Explored	..do	Jig	Drilling	99,000
Argenta	Producing	..do	Jig	..do	265,600
East Northumberland	Shut down	..do	Jig	..do	83,600
Easy Miner	Explored	..do	Jig	..do	67,100
Fish Creek	..do	..do	Flotation	..do	169,200
Greystone	Producing	..do	Jig	..do	337,800
Heavy Spar	Shut down	..do	Jig	..do	50,100
Kay	Explored	..do	Jig	..do	70,100
Lakes	..do	..do	Jig	..do	344,000
Mountain Springs	Producing	..do	Jig	..do	275,700
P & S	Shut down	..do	Flotation	..do	184,900
Rossi	Producing	..do	Jig	..do	295,400
Snoose, Big Ledge, Jungle	Shut down	..do	Jig	..do	74,200
Stormy Creek	..do	..do	Jig	..do	128,000
Total					2,444,700
Washington: Flagstaff Mtn., Bruce Creek.	Developing	..do	Flotation	..do	123,400
U.S. total					3,094,500
Grand total					6,031,000

[1]Estimated 1984 potential product capacity; proposed for nonproducers.
[2]Various gravity methods (jig, tables, spirals, etc.).
[3]Includes both chemical and filler grades.
[4]Some mines use gophering on narrow veins when open pit limits are reached.

U.S. AND WORLD HISTORICAL PRODUCTION

For many years the United States has been the world's leading producer of barite; however, in 1983, China became the leading producer at about 1 million mt. World production statistics from 1950 through 1985 are shown in figure 2. Production levels were affected by the 1982 worldwide recession, showing a more than 30-pct decrease in production between 1981 and 1983.

Because of the amount of its oil and gas drilling, the United States is by far the world's largest consumer of barite and has always been a substantial importer; in terms of footage drilled, the United States has a level of drilling about four times that of the rest of the world combined. China supplied about 50 pct of the U.S. imports in 1985, and even with the downturn in U.S. consumption from 1981 to 1983, imports from China increased at the expense of U.S. production and imports from other countries. Thus, the cheaper barite from China is causing decreases in production from other countries as well as the United States.

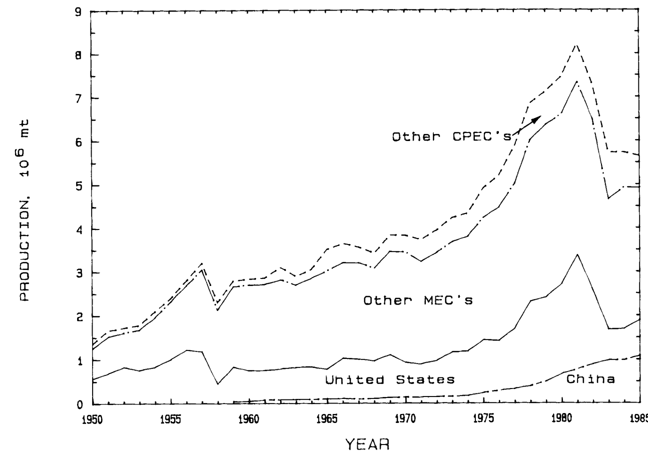

Figure 2.—World barite production, 1950-85.

EXTRACTION AND PROCESSING TECHNOLOGY

Barite mining operations are relatively small; the largest mine evaluated had an estimated capacity of 360,000 mt; the average capacity for all 64 operations is about 94,000 mt/yr.

MINING

Open pit mining is used to mine approximately 88 pct of the demonstrated resources evaluated. The most highly mechanized mining is in the United States and Europe; in low labor cost areas the operations are more labor intensive.

Stripping ratios average between 4:1 and 5:1 for the U.S. hard-rock mines (Arkansas, Nevada, and Washington) and 2:1 and 3:1 for the non-U.S. mines. Detrital ores in Georgia and Missouri require little or no stripping and no blasting.

PROCESSING

Barite is sold as both direct shipping ore and as beneficiated material. Direct shipping ore is sometimes upgraded slightly by crushing and selective screening or a small amount of hand sorting.

The most common type of beneficiation is jigging; in some operations tabling or flotation is used to treat jig tailings to improve recoveries, and in a few cases, flotation is used as a primary method. In terms of total capacity of the deposits evaluated, 26 pct is direct shipping (includes hand sorting or crushing and selective screening); 48 pct is treated by jigs or other gravity methods and 26 pct is upgraded (at least in part) by flotation, including byproduct barite recovery from base metal and fluorspar flotation plants.

For use in drilling, crude barite is ground to about 325 mesh in Raymond-type roller mills. Preferably, the barite is ground near where it is to be used; increased costs for shipping the ground barite (as opposed to unground) are generally more than the cost of grinding. Since grinding is an add-on cost, and is nearly always done by the buyers, the grinding costs are not included in the evaluations.

PRODUCTION COSTS

Weighted-average cost elements for mud-grade operations are shown by country or region in figure 3. All costs are in terms of 1985 U.S. dollars per metric ton of recoverable barite concentrate. Table 3 lists a further breakdown of the costs by category for each of the countries and regions shown in the illustration.

The Asian countries benefit from low labor costs and most of the production is direct-shipping ore, with minimal hand sorting and crushing as the only beneficiation costs. However, some deposits in Morocco and some deposits in northern Thailand have high truck and rail costs to transport concentrates from the interior to the port for export.

Transportation is a major cost in the distribution of barite. Most of the barite must be transported comparatively long distances either by rail, as in the United States, or by ocean shipping for most of the non-U.S. mines.

Operations in the United States have the highest weighted-average total cost of production of all properties evaluated because of high labor costs, lower ore grades, and high shipping costs to market areas.

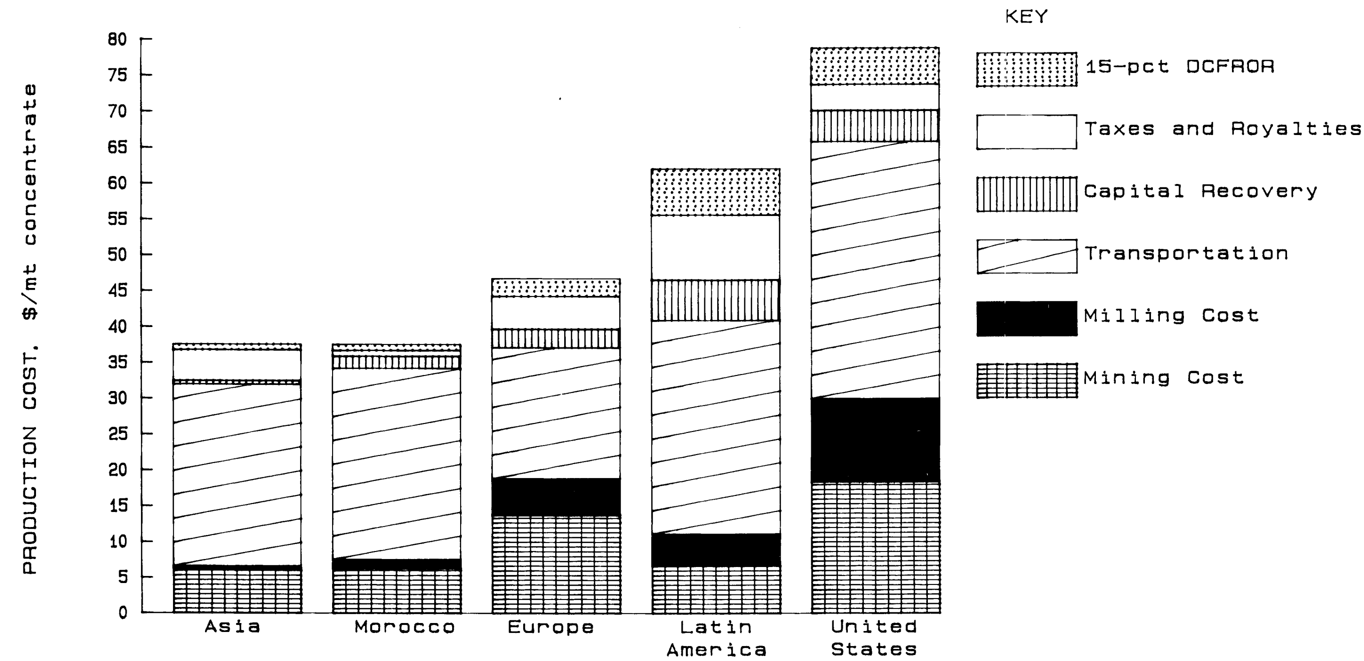

Figure 3.—Barite production costs for selected MEC's (January 1985 U.S. dollars).

Table 3.—Mud-grade barite production costs for producing operations in selected MEC's
(All costs are in January 1985 U.S. dollars per metric ton concentrate on a weighted-average basis)

Country	Operating cost		Trans-portation cost	Net operating cost	Recovery of capital[1]	0-pct DCFROR		15-pct DCFROR		
	Mine	Mill				Taxes and royalties[2]	Total cost[3]	Taxes and royalties[4]	Return on investment[5]	Total cost[6]
Asia	6.24	0.34	25.36	31.94	0.50	3.51	35.95	4.31	0.77	37.52
Europe	13.72	5.09	18.28	37.09	2.57	2.45	42.11	4.60	2.46	46.72
Latin America	6.68	4.39	29.82	40.89	5.64	2.44	48.97	9.06	6.44	62.03
Morocco	6.27	1.23	26.68	34.18	1.67	.15	36.00	.18	1.49	37.53
United States	18.39	11.54	35.82	65.75	4.30	2.40	72.15	3.60	5.10	78.75

[1]Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure as of January 1985, and investments required over the life of the operation.
[2]Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 0-pct DCFROR.
[3]Equal to the sum of net operating costs, taxation and capital recovery determined at a 0-pct DCFROR.
[4]Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 15-pct DCFROR.
[5]Revenue increase necessary per metric ton to obtain a 15-pct DCFROR.
[6]Equal to the sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery *plus* the per metric ton increase necessary to provide a 15-pct DCFROR after taxation.

Nearly 70 pct of the barite consumed in the United States is used in the Gulf Coast States and Oklahoma. Rail shipping costs from major shipping points in Nevada to the Gulf Coast States are estimated at $36/mt to $41/mt for unground barite.

Ocean shipping from other countries to the gulf coast generally is less expensive than U.S. rail shipping. As estimated in this study, shipment to the U.S. gulf ports from points in South America and Morocco would be no more than $12/mt. In 1982, shipping costs to the gulf coast were estimated at $20/mt to $25/mt and $20/mt to $24/mt from Thailand and China, respectively. In early 1985, freight rates were down and costs of shipments of 20,000 to 25,000 mt of barite were reported as low as $7/mt from Morocco to the gulf coast.

Open-pit mining costs range from about $1.50/mt to $10.50/mt ore. On a per ton of material (ore plus waste) basis, the costs range from about $1.10/mt to $7.50/mt. Related to concentrates produced, the costs range from $6.24/mt to $18.34/mt.

The per ton cost difference between ore plus waste and ore is greatest for the hard-rock deposits in the United States, as a result of the higher stripping ratios. The mining cost per ton ore for the detrital ores is the lowest of all operations, but because of the low grade and low recoveries, these mines have the highest cost per ton of concentrate.

Mining costs for underground mines are generally higher than the open pit costs and range from $8/mt to $20/mt ore and from $10/mt to $45/mt concentrate. On a weighted-average basis, the underground mining costs are $12.34/mt ore ($18.60/mt concentrate).

Beneficiation costs per ton of ore feed are generally in the range of $0.15 to $1.50 for hand sorting or crushing, $2.50 to $5 for jigging, and $4.50 to $9 for the combined processes (jigs, tables, and spirals); beneficiation costs per ton of concentrate range from a low of $0.62 for direct shipping ore and hand-sorting methods, to $13 for flotation concentrates.

AVAILABILITY

Of the 64 barite operations analyzed, 42 were producing, 12 were recent (1982) shutdowns (11 in the United States), 2 were in development stages, and 8 were explored. Some shutdown operations could reopen with improved market conditions. In terms of types of products, 48 operations recovered only drilling-mud-grade barite: 10 recovered mud-, chemical-, or filler-grades and/or lead, zinc, silver, or fluorspar byproducts; and 6 recovered only chemical or filler grades. Chemical- or filler-grade barite production amounted to about 8 pct of the evaluated barite production.

The MEC's included in the evaluation produced about 87 pct of the barite from all MEC's between the years 1979 and 1984 and about 80 pct of the 1985 MEC production. In the United States, about 90 pct of the production is from Nevada.

TOTAL RECOVERABLE

Mud-Grade Barite

The total availability of mud-grade barite recoverable as concentrates in terms of producers and nonproducers is shown on figure 4. The producing mines are shown at 0- and 15-pct DCFROR, and the nonproducing deposits are shown at 15-pct DCFROR only. The DCFROR includes the average total costs over the life of the operations. The 0-pct DCFROR covers all costs including recovery of capital, whereas the 15-pct DCFROR includes a 15-pct rate of return on invested capital in addition. As illustrated, the costs including the 15-pct DCFROR increase more towards the high end of the curve where undeveloped operations are more prevalent. Established operations have already recovered initial capital expenditures and have less costs to cover prior to making the 15-pct DCFROR. The total costs represent longrun constant 1985 dollar values that include the exploitation of total demonstrated resource. The January 1985 market price at gulf coast ports ranged from about $40/mt to $55/mt; the lower price material was a small part of the total. It appears that a more applicable price delivered to the gulf coast grinding plants would be about $60/mt. A little over 20 pct of the U.S. resources are available at an average total production cost of just over $60/mt, whereas 50 pct of the non-U.S. resources are available at or below this average total cost.

Chemical- and Filler-Grade Barite

The total availability from 15 operations proposed to recover unground chemical- and filler-grade barite is shown in figure 5. All but one of the chemical- and filler-grade operations are producers. This illustration includes curves representing the 0-pct (breakeven) and 15-pct DCFROR. Market prices for products were averaging between $80/mt and $165/mt in early 1985, and all of the evaluated tonnage recovered was potentially available below $150/mt, including a 15-pct DCFROR. About 40 pct (2.1 million mt) could potentially be produced for less than $95/mt, an average U.S. market price in 1985.

ANNUAL CAPACITY

The 1985 potential production of mud-grade barite from producing mines (at full capacity) at a 0-pct DCFROR is shown in figure 6. Potential 1985 mud-grade production at

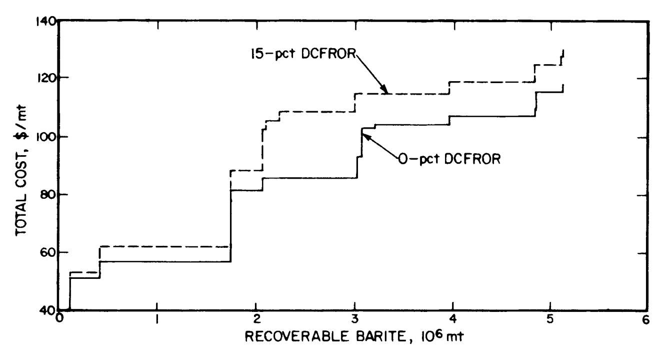

Figure 5.—Potential total chemical- and filter-grade barite available from MEC's (January 1985 U.S. dollars).

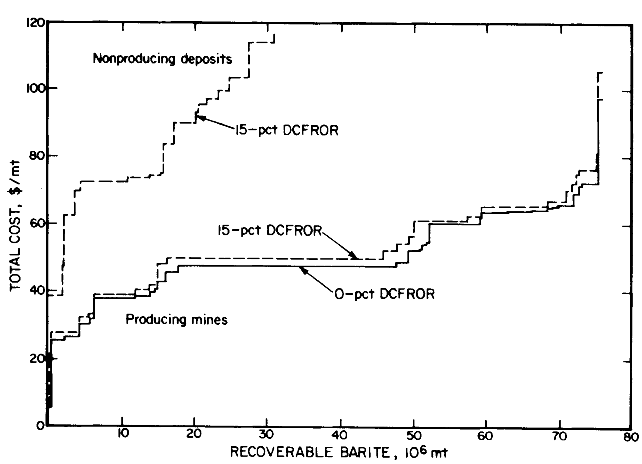

Figure 4.—Potential total mud-grade barite available from MEC's (January 1985 U.S. dollars).

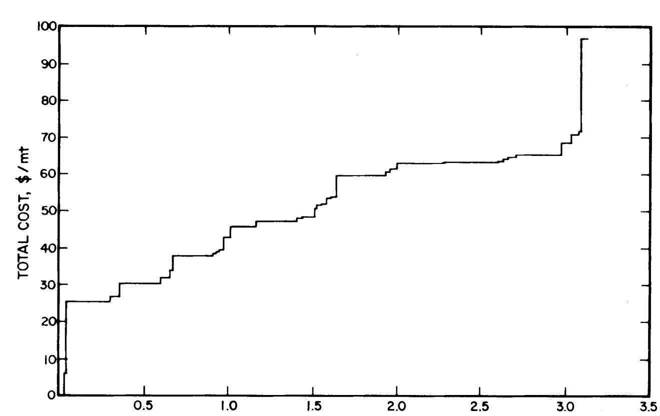

Figure 6.—1985 mud-grade barite capacity from producing mines in MEC's (January 1985 U.S. dollars; 0-pct DCFROR included).

costs of up to $105/mt (beyond the scale of the figure) amounts to about 3.4 million mt or about 89 pct of actual 1984 MEC production. At just over $60/mt, potential annual production is about 1.85 million mt. Potential annual production from producing mines for chemical- and filler-grade barite is about 290,000 mt/yr.

CONCLUSIONS

Because over 90 pct of the barite produced is used as drilling mud, barite production varies according to the amount of oil and gas well drilling activity.

As of January 1985, demonstrated resources for evaluated properties amount to about 128 million mt of contained $BaSO_4$; this could produce approximately 111 million mt of concentrates. The total identified resource is about 212 million mt (contained $BaSO_4$). As previously stated, resource estimates are considered to be conservative. Because of an extremely fluctuating market, most mining companies do not explore beyond a few years of production.

The United States will continue to rely heavily on imports in the future. Supply interruptions from individual countries would probably only affect the market in the short term, since there is such a broad production base (about 35 countries). If capacity were lost in one area of the world, it would be fairly easily compensated for by production increases in another area.

China is the largest producer of barite and has gradually taken over portions of the market previously held by other countries. It was not included in the analysis because of the lack of available data.

Including a 15-pct DCFROR, approximately 59 pct of mud-grade barite is potentially available below $60/mt. All of the chemical-grade concentrates evaluated could be recovered at costs below $150/mt.

The 1985 barite capacity levels estimated for producing mines evaluated in the study totaled about 3.7 million mt, which is approximately equal to the 1985 production reported for the MEC's. The capacity of the U.S. producing mines accounts for about 36 pct of the total. The total annual capacity of the operations shut down since 1982 amounts to nearly 900,000 mt; about 95 pct of this shut down capacity is in the United States, which amounts to about 40 pct of the U.S. capacity of producing mines. The average production of the U.S. producing mines in 1985 was about 60 pct of their evaluated 1982 capacity.

The United States will continue to depend upon other nations, primarily China, India, Morocco, Mexico, and Peru for a percentage of its barite needs. Between the years 1979 and 1983, U.S. imports represented 48 pct of domestic consumption. Competition from lower cost imports makes costs of long transportation to the major domestic consuming areas difficult to overcome.

REFERENCES

1. Coffman, J. S., and C. C. Kilgore. Barite Availability—Market Economy Countries. A Minerals Availability Program Appraisal. BuMines IC 9115, 1986, 25 pp.

2. Ampian, S. G. Barite. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 16-17.

3. ____. Barite. Ch. in Minerals Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 65-74.

4. ____. Barite. Ch. in BuMines Minerals Yearbook 1984, v. 1, pp. 129-139.

5. Brobst, D. A. Barite, Ch. in United States Mineral Resources: U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 75-84.

BERYLLIUM

INTRODUCTION

Beryllium is an important constituent of components for defense, nuclear, and aerospace industries. Three functional forms of beryllium—oxide, metallic, and alloy—represented 15, 20, and 65 pct, respectively, of beryllium consumption in 1983. Beryllium-copper alloys are fabricated into springs, connectors, and switches for electrical industries. Beryllium oxide is converted to ceramics for computer hardware and other electronic industry applications: connector components, transistors, logic chips, silicon-controlled rectifiers, electronic automotive ignitions, voltage regulators for electronic motors, and solid-state controls for appliances. In other applications, ceramics are used for precision tools, radar and navigation instruments, chip carriers, and microwave and diode packages. Aerospace industries use metallic beryllium for space shuttle brake disks, windshield frames, and umbilical doors. Metallic beryllium applications in other aerospace industries include guidance systems, meteorological satellite instruments, telescopes, and scanning mirrors. The aerospace industry also uses beryllium in alloy and ceramic forms. In the nuclear industry, metallic beryllium reflects neutrons in nuclear reactors and atomic reactor reflector cores.

The physical and mechanical properties to which beryllium owes its importance are light weight, high stiffness, strength, and heat absorption. Suitability as a neutron moderator, X-ray transmitter, and electrical conductor also contributes to beryllium's importance.

The beryllium industry is relatively small, which makes an availability analysis somewhat limited in scope when compared with other major commodities. In the Western Hemisphere, only one mine in the United Staes produces bertrandite; five districts in Brazil produce beryl ore. Of market economy country (MEC) beryl ore producers, Brazil ranks first in the production of beryl ore. Other MEC's accounted for only 2 pct of total world production. In the Eastern Hemisphere, China and the U.S.S.R. produce the greatest portion of world production of beryl ore.

Throughout the world, there are few processing plants that extract beryllium from ore. The United States has the only facility in the world that extracts beryllium from bertrandite and beryl ore. The extraction facility, owned by Brush Wellman Inc., produces beryllium hydroxide that is further refined into beryllium metal, alloys, and ceramics. In the Eastern Hemisphere, Japan, France, and the U.S.S.R. currently have beryllium processing plants, and China has plans to construct a beryllium oxide plant in Xinjiang Uygar, an autonomous region (*1*).[1]

Because bertrandite mining and milling costs and beryllium extraction and processing costs are proprietary, availability analyses of bertrandite and beryllium are not presented in this study. Lack of data on individual deposits and mining regions of China and the U.S.S.R. precludes an analysis of beryl ore availability from these countries as well; therefore, availability analysis is limited to beryl ore production in Brazil.

Most of the information contained in this summary report was obtained from Bureau of Mines Information Circular 9100 "Beryllium Availability—Market Economy Countries. A Minerals Availability Appraisal," and other Bureau of Mines publications (*2-4*).

GEOLOGY AND RESOURCES

GEOLOGY

Within the Earth's crust, beryllium has been identified in at least 56 minerals, about half of which contain in excess of 1 pct Be. Beryl, bertrandite, phenacite, helvite, and barylite are the major minerals that are prospected for by beryllium producers. Bertrandite and beryl are currently the only beryllium minerals mined among MEC's for the production of beryllium concentrates.

Beryl, $Be_3Al_2Si_6O_{18}$, contains the equivalent of 10 to 14 pct beryllium oxide (BeO). Beryl, hexagonal crystal in form, is usually light, opaque green, but can also be white or yellow. Gem-quality beryl is transparent, being either dark green, light green, light blue, pink, or yellow in color (*5*).

Beryl commonly occurs within dikes of zoned and unzoned pegmatites. Most pegmatites have simple mineralogy and have no well-defined zones; however, economic deposits of beryl are usually associated with zoned pegmatites. Feldspar, mica, and spodumene are mined from zoned pegmatites as primary products, with beryl as a byproduct. Zoned pegmatites and most of the larger, unzoned pegmatites are igneous in origin.

Zoned pegmatites are distinguished by concentric zones. Four basic zones have been identified: border zone, wall zone, intermediate zone, and core; however, not all zoned pegmatites have all four zones. Within the individual pegmatites, zones are irregular and may be discontinuous. The texture of grains in the zones becomes progressively coarser toward the inner zones. Giant beryl crystals

[1]Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

sometimes form within the inner zones. Beryl also occurs as a fine-grained accessory mineral within the border zone and as fragments in the inner zones.

Major nonpegmatitic districts have three characteristics in common: high angle faults, fluorite metallogenic provinces, and calcareous rocks. Based on U.S. Geological Survey estimates, beryllium resources derived from nonpegmatitic districts are much greater than from pegmatitic districts. Of the known nonpegmatitic districts in the United States, the three largest—Gold Hill and Spor Mountain, UT, and Seward Peninsula, AK—contain the majority of estimated resources from these districts (*6*).

One nonpegmatitic beryllium-bearing resource currently yielding bertrandite ore is an epithermal deposit at Spor Mountain in Utah. Bertrandite, $Be_4Si_2O_7(OH)_2$, is an orthorhombic crystal containing from 39.6 to 42.6 pct BeO. Mineralization, which includes bertrandite, fluorspar, manganese oxides, and uranophane, occurs in a water-laid volcanic tuff bed. Overlying the stratiform ore deposit are rhyolites; calcareous rocks underlie the tuff bed containing the mineralization.

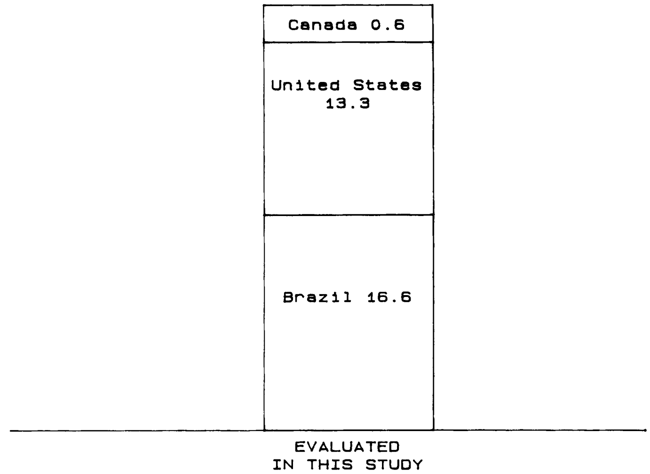

Figure 1.—Estimates of world beryllium resources (thousand metric tons).

RESOURCES

The demonstrated in situ resources of the deposits and districts evaluated in this study contain 30,500 mt of beryllium metal as shown in table 1 and figure 1. Approximately 1,577 mt (5 pct) of the 30,500 mt beryllium is a byproduct associated with other primary mineral deposits. Although beryllium resources exist in Africa, Asia, Europe, Australia, and centrally planned economy countries (CPEC's), demonstrated resource estimates were not available. The majority of MEC resources occurs in Brazil and the United States. Table 2 lists the properties included in this study.

Table 1.—Summary of selected MEC's demonstrated beryllium resources evaluated for this study, as of January 1985

Region and country	In situ resource: Ore, 10^3 mt	In situ resource: Grade,[1] wt pct BeO[1]	Contained beryllium metal,[2] mt
North America:			
Canada	835	0.20	600
United States	10,375	.36	13,300
South America: Brazil	105,594	.04	16,600
Grand total or weighted av	116,804	.08	30,500

[1]Weighted average by country.
[2]Obtained by dividing the product of (in situ ore times BeO grade) by 2.78. BeO grade must be converted from weight percent to decimal (divide by 100) before multiplying.

Table 2.—MEC beryllium properties included in this study

Country and deposit name	Owner	Current status[1]	Primary mineral	Be minerals	Mining method[2]	Milling method[3]	Deposit type[4]	Year of initial production	Feed grade, BeO wt pct	Mill ore capacity, mt/yr	Product grade, BeO wt pct
Brazil:											
Cascavel-Cristais	Private land owners	P	Beryl	Beryl	OP	HS	ZP	1942	0.008	40	10
Minas Gerais	..do	P	..do	..do	OP	HS	ZP	1942	.044	433	10
Picui-Parelhas	..do	P	..do	..do	OP	HS	ZP	1939	.008	150	10
Quixeramobim-Solonopole.	Private land owners and some government land.	P	..do	..do	OP	HS	ZP	1945	.008	40	10
Southern Bahia	Private land owners	P	..do	..do	OP	HS	ZP	1942	.044	238	10
Canada: Bernic Lake	Tantalum Mining Corp., Manitoba Government (25 pct), Kawecki Berylco (37 pct), and Hudson Bay Mining.	T	Tantalum	..do	RP	F	ZP	1969	.18	180,000	8.9
United States:											
Custer	Numerous owners	PP	Mica	..do	OP	F	ZP	1914	.06	144,000	8.9
Hill City	..do	PP	..do	..do	OP	F	ZP	1914	.044	144,000	9.2
Keystone	..do	PP	..do	..do	OP	F	ZP	1914	.039	144,000	8.9
Mount Wheeler	Mt. Wheeler Mines, Inc., subsidiary of National Treasure Mines Co.	PP	Scheelite	Betrandite, phenacite, beryl.	OS	F	ZP	1911	1.0	35,000	20
Muscovite Mine	Unknown	PP	Mica	Beryl	OP	HS	ZP	1988	NAp	NAp	NAp
Railway Dike	Meridian Land and Mining Co.; Burlington Northern.	E	Beryl	..do	OP	F	ZP	Unk	.3	36,400	7.0
Spor Mountain	Brush Wellman, Inc.	P	Bertrandite	Bertrandite	OP	S	NP	1969	.614	99,000	NAp

Unk Unknown.
NAp Not applicable.
[1]P, Producing; E, explored; T, temporarily shut down; PP, past producer.
[2]OP, open pit; RP, room and pillar; OS, open stope.
[3]HS, hand sort; F, flotation; S, size.
[4]ZP, zoned pegmatite; NP, nonpegmatite.

U.S. AND WORLD HISTORICAL PRODUCTION

Estimated world production for 1985 was approximately 352 mt of contained beryllium from 8,800 mt of bertrandite and beryl ore. Rwanda and Zimbabwe contributed approximately 1 pct of the world production; the United States, 59 pct; Brazil, 17 pct; other MEC's contributed 1 pct of the world total; and CPEC's approximately 22 pct (*4*).

Prior to 1969, beryl ore was the major resource available for the production of beryllium; subsequently, one large bertrandite mining operation began production and displaced a large portion of the beryl ore required to produce the same amount of beryllium. Figure 2 depicts historical trends of beryl ore production for Brazil and other world producers in 5-yr intervals from 1950 through 1985. U.S. bertrandite production was not reported prior to 1980 and is therefore not represented in figure 2. The United States produces a small amount of beryl ore as a byproduct of mica, feldspar, or spodumene operations; most of the beryl ore used to supplement U.S. output of bertrandite ore in the production of beryllium is imported.

U.S. production of beryllium depends upon Brush Wellman, which is involved in the beryllium industry from the mining of bertrandite at Spor Mountain to developing and fabricating endproducts. Brush Wellman recently experimented with the possibility of processing beryl ore having a BeO grade lower than the average 11 pct currently being processed.

The Cabot Corp., a company involved in the U.S. beryllium industry since the 1930's, has sold its beryllium alloy manufacturing plant in Pennsylvania to NGK Insulators, Inc., of Japan. NGK is expected to continue production of beryllium-copper alloy in the United States.

EXTRACTION AND PROCESSING TECHNOLOGY

Basic mining techniques of the two largest MEC ore producers, Brazil and the United States, consist of open pit methods; however, the U.S. open pit is mechanized whereas Brazil's mines are predominantly nonmechanized. Beneficiation techniques differ because of the dissimilarity of the ores and equipment used in each of the two countries.

The United States is the largest producer and consumer of beryllium among MEC's. The Spor Mountain bertrandite mine in Utah, owned and operated by Brush Wellman, Inc., is the only U.S. primary source of ore from which beryllium hydroxide (an intermediate beryllium compound needed to make beryllium products) is produced. Many U.S. pegmatite districts have supplied beryl ore in the past, but currently produce very small amounts of beryl (almost insignificant in terms of U.S. consumption). Other MEC's and CPEC's produce beryl ore concentrates from beryl-rich pegmatites. Brazil is the largest supplier of beryl ore to the United States.

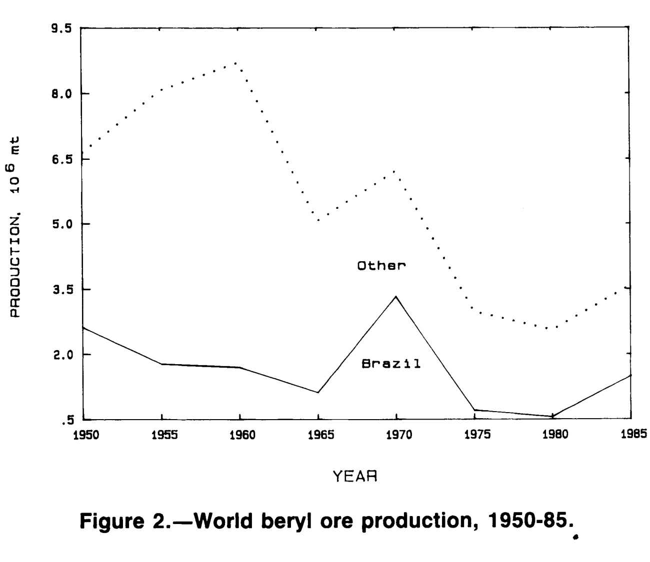

Figure 2.—World beryl ore production, 1950-85.

MINING

Table 2 lists salient information regarding deposits and districts included in this study. All the operations evaluated in this study with the exception of Spor Mountain are zoned pegmatite deposits. The majority of beryl ore mining in Brazil is labor-intensive with minimal capital costs. Mining of beryl from pegmatites involves manual open pit mining techniques requiring simple handtools, picks, shovels, and wheelbarrows; in some areas, explosives and drills are used. Many garimpeiros (laborers) work independently, usually in groups of three to five men. The garimpeiros mine weathered pegmatites by stoping into the side of the pegmatite with picks and shovels. Beryl crystals are picked out by hand and placed in burlap bags. The mucked-out material is heaped in dumps from which women and children recover beryl crystals that were undetected by the men. Recovered beryl ranges in grade from 6 to 8 pct BeO. Each garimpeiro trades or sells the beryl to local merchants.

The only mine in the United States producing bertrandite is Spor Mountain, an open pit operation. After clearing the overburden with scrapers and dozers, front-end loaders and off-road haulage trucks remove the blasted rhylite, exposing the underlying tuff. The tuff requires ripping with a dozer. The ore is loaded with self-loading scrapers and then stockpiled. Because the ore varies in grade, stockpiled ore is layered and then sampled to achieve an economical blend of ore grades for the mill circuit.

PROCESSING

In Brazil, beryl is stored in sheds where further sorting and upgrading by hand produces beryl ore with an equivalent BeO grade of at least 10 pct. Beryl crystals and fragments, approximately 2 cm or greater in size, are upgraded by chipping away the gangue with handtools. The beryl ore is packed in burlap bags and trucked to large port cities in Brazil.

Fine-grained beryl mined as a byproduct of mica can be recovered by flotation. The ore is first ground to minus 35 mesh, and then magnetic or electrostatic separators remove other heavy minerals from the ore. In the first series of flotation units, mica is recovered and the tailings from the circuit are conditioned and floated to produce a beryl-feldspar bulk concentrate. The bulk concentrate is conditioned with hypochlorite, washed, conditioned with sulfuric acid and petroleum sulfonate, and diluted to 25- to 50-pct solids to depress the feldspar. In the second series of flotation units, beryl is recovered, then cleaned and filtered.

Brush Wellman, Inc., operates the only beryllium extraction plant in the Western Hemisphere. Both bertrandite and beryl ore are processed in Brush Wellman's Delta plant located near Lynndyl, UT. In the initial phase of the extraction process, bertrandite and beryl enter separate circuits.

Bertrandite ore is crushed, wet ground, and wet screened to a minus 20-mesh slurry. The slurry is leached by using sulfuric acid heated to a temperature near boiling point. The leach process solubilizes beryllium contained in the bertrandite, and the resulting leachate solution enters countercurrent decantation thickeners to separate solids from the beryllium sulfate.

Beryl is crushed, melted at 1,625 °C, then quenched in water to produce a beryl glass that is reheated and ground to minus 200-mesh powder. A slurry of beryl powder and sulfuric acid heated to 250° to 300 °C converts to beryllium sulfate. Beryllium sulfate is then separated from the solids by a countercurrent decantation process, after which the beryllium sulfate is combined with that extracted from the bertrandite. After the sulfate solutions are combined, beryllium is recovered as beryllium hydroxide through a liquid ion exchange process.

PRODUCTION COSTS

Weighted-average production costs in U.S. dollars as of January 1985 for five producing Brazilian beryl districts are shown in table 3. Mining costs ranged from $189/mt to $344/mt of beryl ore. Milling costs ranged from $134/mt to $453/mt, and transportation costs ranged from $16/mt to $63/mt.

A number of factors are responsible for the wide range in costs. Labor costs constitute the major portion of mining and milling costs. Garimpeiros are self-employed and each must develop individual terms of trade with local ore buyers, thus the cost of labor is highly flexible rather than fixed. Semimechanized mining has somewhat higher costs than nonmechanized mining because of the additional costs for fuel and explosives. Transportation costs vary according to the distance the ore must be trucked from the local ore buyers to Brazilian port cities.

Since 1981, a high rate of inflation has prevailed in Brazil, while the cruzeiro has been devalued in terms of U.S. dollars. Although costs in Brazil have been escalating since 1981, beryl ore production costs appear to be decreasing when compared in U.S. dollars rather than in cruzeiros.

No production costs are given for the United States because beryl ore is mined as a byproduct of other minerals. Production costs for the producing bertrandite mine and one nonproducing beryl deposit are withheld to preserve confidentiality.

Table 3.—Beryl ore production costs for Brazil open pit producing mines

(All costs are in January 1985 U.S. dollars per metric ton of beryl ore on a weighted-average basis)

Mining method	Operating cost		Transportation to plant and port[1]	Net operating cost	Recovery of capital[2]	0-pct DCFROR		15-pct DCFROR		
	Mine	Mill				Taxes and royalties[3]	Total cost[4]	Taxes and royalties[3]	Return on investment	Total cost[5]
Open pit: Producers	233.00	328.00	31.00	592.00	14.00	103.00	709.00	108.00	7.00	721.00

[1]Transportation costs to upgrading and storage area and to port.
[2]Includes cost of recovering remaining undepreciated investments and reinvestments required over the life of the operation.
[3]Includes Federal taxes and royalties.
[4]Includes recovery of all costs of production including capital but does not include profit.
[5]Includes recovery of all costs of production including capital and return on investment at a 15-pct DCFROR.

AVAILABILITY

Because costs associated with beryllium extraction are not known, an availability analysis of beryllium is not included. Bertrandite ore is not sold or traded in the market and, therefore, there is no market price to compare to the cost of production. Moreover, mining and milling costs of the one bertrandite producer are confidential and an availability analysis of bertrandite cannot be presented. An analysis of total availability of beryl ore from Brazil (the world's largest MEC producer of beryl ore) is presented.

The total amount of Brazilian beryl ore potentially available is nearly 400,000 mt, containing an equivalent 10 pct BeO (approximately 14,400 mt Be contained in recovered beryl ore). Figure 3 shows the total availability of beryl ore from five Brazilian districts.

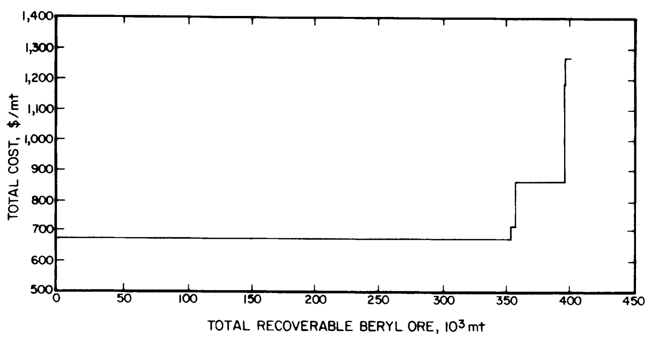

Figure 3.—Potential total beryl ore available from Brazil (January 1985 U.S. dollars; 0-pct DCFROR included).

Annual production from the five Brazilian districts for 1985 was estimated at 779 mt of beryl ore containing approximately 28 mt Be. Estimated annual production of contained beryllium represented only 47 pct of Brazil's actual production of 60 mt contained beryllium. Annual availability cannot be adequately predicted for several reasons. The amount of beryl ore produced is directly related to the number of garimpeiros mining beryl. The number of garimpeiros mining in each of the five districts fluctuates according to the season, unemployment rate, and economic factors. Inflation and currency devaluation in Brazil have had the greatest influence on beryl production in the past 5 yr. Market price of beryl ore had a subordinate role in influencing annual production.

Beryl ore prices calculated at 0-pct discounted-cash-flow rate of return (DCFROR) for each Brazilian district ranged from $678/mt to $1,270/mt. The price range of beryl ore calculated at 15-pct DCFROR does not differ greatly from that calculated at 0 pct because of the insignificant amount of capital expenditure associated with Brazil's beryl ore mining. The January 1985 market price for beryl ore was $1,058/mt; the market price of beryl ore includes the shipping costs to U.S. ports and insurance. Estimated shipping cost to U.S. west coast ports ranged from $104/mt to $130/mt of beryl ore. Comparison of calculated prices of beryl ore with that of the January 1985 market price indicated that two of the five districts require a higher price than that of the January 1985 beryl ore market price to meet their costs of production.

CONCLUSIONS

Of the total demonstrated resources evaluated in this study, 30,500 mt of contained beryllium, 98 pct occurs in the United States and Brazil. Demonstrated resources alone are sufficient to provide beryllium through the mid-21st century.

Brazil produced 17 pct of the total world production of beryllium contained in ore. In 1985, estimated production from five Brazilian districts was 779 mt of recovered beryl ore containing 28 mt of beryllium. Estimated ore production from the five districts represented approximately 42 pct of Brazil's total ore production for 1985. In the past 5 yr, of all factors affecting production of beryl ore, economic conditions have had the greatest impact.

The United States produced the largest percentage (59 pct) of the total world production of beryllium contained in ore. In the Western Hemisphere, Brush Wellman, Inc. is the sole producer of beryllium extracted from domestic ore (bertrandite) and beryl ore. Throughout the world, few beryllium producers extract beryllium from ore. Cost data pertinent to the extraction of beryllium could not be obtained, thus precluding a complete analysis of beryllium availability.

REFERENCES

1. International Mining. News China. Oct. 1985, p. 64.
2. Soja, A.A., and A.E. Sabin. Beryllium Availability—Market Economy Countries. A Minerals Availability Appraisal. BuMines IC 9100, 1986, 19 pp.
3. Kramer, D. Beryllium. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 20-21.
4. Petkof, B. Beryllium. Ch. in Mineral Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 75-82.
5. Mason, B., and L.G. Berry. Elements of Mineralogy. Freeman, 1968, pp. 479-480.
6. Griffiths, W.R. Beryllium. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 85-93.

CHROMIUM

INTRODUCTION

Chromite products (ore and concentrates of ore that contain chromium) and their associated products (ferrochromium, chromium chemicals, chromium metal, refractory bricks, etc.) are among the most important strategic and critical materials to the U.S. economy. Except for a small amount of production in 1976, no chromite has been produced in the United States since 1960. Over the last 5 yr, net chromium import reliance (i.e., chromium contained in all product forms) as a percent of apparent consumption has ranged from a high of 90 pct in 1981 to 73 pct in 1985. The differences between imports and consumption, 10 pct in 1981 and 27 pct in 1985, represent the recycling of purchased stainless steel scrap. Chromium is classified as a strategic material. Only two countries, the Republic of South Africa and Zimbabwe, have accounted for 67 pct of annual U.S. imports since 1981 (*1*).[1]

The critical nature of chromium products to the U.S. economy is primarily due to its essential use in producing stainless and heat-resistant steels and in producing superalloys for use in the aerospace industry. Its use in the nonferrous alloys and steel alloys industries is also of importance to the U.S. economy. All of the uses cited represent metallurgical consumption of chromium that utilizes intermediate chromium products in the form of ferrochromium (both high- and low-carbon), ferrochromium silicon, and chromium metal. The criticality of chromium is primarily that it imparts oxidation and corrosion resistance to stainless steels and superalloys.

Subordinate to the metallurgical uses for chromium are chemical and refractory uses. Primary chemical applications are in the production of pigments, in metal plating and other metal treatment, in leather tanning, and as a catalyst. Refractory applications involve the use of chromite ores and concentrates in making basic refractory bricks of chromium-magnesia or magnesia-chromium refractory composition for use primarily in open-hearth furnaces. Also included in this refractory use category are foundry sands primarily produced in the Republic of South Africa and Finland, which are used for molds. Of the chromite consumed in the United States, the metallurgical and chemical industries account for 86 pct and the refractory industry accounts for 14 pct (*2*). These consumption percentages apply, in general, to other major industrial nations.

The mineral chromite represents the base product from which all chromium ferroalloy, metal, chemical, and chromite refractory products are derived. For this reason, in this study, the availability of chromite (in ore and concentrate form) determines the availability of other chromium products. Chromite is the beneficiated, postmill product of chromium ore, that is traded for its chromium or chromite mineral content.

Chromite is produced to many different specifications. Not all mines can produce all grades, but mines do produce more than one grade. In this study, chromite is treated as a single material for the purpose of a general overview of chromium availability. Chromite quantities are expressed as gross weight, metric ton, with an associated chromic-oxide content (grade) expressed as percent Cr_2O_3.

Most of the information in this chapter is an updated summary of chromite product data and analysis included in Bureau of Mines Information Circular 8977 "Chromium Availability—Market Economy Countries. A Minerals Availability Program Appraisal" (*2*). Additional information on international chromium deposits is available from the U.S. Geological Survey (*3*).

GEOLOGY AND RESOURCES

GEOLOGY

Although the element chromium is found in a variety of minerals, all present commercial resources and reserves of chromium are contained in the mineral chromite. Chromite is a member of the spinel crystalline structure class and, in its pure form, has the chemical formula $FeO \cdot Cr_2O_3$. However, in natural deposits magnesium substitutes for iron while aluminum and iron substitute for chromium in the crystalline structure, with the result that the natural chromite is more closely described by the formula $(Fe,Mg) \cdot (Cr,Al,Fe)_2O_3$. By weight, the maximum Cr_2O_3 content in the pure mineral chromite would be about 68 pct; however, the substitution effect results in a range of from 15 to 64 pct (*4*, pp. 1, 5). By contrast, the weight of chromium in Cr_2O_3 would be a constant 68 pct.

At present, marketable chromite producers have Cr_2O_3 weight percentages ranging from the low thirties to the low fifties with the vast majority of producers in the 40- to 50-pct Cr_2O_3 range. These products are mostly produced from in situ ores grading from 20 to 50 pct Cr_2O_3. Natural characteristics such as the chromium-to-iron ratio, the chromite grain size, and the SiO_2, Al_2O_3, MgO, and P contents can be as important as the Cr_2O_3 content in determining the suitability of a chromite product for varying end uses.

[1]Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

Chromite mineralization occurs as massive, disseminated, or schlieren (banded) chromite grain accumulations in three broad types of structural or genetic classifications. The three classes—podiform, stratiform, and eluvial-alluvial-lateritic deposits—are described in the following paragraphs.

Podiform deposits (literally, in the form of pods) essentially have three large dimensions (relative to one another) and exhibit a variety of shapes such as pods, lenses, and pipes, even including U shapes. In any particular area of podiform chromite deposits, the vast majority of the deposits are of small size (less than 100,000 mt). In all of the nations where the majority of past production has come from podiform deposits (e.g., the Philippines, Turkey, Greece, Madagascar, Zimbabwe, the Soviet Union, and Albania) only a small percentage of the total number of deposits have provided the bulk of past production. In situ grades of the podiform deposits analyzed in this study range from 18 to 50 pct Cr_2O_3 and all have chromium-iron ratios greater than 2.0.

Stratiform deposits are also referred to as seam deposits and are characterized by very large extents in two dimensions (length and width) and small thicknesses (less than 3 m). In every major example of this type of chromite deposit (producers in the Republic of South Africa, Finland, Brazil, and the Great Dyke of Zimbabwe; nonproducers in the Stillwater Complex in Montana, United States, the Fiskenaesset Complex in Greenland, and the Bird River Sill in Manitoba, Canada), the entire sequence contains more than one, and sometimes many different, seams. In situ Cr_2O_3 grades range from the low teens to low twenties for the nonproducing deposits and from the high twenties to the high forties for producing stratiform deposits. Chromium-iron ratios are low in the vast majority of stratiform deposits—1.0 to 1.5 for the Greenland and Canadian deposits as well as the UG 2 seam of the South African Bushveld Complex; 1.5 to 2.2 at the Brazilian, U.S., Finnish, and all other Bushveld Complex seams; and 2.0 and above at the Great Dyke seam deposits in Zimbabwe.

Eluvial-alluvial-lateritic chromite deposits represent a genetic classification resulting from the weathering of originally emplaced podiform or stratiform chromite deposits. They are low-grade (less than about 15 pct Cr_2O_3 in situ) and consist of disseminated chromite grain accumulations either left in place or transported from the original deposit. Examples of this chromite deposit classification include the eluvial deposits of Zimbabwe (which have seen production in the past), the Oregon beach sand deposits of the United States; numerous deposits in the Philippines; the Ramu River deposit in Papua New Guinea; and the Plum deposit located offshore of New Caledonia.

RESOURCES

According to reference 1, "World resources total about 33 billion mt of shipping grade[2] chromite, sufficient to meet conceivable demand for centuries. Over 99 pct of these resources are in southern Africa, nearly 23.6 billion mt in the Republic of South Africa, and 10 billion mt in Zimbabwe." These resources are shown graphically in figure 1. The resources of other countries are negligible by comparison. Resources in the United States are mostly in the Stillwater Complex of Montana, some beach sands of Oregon, and numerous podiform deposits in northern California and southern Oregon as well as some laterite deposits in California.

In situ country reserve base values for chromite are also shown in figure 1. As is the case with world resources, only two countries, the Republic of South Africa and Zimbabwe, account for 95 pct of the 6.8 billion mt reserve base; as a result, the market economy countries (MEC's) as a group represent over 97 pct. The world reserve base value is only 21 pct of in situ world resources but still represents several hundred years of production at current levels. Evaluated MEC chromite resources total only 1.2 billion mt in situ. Although they represent only a small subset of the MEC reserve base value, they still represent over 120 yr of MEC chromite production at current levels. Table 1 summarizes evaluated chromite resources by country and shows the (1) demonstrated in situ chromite resource and weighted-average grade, and (2) recoverable chromite products marketable at an associated postmill product grade.

It is important to note that a large majority (approximately 67 pct or 4.6 billion mt) of the chromite reserve base value is contained within the UG-2 reef of the Bushveld Complex in South Africa. The UG-2 reef has only been mined for its platinum-group metals content since 1983 and

[2]Shipping grade is defined as deposit quantity and grade normalized to 45 pct Cr_2O_3 for high-chromium and high-iron chromite and 35 pct Cr_2O_3 for high-alumina chromite. High-alumina chromite is defined as chromite containing more than 60 pct combined Cr_2O_3 and Al_2O_3. The in situ data alone are normalized to allow for quantitative in situ cross-country resource comparisons. These normalized resource data are not directly comparable with the evaluated in situ chromite tonnages, which represent actual resource quantities and grade data.

the tailings from the platinum-group metals processing contain between 30 to 32 pct Cr_2O_3 with a chromium-iron ratio of between 1.2:1 and 1.3:1. These tailings are a potential source of chromium because the technology to extract the chromite from the tailings and produce ferrochromium has been demonstrated, but to date no marketable chromium products of any form have been produced on a commercial scale. The reasons for this are many and complex and include a reluctance on the part of the mining companies to make the substantial investment required to develop large-scale facilities to utilize the tailings. A major reason for this reluctance is the uncertainty of a market for the chromite or ferrochromium (if it were processed to that stage), because the ferrochromium produced from this material would grade lower in contained chromium than any currently being marketed.

Table 2 lists the mines and deposits that compose the evaluated chromite resources of this study.

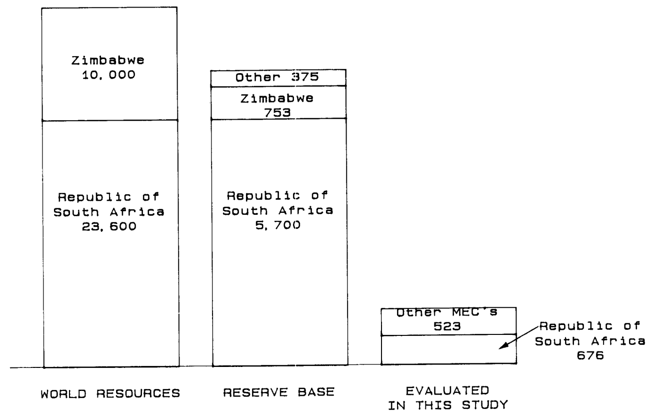

Figure 1.—Estimates of world chromium resources (million metric tons).

Table 1. - Summary of MEC demonstrated chromium resources evaluated for this study, as of January 1985

Country	Demonstrated resource, 10^3 mt	Weighted-average grade, pct Cr_2O_3	Contained chromium, 10^3 mt	Concentration ratio, crude ore to chromite product	Postmill product grade, pct Cr_2O_3	Recoverable chromite products, 10^3 mt	Percent of total estimate
South Africa, Republic of	675,786	41.0	277,072	1.2	43.0	466,631	68.6
Zimbabwe:							
Seam type	171,172	49.0	83,874	1.1	NAp	114,195	16.8
Podiform type	14,357	46.5	6,676	1.2	NAp	11,730	1.7
Eluvial type	3,250	20.0	650	5.6	NAp	422	.1
Total or average	188,779	49.0	91,200	1.1	50.0	126,347	18.6
Philippines:							
Low grade	177,224	2.0	3,544	30.0	NAp	5,765	.9
High grade[1]	18,559	28.0	5,196	2.2	NAp	7,640	1.1
Total or average	195,783	4.5	8,740	14.0	47.0	13,405	2.0
Turkey[2]	8,300	38.0	3,154	1.3	46.0	5,846	.8
India	79,835	32.0	25,547	1.5	46.0	42,185	6.2
Brazil[3]	13,450	21.0	2,824	3.5	44.0	3,538	.5
Finland	25,200	27.0	6,804	1.6	31.0	17,377	2.6
New Caledonia	600	44.0	264	1.5	51.0	409	.1
Greece	1,653	18.0	298	3.5	51.0	450	.1
Madagascar	8,935	32.0	2,859	2.5	49.0	3,435	.5
Grand total	1,198,321	NAp	418,762	NAp	NAp	679,623	100.0

NAp Not applicable.
[1]Does not include 21 nonrefractory deposits containing about 1.8 million mt of in situ resource; 18 are very small (average 40,000 mt reserve); 3 are fairly large (average 360,000 mt). Does not include a further 12 refractory deposits containing about 1.1 million mt of in situ resource; 10 are very small (average 50,000 mt); 2 are fairly large (average 300,000 mt). Small size or lack of data precludes estimation of costs.
[2]Does not include the many small deposits and operations, representing 20 to 30 pct of Turkey's overall total production of chromite products, which are too small or sporadic to evaluate for costs.
[3]Represents resource in the Campo Formoso District only.

Table 2.—MEC chromite properties[1] included in this study

Country	Ownership	Type of occurrence	Status[2]
Brazil:			
Pedrinhas (Campo Formoso).	Ferbasa	Stratiform	P/S
Limoeira (Campo Formoso).	..do	..do	P/S
Finland: Kemi	Outokumpu Oy	..do	P/S
Greece: Xerolivado	Gemee	Podiform	P/S
India:			
Byrapur	Mysore Minerals Ltd	..do	P/S
Jambur-Tagadur	..do	Stratiform	E
Cuttack District-low grade.	Various owners	Stratiform-pod	E
Cuttack District-high grade.	..do	..do	P/S
Keonjhar District-low grade.	..do	..do	E
Keonjhar District-high grade.	..do	..do	P/S
Madagascar:			
Andriamena	Kraomita Malagasy (government).	Podiform	P/S
Ranomena	..do	..do	P/S
New Caledonia: Tiebaghi (Chromical).	Inco Metals Ltd	..do	P/S
Philippines:			
Masdang	Acoje Mining Co	..do	P/S
Narra	Trident Mining & Industrial Corp.	..do	P/S
Acoje (Santa Cruz)	Acoje Mining Co	..do	P/S
Candalaria	Intercontinental Mineral Co.	..do	P/S
Lagonoy	New Frontier Mines Inc	..do	P/S
Llorente	Pacific Shore-Rio Chico Mining.	Eluvial-alluvial.	E
Bicobian	Island Mining & Indust Corp.	Lateritic-eluvial	E
Batang-Batang	Philchrome Mining Corp	Alluvial	P/S
Bacungan	Golden Arrow Mining Inc.	Podiform-eluvial	P/S
Irahuan	Palawn Consolidated Mining Co.	..do	P/S
Coto-Masinloc	Consolidated Mines Inc	Podiform	P/S
Kinmalgin	C-Square Consolidated Mines Inc.	..do	P/S
South Africa, Republic of:			
Zwartkop	Samancor (Cromore)	Stratiform	P/S
Rustenburg	Associated Ore and Metal Corp. Ltd.	..do	P/S
Ruighoek	Samancor (Cromore)	..do	P/S
Ntuane	Anglo Transvaal Cons Invest.	..do	P/S
Waterkloof	Samancor (Cromore)	..do	P/S
Millsell	Rand Mines	..do	P/S
Kroondal	Samancor (Cromore)	..do	P/S
Bayer A.G. (Chrome Chemicals).	Bayer A.G. (Chrome Chemicals).	..do	P/S
Henry Gould	Rand Mines	..do	P/S
Mooinooi	Samancor (Cromore)	..do	P/S
Jagdlust	..do	..do	P/S
Winterveld (TCL)-North Section.	Rand Mines	..do	P/S
Groothoek	Samancor (Cromore)	..do	P/S
South Africa, Republic of—Con.			
Dilokong	LeBowa Development Corp. Ltd.	Stratiform	P/S
Montrose (Hendriksplaats).	Samancor (Cromore)	..do	P/S
Winterveld (TCL)-South Section.	Rand Mines	..do	P/S
Lavino (Grootboom)	Lavino S.A.	..do	P/S
Grasvally	Samancor (Cromore)	..do	P/S
Marico (Nietverdiend)	Vereeniging Refractories Ltd.	..do	P/S
Zeerust	Associated Ore & Metal Corp. Ltd.	..do	P/S
Tweefontein	Samancor (Cromore)	..do	P/S
Chromebroone	Leo Raphaely and Sons	..do	P/S
Turkey:			
Kefdag	Etibank	Podiform	P/S
Soridag	..do	Stratiform	P/S
Uckopru	..do	Podiform	P/S
Kavak	Turk Maaden Sirkati	..do	P/S
Kopdag West-North Zone.	Various private owners	..do	P/S
Kondak	Kondak Mining Co	..do	P/S
Zimbabwe:			
Glenapp-Ivo	African Chrome Mines Ltd.	Stratiform	P/S
Impinge	..do	..do	P/S
Sutton-Rodcamp	Zimbabwe Alloys Ltd	..do	P/S
Vanad	..do	..do	P/S
Caesar	..do	..do	P/S
Crown-Divide North	Zimbabwe Mining & Smelting Co.	..do	P/S
Glenapp-Hay-Noro	..do	..do	P/S
Umvukwes	..do	..do	P/S
Ore Recovery Tribute	..do	..do	P/S
Greenvale	Zimbabwe Alloys Ltd	..do	P/S
Maryland	..do	..do	P/S
McGowan	Rhoswa Mining & Developmentr Co.	..do	P/S
Divide	Zimbabwe Alloys Ltd	..do	P/S
Rutala	Rutala MInes (Pvt.) Ltd	..do	P/S
Umsweswe	Zimbabwe Mining & Smelting Co	..do	P/S
Umsweswe-Bee	African Chrome Mines Ltd.	..do	P/S
Windsor/York/York West.	Unknown	..do	P/S
Bat Claims	..do	..do	P/S
Lalapanzi	Zimbabwe Mining & Smelting Co.	..do	P/S
Netherburn	Zimbabwe Alloys Ltd	..do	P/S
York	Unknown	..do	P/S
Railway Block	Zimbabwe Mining & Smelting Co.	Podiform	P/S
Selukwe Peak	..do	..do	P/S
Valley Chrome	..do	..do	P/S
Magazine Hill	..do	..do	P/S
Iron Sides	..do	..do	P/S
Iron Ton	..do	..do	P/S
Belingwe District	Various owners	..do	P/S
Impinge (eluvial)	African Chrome Mines Ltd.	Eluvial	P/S

[1]Refers to either an individual mine or group of mines, or an area, section, or district depending upon the criteria of each individual nation under study.
[2]P/S, current or past producer; E, explored.

U.S. AND WORLD HISTORICAL PRODUCTION

Figure 2 shows world production of marketable chromite products in 5-yr intervals from 1960 through 1985. Total world production of chromite has more than doubled over the last 26 yr. The increase in annual world production has been shared almost equally between the MEC and centrally planned economy countries (CPEC's). Approximately 90 pct of the MEC increase over this period came from only one country—Republic of South Africa—which increased its annual production by 2.2 million mt and now accounts for 53 pct of MEC production and 31 pct of world production. Other significant MEC production increases since 1960 have occurred in India (+354,000 mt/yr), Finland (+272,000 mt/yr), Brazil (+266,000 mt/yr), and Turkey (+154,000 mt/yr of output). Significant decreases to annual output in major MEC producing countries since 1960 have occurred in the Philippines (−462,000 mt/yr) and Zimbabwe (−185,000 mt/yr).

The Soviet Union accounts for 75 pct of CPEC production and 31 pct of world production. Soviet chromite production is estimated to have increased by 2.0 million mt/yr since 1960. Albania has become the third largest world chromite producer and now accounts for roughly 9 pct of world production. As a group, CPEC's account for roughly 40 pct of world production.

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

The vast majority of the resources in the Republic of South Africa and Zimbabwe are mined by underground methods. The Republic of South Africa's mines use breast stoping to mine seams 0.4- to 1.8-m thick. The podiform deposits in Zimbabwe are mined using sublevel stoping, while the very thin seam deposits are mined using a method called resue mining.

In general, the mining method in use on the Bushveld Complex, breast stoping, does not vary a great deal from operation to operation since the seams all dip gently (5°–25°) and have similar structural and mineralogical characteristics. The most significant difference in terms of effect on mine operating cost lies with the thickness of the chromite seam in relation to the stoping height.

In the majority of the Republic of South Africa operations, the seam thickness is 0.9 m or greater and usually represents 90 pct of the stope height, which means that relatively little waste is produced and can be packed back with little trouble. However, some seams are as thin as 0.4 m, in which case fully half of the material blasted will be waste material. In this case, it is very difficult to pack all of the waste back and as much as 50 pct has to be transported to the surface. This can add as much as 60 pct to the mine operating cost on a crude ore basis, everything else being equal. In Zimbabwean Great Dyke seam mining, by comparison, the seams range in thickness from only 0.1 to 0.3 m, which means that with a typical stope height of 0.9 m, anywhere from 33 to 90 pct of the blasted material is waste, with generally 40 to 50 pct of the waste having to be hoisted to the surface. Because of this, the cost to mine 1 mt of chromite from the stratiform deposits of the Great Dyke is on average roughly four times higher than mining the stratiform deposits of the Bushveld Complex in the Republic of South Africa. Because 99 pct of world resources are contained in South African and Zimbabwean deposits, and since about 90 pct of Zimbabwe's chromite is contained in the Great Dyke, this difference in seam thickness and its effect on mining cost is highly significant.

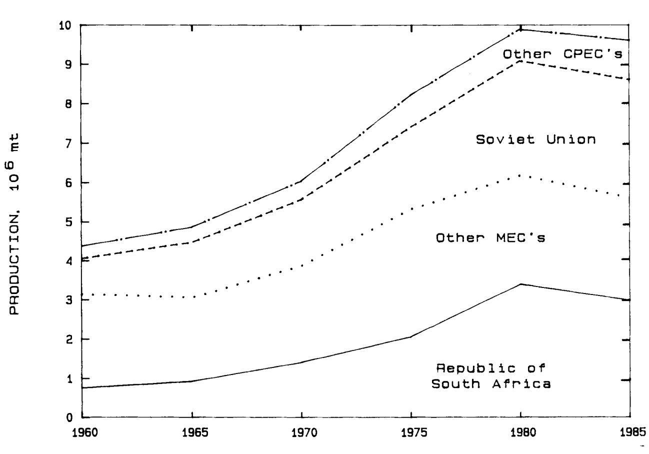

Figure 2.—World chromite production, 1960-85.

The underground mining of the podiform deposits in the Shurugwi and Belingwe Districts of Zimbabwe has involved many different mining methods over time. At Shurugwi, the prevalent present method is sublevel stoping, which is estimated to have a weighted-average mining cost of \$34/mt of recovered product. Other large-scale underground operations mining podiform deposits are the two major high-grade Philippines producers, the six major Turkish operations, and the operations in Greece and New Caledonia. Mining methods differ somewhat for all of these mines but all of the large-scale underground mines extracting ore from podiform deposits have extremely competitive mining costs.

Actual or proposed surface mining operations represented in this study include the great majority of the resources and properties in India and the Philippines as well as all of the properties in Brazil, Madagascar, and Finland. There is an extremely wide range of operational aspects (e.g., waste-to-ore ratios, haulage distances, and capacities) represented by the properties included in the surface mining category. Hence, the relative economics of these properties also show a wide range.

PROCESSING

In chromite processing, there are four major aims in producing marketable products. They are (1) separation of lump chromite from fines chromite; (2) upgrading the chromium oxide content to desired levels (generally above 40 pct Cr_2O_3); (3) elimination of as much excess iron (iron not contained in the chromite mineral structure) as possible; and (4) lowering of the silica content. The various flow sheets range from simple hand-sorting and screening operations to complex mills utilizing a combination of heavy media separation, magnetic separation, and multistage gravity separation. The separation of lump material from fines material and the upgrading of the chromium oxide content are especially important at operations producing products for the metallurgical industry, while the elimination of excess iron is particularly important in processing low-grade ores, especially soil deposits. The lowering of the silica content is very critical in producing chemical and refractory chromite products.

The separation of lump ore from fines ore is accomplished either by sorting and screening or by heavy media separation, the latter being more efficient under most circumstances. Magnetic separation is used at a limited number of properties to eliminate as much excess iron as possible. Nearly all of the analyzed operations require gravity separation methods to upgrade the chromium oxide content and/or lower the silica content, the only exception being some of the high-grade Great Dyke operations in Zimbabwe where only hand sorting and screening are needed to produce products sufficient for metallurgical uses.

From an economic standpoint, the beneficiation cost is nearly always less than mining or transportation. An exception to this are the low-grade alluvial-eluvial-lateritic soil deposits of the Philippines. Because of their very low grades (weighted average of only 2.0 pct Cr_2O_3) the overall concentration ratio for these properties is 30:1. This level is very high relative to the other evaluated properties which have concentration ratios that are usually less than 2:1.

PRODUCTION COSTS

Table 3 and figure 3 illustrate average production costs for five major producing countries. Chromite production in the Republic of South Africa and Turkey is clearly economic, as in production from the high-grade resources of the Philippines and India. Production costs for these resources range from \$57/mt to \$64/mt of chromite product at the 0-pct discounted-cash-flow rate of return (DCFROR) level and from \$62/mt to \$69/mt of chromite product at the 15-pct DCFROR level.[3]

On an overall basis, Zimbabwean production of chromite is clearly uneconomic owing to both the high cost of mining the Great Dyke seam resources and the high cost of transporting chromite through Port Elizabeth in the Republic of South Africa. This port has been handling effectively 100 pct of Zimbabwe's chromium exports since mid-1984. Zimbabwe has not exported any significant amounts of chromite since 1980; instead the mining industry converts chromite locally to high- and low-carbon ferrochromium and exports these products at a competitive cost level.

Similarly, the low-grade resources of the Philippines and India are not considered economic by the criteria of this analysis given that the sale of chromite product would have to cover the full cost of production. However, as stated earlier, no attempt is made in this study to report the availability of chromium products beyond chromite. As a result, some resources that appear to be uneconomic in terms of chromite may become economic if value-added processing to other chromium products occurs. For example, the low-grade resources of India (which appear uneconomic in terms of chromite) will likely be employed in the production of high-carbon ferrochromium (and thereby, may become economic) as this industry expands. As of 1985, a few of the low-grade resources of the Philippines have had sporadic production; one was brought into production in 1982 and two are indicated to be in development. Of these three properties, one is producing concentrates mostly for chemical use, one is planned to produce metallurgical grade concentrates, and one is proposed to produce prereduced pellets for sale to high-carbon ferrochromium producers.

Other producing countries, not shown in the table, include Finland, Brazil, Madagascar, New Caledonia, and Greece. Finland is a very economic producer of chromite. Brazil utilizes its chromite almost entirely for domestic consumption in the production of high-carbon ferrochromium. Madagascar and New Caledonia produce and export chromite primarily to Japan. As a standalone product, Greek chromite is uneconomic; however, Greece utilizes all of its domestic chromite production for domestic ferrochromium production.

For established mines with relatively long production histories, capital recovery and taxation expenses are minimal. For newer mines or nondeveloped deposits, capital recovery and taxation are significant costs especially when the criterion of earning a 15-pct DCFROR is imposed upon the analysis or where the chromite mining operation represents an uneconomic standalone producer of that product form.

[3]A comparison of f.o.b. postproduction costs for all chromite products is not precisely comparable with market prices because many chromite products are actually used internally for producing metallurgical products and in the country's own chemical and refractory markets.

Table 3.—Chromite production costs for producing operations in five selected MEC's

(All costs are in January 1985 U.S. dollars per metric ton of chromite product on a weighted-average basis)

Country, deposit type	Operating costs		Trans-portation cost[1]	Net operating cost	Recovery of capital[2]	0-pct DCFROR		15-pct DCFROR		
	Mine	Mill				Taxes and royalties[3]	Total cost[4]	Taxes and royalties[5]	Return on investment[6]	Total cost[7]
South Africa, Republic of	26	3	21	50	6	1	57	3	3	62
Zimbabwe:										
Seam type	100	3	81	184	17	1	202	8	11	220
Podiform type	34	7	70	111	2	1	114	2	24	139
Average	81	4	75	160	16	1	177	7	15	198
India:[8]										
High grade	33	2	18	53	3	1	57	4	3	63
Low grade	57	7	20	84	15	6	105	44	16	159
Philippines:[9]										
High grade	26	7	8	41	8	4	53	6	10	65
Low grade	31	52	13	96	19	4	119	19	17	151
Turkey	17	3	31	51	8	5	64	7	3	69

[1]Includes cost of transportation plus loading and handling charges f.o.b. the port of exportation.
[2]Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure as of January 1985, and investments required over the life of the operation.
[3]Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 0-pct DCFROR.
[4]Equal to the sum of total operating costs, taxation, and capital recovery determined at a 0-pct DCFROR.
[5]Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 15-pct DCFROR.
[6]The revenue increase per metric ton necessary to obtain a 15-pct DCFROR.
[7]Equal to the sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery *plus* the per metric ton increase necessary to provide a 15-pct DCFROR.
[8]Chromite resource separated into high and low grade for economic analysis. Low-grade resource represents chromite in the Cuttack and Koenjhar Districts grading 30 pct Cr_2O_3 on average.
[9]Chromite resource separated into high and low grade for economic analysis. Low-grade resource represents chromite in alluvial, eluvial, and lateritic deposits with a weighted average grade of 2 pct Cr_2O_3.

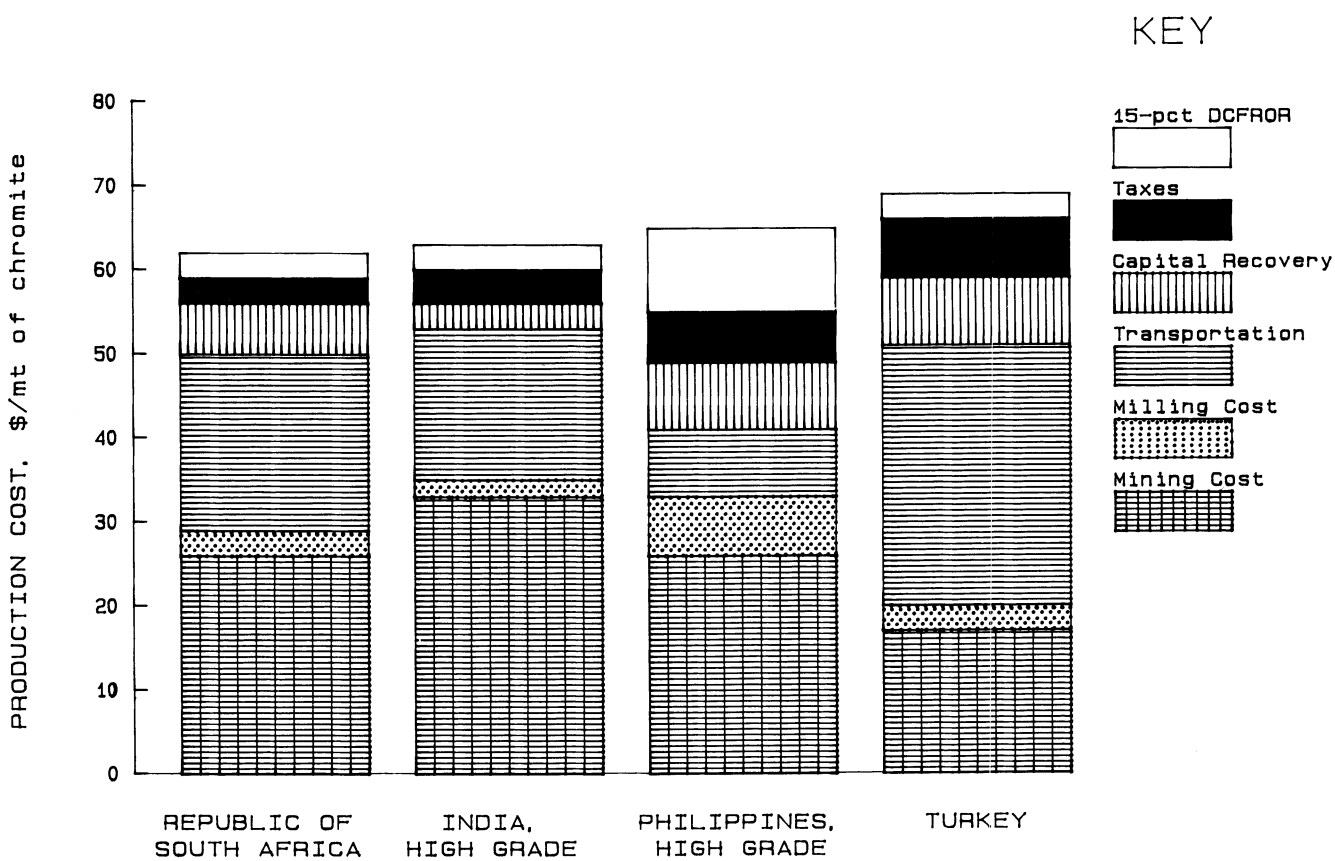

Figure 3.—Chromite production costs for producing mines in selected MEC's (January 1985 U.S. dollars).

AVAILABILITY

TOTAL RECOVERABLE

The 82 evaluated mines and deposits contain a total of 680 million mt of recoverable chromite. Figure 4 shows total recoverable chromite available at increasing total production costs determined at the 0- and 15-pct DCFROR levels. Chromite market prices vary from as low as $40/mt for the Republic of South Africa high-iron chromite (received on average during 1985) to a list price of $110/mt for Turkish high-chromium chromite (*1*). Individual product prices are as varied as chromite products themselves, which can be processed to the unique specifications of the customer. Prices also fluctuate with demand from year to year and are influenced by a host of factors, not the least of which are currency exchange rates.

In general, historical prices have been sufficiently high to render a great abundance of chromite economic. At $60/mt of chromite, for example, some 291 million mt (approximately 50 yr of MEC production at current rates) is economically recoverable at a breakeven (0-pct DCFROR) level. At $90/mt, over 516 million mt of chromite is economically recoverable (76 pct of the total evaluated or approximately 90 yr of MEC production). The only evaluated chromite resources that require in excess of $90/mt to cover total production costs are those of Zimbabwe, some of the low-grade resources of the Philippines and India, the Greek chromite mine that serves as part of an integrated ferrochromium operation, and one operation in Brazil that is currently not producing.

ANNUAL CAPACITY

Installed capacity in 1985 was more than sufficient to fulfill all demand for chromite for all markets. This is especially true of the chromite mines in the Republic of South Africa, which have a tremendous flexibility to expand capacity and production. In the case of the Republic of South Africa, if all 22 mines were fully developed and operated at capacity according to the development schedules of this evaluation, an average of approximately 5.3 million mt of chromite products could be produced annually, a level some 75 pct above 1985's actual production. This level of output could readily be met given a 2-yr lead time and at a nonprohibitive cost.

In the Republic of South Africa, it is most likely that future chromite capacity increases will come either as a result of increasing domestic ferrochromium production capacity (given the continuing decline of ferrochromium capacity in other chromite-importing countries) or from an increase in demand for chemical-grade concentrates. In Zimbabwe, any expansion of chromite mining capacity would be driven by an increase in ferrochromium smelting capacity. In particular, the Great Dyke seam deposits would most likely require an increase in low-carbon ferrochromium demand for all of that resource to be economically competitive in the long run.

Figure 5 shows 1985 chromite production potential from producing mines relative to increasing levels of cost at the 0-pct DCFROR level and operating cost. At a combined longrun operating cost of $60/mt, two-thirds (5.5 million mt) of 1985's potential annual chromite production was able to cover all operating costs. At a combined operating cost of $75/mt, approximately 87 pct (6.9 million mt) of production potential could cover all operating costs. The only operating mines with an operating cost above $75/mt are those in Zimbabwe.

Although nondeveloped deposits are not included on the curve, potentially significant deposits include the Ramu River deposit in Papua New Guinea, the Plum deposit in New Caledonia, and the Stillwater Complex in Montana. Although the Stillwater Complex was not evaluated in this study, the Bureau has recently completed an evaluation of the property (*6*). If developed, these deposits would add significantly to the availability of chromite outside of southern Africa. These deposits, however, all require higher prices to cover the total cost of development and operation, at least 3 yr to develop, and perhaps most importantly, a market for their product. With metallurgical consumption of chromite (in the form of ferrochromium production) continuing to shift to those countries that mine chromite, the availability of an export market for new producers of chromite is being reduced.

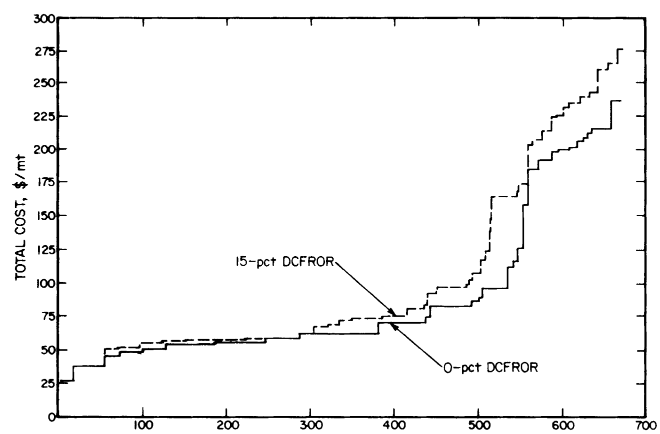

Figure 4.—Potential total chromite available from MEC's (January 1985 U.S. dollars).

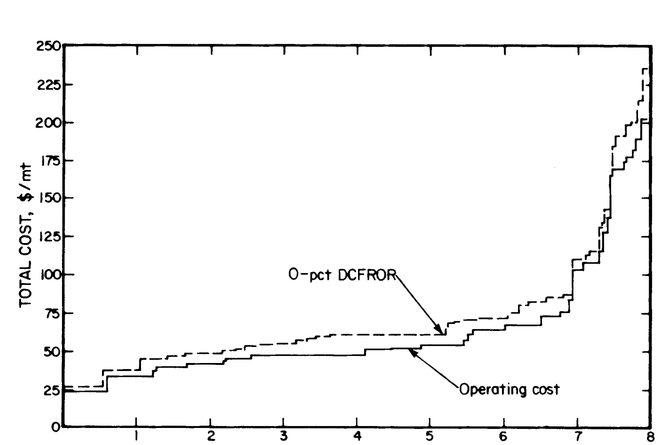

Figure 5.—1985 chromite capacity from producing mines in MEC's (January 1985 U.S. dollars).

CONCLUSIONS

World chromite resources are sufficient to satisfy consumption for the foreseeable future. The amount of chromite available from the demonstrated resources of the 82 mines and deposits evaluated in this study is sufficient to satisfy MEC consumption throughout the 21st century.

The Republic of South Africa possesses by far the world's largest resource and reserve base, as well as the largest installed capacity to produce chromite and potential to expand chromite capacity and production. Through its control of southern Africa's transportation infrastructure, the Republic of South Africa controls access to 99 pct of the world's known chromite resources and 87 pct of the total available chromite products evaluated in this study. It is the largest MEC producer of chromite and has the greatest influence on setting the floor price of high-iron chromite and charge grade, high-carbon ferrochromium. If production of chromium (in any product form) from the UG-2 reef were to eventuate, the dominance of the Republic of South Africa as a chromium producer would certainly increase.

Zimbabwe contains large chromite resources and possesses the ability to significantly expand chromite production capacity. It does not export chromite, however, primarily because of the high cost of mining Great Dyke seam deposits as well as the high cost of transporting chromite through the Republic of South Africa, currently its only available outlet.

The United States does not produce chromite and has no chromite reserves, although it does possess significant uneconomic resources in the Stillwater Complex of Montana and the podiform deposits of California and Oregon. In addition, other large chromite resources exist in New Caledonia and Papua New Guinea but have not as yet warranted exploitation. All other MEC chromite producers, in total, do not possess sufficient current production capacity nor a level of resources or economic conditions that are sufficient to replace the current dominance of the Republic of South Africa as a supplier of chromite.

REFERENCES

1. Papp, J. F. Chromium. Sec. in BuMines Commodity Summaries 1986, pp.34-35.

2. Thomas, P. R., and E. H. Boyle, Jr. Chromium Availability—Market Economy Countries. A Minerals Availability Appraisal. BuMines IC 8977, 1984, 86 pp.

3. DeYoung, J. H., M. P. Lee, and B. R. Lipin. International Strategic Minerals Inventory Summary Report—Chromium. U.S. Geol. Surv. Circ. 930-B, 1984, 41 pp.

4. Roskill Information Services, Ltd. The Economics of Chromium. 5th ed., 1985, p. 1.

5. Thayer, T. P. Chromium. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 111-121.

6. Wetzel, N., G. Gale, and S. Stebbins. Strategic Mineral Assessment of the Stillwater Complex, Southwest Montana. Sec. in BuMines Minerals and Materials, A Bimonthly Survey. Aug./Sept. 1985, pp. 22-31.

COBALT

INTRODUCTION

Cobalt is considered to be a critical commodity for the United States, owing to its extensive use in defense related industries and to U.S. reliance on imports for 100 pct of primary cobalt consumption. When a component of alloys, cobalt imparts heat and wear resistance, high strength, and superior magnetic properties. Its application in the manufacture of jet turbines and high-temperature resistive alloys makes cobalt a critical commodity for defense and other technologies. The strategic nature of cobalt to the United States, as well as the uncertain future of availability from its suppliers, encouraged the Government to develop a strategic stockpile of cobalt. Over the last decade, political instabilities in the African countries have encouraged the United States to increase the quantity of cobalt in the National Defense Stockpile. As of September 30, 1985, there was about 24,000 mt of cobalt on hand with an eventual goal of 38,737 mt (*1*).[1] These amounts are equal to about 3.5 and 6.3 yr, respectively, of apparent current domestic consumption.

Currently, there are no operating mines in market economy countries (MEC's) that produce cobalt as a primary commodity. All cobalt produced from mines is a byproduct of another commodity, principally copper or nickel, and to a much lesser extent, platinum-group metals (PGM's). Therefore, unless cobalt prices increase dramatically, cobalt availability will continue to be dependent on the production of other commodities. The determination of cobalt available from MEC copper production (specifically Zaire and Zambia) is of greatest importance since over 75 pct of 1985 MEC cobalt came from these copper mines.

A portion of the information presented in this chapter is an update of Bureau of Mines Information Circular 9012 "Cobalt Availability—Market Economy Countries. A Minerals Availability Program Appraisal" (*2*). More information is available from additional Bureau publications (*1*, *3-4*) and other sources (*5-6*).

GEOLOGY AND RESOURCES

GEOLOGY

Cobalt is present in various geologic environments (*7*). It is most often found in economically recoverable quantities when associated in three major types of ore deposits: (1) stratabound copper deposits (Zaire and Zambia), (2) copper-, nickel-, and platinum-enriched magmatic deposits (Canada, Republic of South Africa, and the United States), and (3) nickel-cobalt laterite deposits (New Caledonia, Philippines, Australia, and the United States). Although additional cobalt can be recovered from other deposit types, over 95 pct of MEC cobalt production originates from the first two types. Table 1 lists properties included in this study.

The stratabound sedimentary copper belts in Zaire and Zambia are the largest producing cobalt regions in the world. The belts are Precambrian in age and consist of low- to medium-grade metasediments. They are more than 500 km long and 30 km wide. The copper occurs in both sulfide and oxide zones. Overall average copper grades generally vary between 1.8 and 5 pct while cobalt grades range between 0.13 and 0.42 pct. Cobalt grades may vary greatly within the ore bodies.

The magmatic deposit type is a broad category that includes the Sudbury Complex and Thompson Nickel Belt District in Canada, Republic of South Africa Merensky Reef, and the Duluth Gabbro Complex in the United States. The Sudbury Complex consists of a generally differentiated magmatic body that contains a mineralized unit called the nickel irruptive. The Sudbury Complex is the largest nickel-producing district in the world. Nickel grades generally range between 0.9 and 2.5 pct. In addition to nickel, the complex produces copper, cobalt, PGM's, and other precious metals. Cobalt grades are variable and generally range between 0.02 and 0.08 pct with an average of about 0.04 pct. The Thompson Nickel Belt in Manitoba contains several ore bodies that are currently being worked. Cobalt grades are usually lower than Sudbury, generally ranging between 0.02 and 0.06 pct.

The Merensky Reef contains significant quantities of recoverable PGM's from which byproduct nickel, copper, and minor amounts of cobalt are recovered. Cobalt content in the ore is variable and is very low grade, generally about 3 ppm.

The Duluth gabbro region, in Minnesota, is one of the world's largest basic igneous intrusions and, although low grade, is also the largest single nickel-cobalt resource in the United States. Cobalt grades are quite low, ranging between 0.01 and 0.02 pct. Locally there has been extensive exploration but no active mining as yet.

The Blackbird deposit (Idaho), a past producer, contains sulfides and arsenides along with copper-cobalt mineralization. The Precambrian deposit is believed to be syngenetic in origin with subsequent remobilization during diagenesis,

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

Table 1.—MEC cobalt properties included in this study

Country and property name	Owner	Primary commodity	Current status[1]	Mining method[2]
Australia:				
Agnew	Agnew Mining Co.	Ni	P	U
Greenvale	Metals Exploration, Freeport	Ni	P	S
Kambalda	Western Mining Corp.	Ni	P	U
Mt. Keith	Metals Exploration, Freeport	Ni	N	S
Botswana: Selebi-Phikwe	BCL Ltd.	Ni	P	U
Brazil:				
Niquelandia (CNT)	Companhia Niquel De Tocantins	Ni	P	S
Niquelandia (Codemin)	CODEMIN	Ni	N	S
Canada:				
Manitoba: Inco Operations[3]	Inco	Ni	P	C
Ontario:				
Falconbridge Operations[3]	Falconbridge	Ni	P	U
Inco Operations[3]	Inco	Ni	P	C
Saskatchewan: Key Lake	Key Lake Mining Corp.	U	N	S
Finland:				
Keretti Mine	Outokumpu Oy	Cu	P	U
Luikonlahti	Myllykoski Oy	Cu	P	U
Vuonos Mine	Outokumpu Oy	Cu	P	U
Guatemala: Eximbal	Inco and Hanna	Ni	S	U
India: Sukinda	Tata Iron and Steel	Ni	N	S
Indonesia: Gag Island	P.T. Pacific Nickel	Ni	N	S
New Caledonia:				
Goro	Inco	Ni	N	S
Ile Art	Cofremmi	Ni	N	S
Kouaoua	Societe Le Nickel	Ni	P	S
Moneo	OGMC Ballande	Ni	P	S
Nakety	Societe Le Nickel	Ni	P	S
Nepoui	..do	Ni	P	S
Ouaco	..do	Ni	P	S
Ouinne	..do	Ni	P	S
Poro	..do	Ni	P	S
Poum	Cofremmi	Ni	N	S
Prony	Penamex	Ni	N	S
Thio	Societe Le Nickel	Ni	P	S
Tiebaghi	Cofremmi	Ni	N	S
Philippines:				
Infanta	Philippine Government	Ni	N	S
Nonoc Mine	Nonoc Mining and Industrial Corp.	Ni	P	S
Soriano	Soriano Corp.	Ni	N	S
South Africa, Republic of:				
Der Brochen	Geduld proprietary, East Rand Construction	PGM	N	U
Impala	Platinum Mines Union Corp. and others	PGM	P	U
Rustenburg	Rustenburg Platinum Ltd.	PGM	P	U
Western Platinum	Western Platinum Ltd.	PGM	P	U
Uganda: Kilembe	Uganda Government	Cu	N	U
United States:				
Alaska: Yakobi Island	Inspiration Development Co.	Cu, Ni	N	U
California:				
Pine Flat area	Hanna Mining Corp.	Ni	N	U
Gasquet	California Nickel Corp.	Ni	N	S
Idaho: Blackbird	Noranda	Co, Cu	N	U
Maine: Crawford Pond	Knox Mining Corp.	Ni	N	U
Minnesota:				
Birch Lake	Inco, Hanna, Duval	Cu, Ni	N	C
Dunka River	AMAX	Cu, Ni	N	U
Ely Spruce Mine	Inco	Cu, Ni	N	S
Minnamax	AMAX	Cu, Ni	N	U
Partridge River	United States Steel	Cu, Ni	N	S
Missouri: Madison	Anschutz Corp.	Co, Ni, Cu	N	U
Oregon: Red Flat	Hanna Mining, Red Flats	Ni	N	S
Puerto Rico: Guanajibo	Puerto Rican Government	Ni	N	S
Zaire				
Dikuluwe, Mashamba	Gecamines	Cu	P	S
Kakanda-Diselle	..do	Cu	P	S
Kambove	..do	Cu	P	S
Kamoto underground mine	..do	Cu	P	U
Kov open pit	..do	Cu	P	S
Tenke Fungurume	..do	Cu	N	U
Zambia:				
Baluba	Zambian Consolidated Copper Mines (ZZCM)	Cu	P	U
Chibuluma	..do	Cu	P	U
Nchanga Div	..do	Cu	P	U
Nkana Div	..do	Cu	P	U
Zimbabwe:				
Empress	Rio Tinto Mining	Ni	N	U
Shangani	Trojan Nickel Mine Ltd.	Ni	P	U
Trojan	..do	Ni	P	U

[1] P, producing; N, nonproducing; S, standby.
[2] S, surface; U, underground; C, combined surface and underground.
[3] Cobalt availability and production is analyzed on a combined operational basis.

tectonism, and metamorphism to form veins. Cobalt grades average about 0.6 pct but exceed several percent at some locations.

The cobalt, copper, and nickel mineralization of the deposit at the Madison Mine, a past producer in Missouri, is developed along the contacts between sedimentary units and basement, or as replacement features in sandstones. Potential cobalt grades average about 0.2 pct with additional potential for recoverable cobalt in tails.

Other magmatic deposits with recoverable byproduct cobalt are located in Alaska, Australia, Botswana, and Zimbabwe, with cobalt grades averaging about 0.2 pct.

Nickeliferous laterites are currently considered a relatively minor source of cobalt production and supply, but may have large potential for the future. Laterites form as a result of chemical weathering of ultrabasic bedrock, generally in equatorial regions. Nickel and cobalt are leached from decomposing bedrock by the downward percolation of rainwater and are redeposited at depth by chemical precipitation. This process may result in a zone of enrichment, which, in some cases, can be mined. Laterite ores are mined primarily for their nickel content, which generally exceeds 1.0 pct. Cobalt grades rarely exceed 0.2 pct. Laterites are basically of two types, siliceous and limonitic. The limonitic ores are most amenable to cobalt recovery, generally by hydrometallurgical methods.

Ocean crusts and nodules have been discovered over large areas of the ocean floor. These deposits contain potentially recoverable quantities of cobalt, nickel, copper, and manganese, as well as other metals. Crusts and nodules with cobalt contents of up to 2.0 pct have been collected. Average grades range between 0.27 and 0.63 pct Co. The mechanism of origin of this potential resource is still not fully understood.

RESOURCES

Figure 1 shows identified cobalt resources, reserve base, and those demonstrated resources evaluated in this study. The world reserve base estimate is larger than the resource estimate for the properties evaluated in this study since it includes a larger area of the Duluth Gabbro Complex (U.S. resource estimates) and also includes centrally planned economy countries (CPEC's). The "others" category (fig. 1) is larger in the reserve base because it includes deposits for which engineering and cost data were unavailable. The tonnages used in this study are based on demonstrated resources in MEC's for which there was information that could be used to relate the resource characteristics to appropriate mining and processing technologies and their associated costs.

As listed in table 2, total potentially recoverable cobalt from the deposits evaluated is approximately 1.7 million mt. Copper properties compose about 40 pct of the total while nickel or nickel-copper deposits make up about 55 pct. The remaining portion is in PGM and cobalt properties.

Zaire's copper deposits contain the largest cobalt resource, followed by New Caledonia's nickel laterite resources. The largest currently mined cobalt resource is in the African copper belts of Zaire and Zambia. Based on the demonstrated resources of these deposits, there is in excess of 543,000 mt of recoverable cobalt in producers in Zaire and at least an additional 94,000 mt available from nonproducers. There is 73,000 mt of recoverable cobalt from

Table 2.—Summary of MEC demonstrated cobalt resources evaluated for this study, as of January 1985[1]

Country	Primary commodities	Producing mines: Number	Producing mines: Recoverable cobalt, 10^3 mt	Nonproducing mines: Number	Nonproducing mines: Recoverable cobalt, 10^3 mt	Total recoverable cobalt, 10^3 mt
Africa:						
Botswana	Ni, Cu	1	3	0	0	3
South Africa, Republic of[2]	PGM	3	14	1	2	16
Zaire	Cu	3	543	1	94	637
Zaire	Cu	3	73	0	0	73
Zimbabwe[3]	Ni, Cu	2	1	1	1	2
Southwest Pacific:						
Australia	Ni	2	13	1	15	28
Indonesia	Ni	0	0	1	27	27
New Caledonia	Ni	7	5	6	505	510
Philippines	Ni	1	63	2	59	122
Europe: Finland	Cu	1	4	0	0	4
India	Ni	0	0	1	7	7
South America:						
Brazil	Ni	1	12	1	5	17
Guatemala	Ni	0	0	1	10	10
North America						
Canada	Ni, Cu	16	50	9	16	66
United States	Cu, Ni, Co	0	0	20	160	160
Total		40	781	45	901	1,682

[1]Data for all countries, except Zaire and Zambia, are based on January 1981 resource data, which was updated to January 1985 by subtracting estimated production from operating mines.

[2]Because cobalt is a very minor revenue producer, the Republic of South Africa was not economically evaluated as a cobalt resource.

[3]Additional recoverable very low grade cobalt resources are present in the Great Dyke PGM prospects, but are only available at very high cost.

producers in Zambia. Less than 1 pct of New Caledonia's estimated cobalt resource is in producers.

About 60 pct (950,000 mt) of the total evaluated cobalt resource is in sulfides, of which about 72 pct is in mines currently producing or on standby. The remaining 40 pct (750,000 mt) is from laterite deposits, of which less than 15 pct is in properties currently producing or on standby. Nearly 84 pct of this resource is located in New Caledonia and the Philippines, with the remaining amounts available from Australia, India, and the United States.

The United States contains large amounts of potentially recoverable cobalt in copper-nickel, nickel-cobalt, and cobalt deposits. None of these deposits, however, are producing. About half is in the copper-nickel Duluth gabbro properties in Minnesota, which contain approximately 77,000 mt of potentially recoverable cobalt from about 3 billion mt of ore. The Blackbird, ID, Gasquet, CA, and Madison, MO, deposits are probably the most economically viable U.S. resources. During the late 1970's and early 1980's, there was intensive exploration and some preproduction development on these properties. Combined, these three U.S. deposits account for approximately 100,000 mt of additional potentially recoverable cobalt.

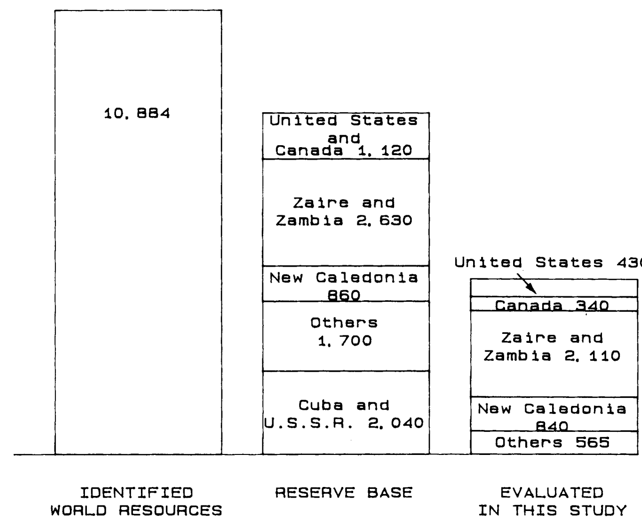

Figure 1.—Estimates of world cobalt resources (thousand metric tons of contained cobalt).

U.S. AND WORLD HISTORICAL PRODUCTION

In the last several years, efforts at cobalt price control by the two major producers (Zaire and Zambia) have been periodically successful, despite low demand. The most dramatic event affecting cobalt prices occurred in 1978 when the cobalt-producing Shaba Province in Zaire was invaded by insurgents. The threatened cutoff of cobalt to the West increased the cobalt price from about $10/lb in 1978 to over $35/lb in 1979. (It was during this time that cobalt was being shipped to the West from Zaire by aircraft. Sabotage to Zaire's transportation system remains the most sensitive aspect for causing interruption of the supply of cobalt to the MEC's.)

A recent Bureau article published in American Metal Market explains the reasons for the volatility of cobalt prices during the last 10 yr (*8*). Briefly, from 1967 to 1976, an average of 6 million lb/yr of cobalt was sold from the National Defense Stockpile, providing for about one-third of domestic needs. When this practice was discontinued in 1976, Zaire, unable to meet customer needs, allocated production at the rate of 70 pct of what had been supplied over the last 16 months. These reductions, combined with the invasion of the Shaba Province that resulted in a suspension of cobalt mining and refining operations, created customer uncertainty and a fear of a severe cobalt shortage. As a result, cobalt prices rose from $6.40/lb in February 1977, to $25.00/lb in February 1979. Since that time, cobalt prices have fluctuated greatly with spot prices as low as $3.90/lb in 1986 (*1*). In recent years, Zaire and Zambia have attempted to stabilize cobalt prices. Although they had some success in 1984 and 1985, 1986 prices once again began to fluctuate.

World cobalt mine production was about 32,000 mt in 1985 (refined cobalt production is significantly lower owing to processing losses). Figure 2 illustrates historical cobalt mine production in 5-yr intervals, which is dominated by Zaire and Zambia. Zaire alone accounted for over 50 pct of total world mine production (39 pct of metal production) and Zambia for about 12 pct (14 pct of metal production) from cobalt originating entirely from copper ore. The U.S.S.R. and Cuba produced approximately 4,145 mt or 11 pct of world mine production (19 pct of metal production). The majority of the remaining portion of world mine production originated from Australia, Canada, Finland, and the Philippines. These four countries account for about 20 pct of the total.

In 1985, Zaire's cobalt metal production resulted from the processing of ore that yielded approximately 410,000 mt of copper metal. An additional estimated 130,000 mt of copper was produced from mines in the country from which cobalt was not recovered.

Although Zairian cobalt refining capacity is approximately 17,000 mt of recoverable cobalt metal, the average production for the 1982-85 period was less than 8,000 mt (recent peak cobalt metal production was 15,000 mt in 1980). During the same period (1982-85), copper production was nearly 95 pct of production capacity. The ratio of copper-cobalt recovery may vary on year-to-year basis because of wide swings in cobalt grade and recoveries and a need to maintain a supply-demand balance for cobalt. Availability of this byproduct, therefore, is dependent on these factors more so than full copper capacity.

Zambia is the second largest cobalt producer. The country's production of copper and byproduct cobalt, government-operated and owned by Zambia Consolidated Copper Mines (ZCCM), resembles that of copper operations in Zaire. It is estimated that Zambia's cobalt capacity is approximately 5,000 mt; however, production has only utilized about 65 to 70 pct of the capacity in recent years. Like Zaire, Zambia does not recover cobalt from some of its copper concentrates. In 1985, Zambian cobalt mine production reached 4,600 mt.

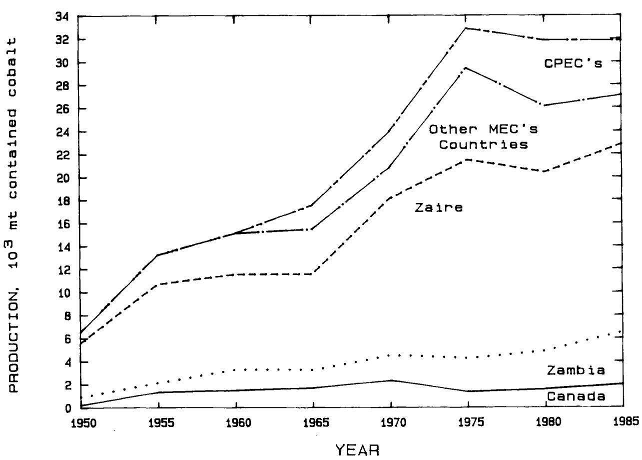

Figure 2.—World cobalt production, 1950-85.

Canada is the largest producer of cobalt as a byproduct of nickel mining and the third largest cobalt producer among MEC's, mining approximately 3,000 mt of recoverable cobalt in 1985. Like Zaire's copper mines, the relationship between Canada's nickel and cobalt production vary. The most important variables include cobalt grades and recoveries. Overall cobalt recovery is estimated at nearly 45 pct but technical advances may take place, resulting in increased production. At full nickel production, Canada has the potential capacity to produce over 2,200 mt of refined cobalt annually. An additional 1,800 mt of refined cobalt can originate from Norway, which processes Falconbridge's Canadian production.

Australia's cobalt production is also primarily a byproduct of nickel, a large portion of which is from laterites. Peak production from Australian sulfide and laterite operations was nearly 3,400 mt in 1977, but recent production averaged about 1,200 mt (835 mt in 1985).

The U.S.S.R. and Cuba's annual cobalt mine production capacity is about 4,150 mt or about 11 pct of world production. Nearly all of this production is consumed in the Soviet bloc.

From 1982 to 1985, the United States depended on Zaire for 40 pct of its imported cobalt requirements, Zambia—16 pct, Canada—18 pct, Norway—6 pct (most of which originated from Falconbridge's Canadian nickel matte, therefore, Canada's actual total share is about 24 pct) and other countries, including Australia and Botwsana *(3)*. Much of the material from Botswana and Australia was refined at AMAX's Port Nickel refinery in Louisiana. In 1985, AMAX closed its Port Nickel refinery, which eliminated all domestic refining of cobalt.

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

Cobalt is recovered as a byproduct of metals that are mined by both surface and underground methods. Table 2 shows the mines and prospects evaluated in this study. Most of the cobalt originating from the copper mines in Zaire and Zambia is recovered using open pit mining. These mines require blasting of rock and the construction of benches in order to truck the ore from the pit. Waste-ore ratios are variable but average about 5:1. Ores in Zaire and Zambia generally consist of sulfide and oxide ores, which require different processing methods. Therefore, efforts are made to mine them separately.

Underground mines in Zaire and Zambia usually employ sublevel open stoping. Australian and Canadian nickel ores are mined primarily by various underground methods. Application of highly mechanized vertical crater retreat mining and other bulk mining methods are becoming increasingly important in an effort to reduce costs, especially in Canada; however, ground support problems have been encountered.

Cobalt-bearing nickel laterites, such as those in New Caledonia and the Philippines, are mined by surface methods. Because laterites are generally of an unconsolidated nature, scrapers, shovels, and front-end loaders are used for mining and loading ore into trucks.

PROCESSING

Beneficiation techniques for the recovery of cobalt vary with ore chemistry. In Zaire and Zambia, revenues from copper are of greater importance than cobalt; therefore, efforts to maximize recovery of copper are usually prioritized at the expense of cobalt. The copper-cobalt mines in Zaire and Zambia require specialized beneficiation methods owing to the presence of several types of ores. Ore, depending on its type, is directed to a specific concentrator designed to treat that specific ore chemistry. Cobalt feed grades may vary from 0.1 to 2.5 pct, but average 0.33 pct with an average mill recovery of approximately 35 pct. Recovery values are low because of variable cobalt grades and efforts to maximize copper recovery at the expense of cobalt.

Most cobalt in Zaire is processed through the DIMA-Kamoto concentrator (8 million mt/yr ore feed). This plant can produce a low-cobalt, high-copper concentrate of 0.5 pct Co and 60 pct Cu or a high-cobalt, low-copper concentrate grading 4 to 5 pct Co and 45 pct Cu. Average overall concentrator recovery of cobalt can be as low as 18 pct. The average recovery, however, is approximately 35 pct. Cobalt associated with oxide mineralization is not recovered.

The metals contained in the concentrates sent to the Luilu or Shituru plant are extracted and separated through leaching, precipitation, and electrolysis. Some copper-cobalt concentrates are shipped to Zambia and Belgium for refining. The cobalt contained in concentrate shipped to the Lubumbashi smelter is not recovered, instead, it is incorporated in the smelter slag. As much as 10,000 mt/yr of cobalt ends up in the slag.

Nickel-cobalt sulfide ores, such as those at INCO's Sudbury operations in Canada, are directed to beneficiation plants where two concentrates, nickel and copper, are produced. Although there is some cobalt contained in the copper concentrates, it is not recovered. Nickel concentrate, from which cobalt is recovered, is sent to the Copper Cliff smelter-refinery complex. A resulting cobalt mixture is recovered from INCO's nickel refining circuit. The cobalt residues are sent to refineries for upgrading. Falconbridge's cobaltiferous nickel-copper matte is sent to its Norwegian refinery for processing.

Historically, two problems have existed in processing of nickel laterites: (1) high energy consumption and (2) inadequate recovery of metals, especially cobalt. Metals in laterite ores are recovered by pyrometallurgical and hydrometallurgical methods. The pyrometallurgical methods entail smelting the laterite to a matte followed by electrolytic separation to recover the cobalt, as is done with matte originating from sulfide ores. Relatively little cobalt is recovered from laterites using this process. The majority of cobalt from laterites is recovered by hydrometallurgical methods. These methods include variations on ammoniacal and sulfuric leach methods. The ammoniacal method was first applied to Cuban laterites during World War II, but low nickel recoveries, high energy costs, and very disappointing cobalt recoveries have rendered this process uneconomic. The sulfuric acid leach process is capable of higher recovery rates, about 90 pct for nickel and cobalt, and relatively low energy consumption. The resulting mixed sulfide concentrate is refined further to recover nickel and cobalt.

NEW TECHNOLOGY

There are mining operations, such as in Zaire, which currently pass cobalt contained in pyritic mine wastes, tails, and smelter slags. New, relatively low-cost methods of recovering this cobalt are being researched.

Canadian mining companies are continuing the development of lower cost bulk mining techniques.

Research continues on developing low-energy methods of recovering nickel and cobalt from laterites and recovering metals from ocean crusts and nodules.

PRODUCTION COSTS

Virtually all production costs for the recovery of cobalt, from mining through refining, are secondary to those incurred for the primary commodity, copper or nickel. For this reason, it is difficult to acquire or estimate separate treatment costs for cobalt. However, an estimated operating cost for the recovery of cobalt in the refinery ranges between \$2/lb and \$3/lb. For these operations, the production of copper and nickel are of the highest priority. A discussion on costs associated with Zaire and Zambia copper operations is presented in the copper chapter and a similar discussion on nickel mines presented in the nickel chapter.

AVAILABILITY

In order to determine the availability of cobalt from MEC's, the Bureau evaluated demonstrated cobalt resources in selected copper mines in Zaire and Zambia and nickel mines in Australia, Canada, Finland, New Caledonia, the Philippines, the United States, and other countries. Actual cobalt grade, recovery data, and other pertinent information is difficult to acquire since cobalt is recovered as a byproduct. The availability of cobalt is limited by the necessity of mining for the primary commodity, the plant's capacity and design, and the additional cost to recover the cobalt.

COBALT AVAILABLE FROM COPPER

Virtually all of the cobalt recovered from copper ores, and at least 75 pct of the annual MEC cobalt mine production, originates from Zaire and Zambia. The Bureau evaluated 10 copper mines and districts in these two countries; current cobalt production originates from seven of these operations. Two operations, Kakanda-Diselle and Kambove in Zaire, contain cobalt in the ore, but it is not currently recovered. There are no known plans to modify these operations for cobalt recovery. The remaining property, Tenke Fungurume in Zaire, is partially developed and may not begin production for several years. It is not known whether cobalt would be recovered from this potential operation.

There is a total estimated recoverable cobalt resource of 637,000 mt in Zaire and approximately 73,000 mt in Zambia. About 87 pct (616,000 mt) of the total resource (710,000 mt) from these two countries is in currently producing mines that, at a 0-pct discounted-cash-flow rate of return (DCFROR), have a weighted-average total production cost of about \$0.55/lb copper. It is likely that there are additional costs to cover security, repayment of debt, plus other overhead, which can not be attributed to any one mining operation.

The two countries have a combined estimated annual capacity of nearly 22,000 mt of refined cobalt, which is about equivalent to the entire 1985 MEC cobalt metal production. Market conditions, however, generally dictate what portion of total capacity will be utilized.

Zaire has an annual cobalt refining capacity of about 17,000 mt, although average annual cobalt production between 1982 and 1985 was about 8,000 mt (recent peak production was 14,000 mt in 1979).

Zambia has an annual cobalt capacity of approximately 5,000 mt, however, only about 65 to 70 pct has been utilized in recent years. The refinery at Chambishi has a capacity of 2,400 mt of cobalt and the new Nkana refinery has a capacity of 2,600 mt. In 1985, Zambian cobalt production reached 3,475 mt. Annual production of cobalt from these two countries can vary widely on a year-to-year basis. This results from (1) large variations in cobalt grade, despite relatively consistent copper grades, which hinder efficient recovery; (2) technical problems associated with poorly maintained and antiquated equipment; and (3) the passing of cobalt through circuits, without recovery, during periods of low metal prices.

Zaire and Zambia were evaluated at a combined annual refined cobalt production level of approximately 17,400 mt (13,000 in Zaire and 4,400 in Zambia) because the countries have not historically utilized their total production capacity of about 22,000 mt. Based on the results of this study, all of this capacity is available at a total cost of production, including a 0-pct DCFROR, of less than $0.55/lb copper (the average 1985 copper price was about $0.66/lb). Based on demonstrated resource estimates these countries could maintain production at this level through at least 1995.

COBALT AVAILABLE FROM NICKEL

Potentially recoverable cobalt from evaluated nickel resources amounts to approximately 928,000 mt and is distributed as follows: New Caledonia—55 pct, Philippines—13 pct, Canada—7 pct, and others—25 pct. Only about 16 pct is in producers.

Approximately 66,000 mt of byproduct cobalt from 8.25 million mt of nickel is potentially available from the nickel mines and prospects evaluated in Canada. About 50,000 mt is potentially available from producers at a cost of less than $2/lb nickel (at a 0-pct DCFROR). The published dealer cathode price was about $2.35/lb in January 1985, but the metal could be bought for considerably less. Canada's Sudbury and Thompson Districts contain virtually all of the evaluated resource. Like the African copper mines, the ratio between Canada's nickel and cobalt production changes because of variable cobalt grades and associated recoveries. Canada has the capacity to produce between 4,000 and 5,000 mt of cobalt annually (including Falconbridge facilities in Norway). In 1985, approximately 4,000 mt of cobalt metal was produced from Canadian nickel ore.

The Philippines contain approximately 122,000 mt of recoverable cobalt, which is probably a somewhat conservative figure. In 1985, about 900 mt of cobalt in nickel laterite ore was produced, although there is a potential capacity to produce in excess of 1,500 mt of cobalt in mixed sulfides and cobalt extracted from raw ore shipped to Japan. High grading and other temporary cost-cutting measures allow nickel to be produced at prices close to the market price, however, suspension of activities at Nonoc will likely result if metal prices do not improve.

Australia's cobalt production is also a byproduct of nickel. Approximately 28,000 mt of potentially recoverable cobalt is contained in the evaluated properties, about half of which is in producers. Two-thirds of the cobalt production originates from the Greenvale deposit. Although Australian sulfide operations are cost competitive with Inco's, the Greenvale laterite operation is more costly. The weighted-average total costs for producing Australian nickel properties is about equal to or slightly higher than $2/lb nickel but less than the January 1985 published dealer cathode price.

Only 1 pct of the New Caledonia laterite resource is in producers and none of the evaluated resource is currently available at a total production cost of less than the January 1985 nickel price ($2.35/lb). New Caledonia does not have any nickel or cobalt refining facilities, instead, the cobalt is recovered at facilities in France. Less than 800 mt of cobalt per year originates from New Caledonia and is not likely to increase in the near future. (Most New Caledonian nickel production is ferronickel.) There is a possibility that New Caledonia will begin shipping laterites to the Greenvale operation in order to supplement the company's diminishing resource.

The largest domestic cobalt resource is contained in the Duluth Gabbro Complex, a copper-nickel resource. Although the deposit may contain approximately 77,000 mt of recoverable cobalt, the resource is huge (over 3 billion mt) and is very low grade (about 0.01 pct). There are no known plans for development to produce cobalt in the near future.

The Madison Mine, in Missouri, is also associated with nickel and copper. This former producer contains an estimated 10,000 mt of potentially recoverable cobalt in ore and mill tailings. Approximately 30 pct of the Madison resource could be mined profitably at less than $10/lb cobalt. The entire evaluated resource, however, would require a total production cost, including a 15-pct DCFROR, of over $20/lb. Original mine feasibility plans called for an annual production of about 900 mt of refined cobalt along with byproduct nickel and copper.

The Gasquet nickel-cobalt laterite project in California is also an important potential domestic cobalt producer. Approximately 15,000 mt of cobalt is contained in the laterite resource and is available at an average total cost of production, including a 15-pct DCFROR, exceeding $20/lb of cobalt. The property has been partially developed, but no plans for production have been announced. Owing to low metal prices, it is doubtful that production will begin in the near future. Operating plans anticipated an annual refined cobalt capacity of about 1,000 mt, 8,600 mt of nickel, chromite concentrate, and magnesium oxide.

OTHERS

The Blackbird deposit, in Idaho, contains high-grade cobalt (0.60 pct) with as much as 6 million mt of potential ore yielding approximately 25,000 mt of refined cobalt. This potential cobalt producer occurs in association with copper, but production is dependent on revenues from cobalt. Although a small portion of the resource may be currently profitable to mine, the entire resource could not be mined at a total production cost, including a 15-pct DCFROR, of less than $20/lb. Average potential annual refined cobalt production is estimated at 1,500 mt and copper at 4,400 mt.

CONCLUSIONS

MEC cobalt availability is a function of copper and nickel production, market demand, and political stability. These elements are critical because the United States is completely dependent on foreign sources for primary cobalt. The largest shares of cobalt imports to the United States come from Canada (24 pct, including Norway), Zaire (40 pct), and Zambia (16 pct). Barring any major interruptions in supply, there is sufficient byproduct cobalt capacity from producing copper and nickel operations to meet current and projected levels of MEC cobalt demand for at least the next 10 yr.

Although there have been successful efforts in finding substitutes for cobalt superalloys and magnets, it is still a very important metal for military applications. The U.S.

strategic stockpile takes on special significance when considering the element of political instability in the African countries and the resulting potential for an interruption of supply. Currently, there is underutilized capacity in the cobalt recovery plants of Zaire and Zambia, as well as underutilization of the nickel operations of Canada. An increase in demand could be easily met by current producers. If there were a temporary interruption of supply from the African nations, stepped-up Canadian production of cobalt is unlikely, since it is a minor byproduct of nickel.

The United States does have cobalt available from its own natural resources, but because of the high costs for development the low-grade byproduct nature in most deposits, current low metal prices and underutilized capacity, no near-term developments are expected. However, if development were initiated in the United States, the Blackbird, Gasquet, and Madison deposits would most likely be developed first. There is about 50,000 mt of potentially recoverable cobalt in these three prospects. However, analyses indicate that cobalt prices of nearly $20/lb would be required to encourage full-scale development and production at all three operations. If they were to be developed concurrently, and were based on currently available mining plans, there could be a combined annual production of about 3,500 mt of cobalt, equaling less than half of 1985 U.S. apparent consumption.

REFERENCES

1. Kirk, W.S. Cobalt. Sec. in BuMines Mineral Commodity Summaries 1987, pp. 38-39.

2. Mishra, C.P., C.D. Sheng-Fogg, R.G. Christiansen, J.F. Lemons, Jr., and D.L. DeGiacomo. Cobalt Availability—Market Economy Countries. A Minerals Availability Program Appraisal. BuMines IC 9012, 1985, 33 pp.

3. Kirk, W.S. Cobalt. Sec. in BuMines Mineral Commodity Summaries 1986, pp.38-39.

4. Peterson, G.R., D.I. Bleiwas, and P.R. Thomas. Cobalt Availability—Domestic. A Minerals Availability Program Appraisal. BuMines IC 8848, 1981, 31 pp.

5. Brook, Hunt and Associates Ltd. (London). Cobalt, A Review and Outlook to 1985. 1981, 101 pp.

6. Shearson-Lehman (London). Annual Review of the World Copper Industry 1984-85. 1985, 102 pp.

7. Vhay, J.S., D.A. Brobst, and A.V. Heyl. Cobalt. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 143-155.

8. Kirk, W.S. A Third Pricing Phase: Stability? Am. Met. Market, Aug. 23, 1985, v. 93, No. 163, pp. 9, 12.

COLUMBIUM

INTRODUCTION

The United States is totally dependent upon foreign sources for all of its columbium (niobium) raw material needs. Limited columbium resources in the United States would require significantly higher market prices before the deposits could profitably be brought into production.

Columbium is of strategic importance to U.S. interest because of its defense related uses in the aerospace, energy, and transportation industries. Columbium's superior performance as an alloying element has led to its increased use in specialty steels. Low concentrations of columbium (less than 1 pct) strengthen steel through grain refinement and precipitation mechanisms (*1*).[1] This strengthening effect has led to widespread and growing use for columbium in high-strength, low-alloy (HSLA) steels. HSLA steels are used in the construction of massive structures and in automobile construction to effect weight reduction for greater fuel economy.

Columbium is also used as an alloying element in stainless steels as a carbide stabilizer to improve the steel's resistance to corrosion. These steels are well suited for use in exhaust manifolds, fire walls, and pressure vessels.

High-purity columbium oxides (99 pct Cb_2O_5), which can be further processed to high-purity ferrocolumbium and nickel-columbium, are gaining increasingly important applications as alloying elements in superalloys. Nickel-, cobalt-, and iron-based superalloys, containing between 1.0 to 5.0 pct Cb, demonstrate superior performance in the high-temperature sections of military jet engines and other turbine engines such as those used in the U.S. Army's XM-1 battle tank (*2*).

New and promising markets are being developed for columbium in the field of superconductors. Columbium has unrivaled properties of superconductivity that have led to its application as the principal alloying element with titanium (CbTi) or tin (Cb_3Sn) in the production of superconductors. Superconducting magnets are used for high-energy physics research as accelerator magnets (atom smashers), magnetic confinement magnets for thermonuclear fusion, and in health care for magnetic resonance imaging equipment that produces cross-sectional images of the human body (*3*). Other potential high-volume uses include magnetohydrodynamics, energy storage, levitated transportation, motors and generators, superconduction transmission lines, and superconducting computers. Although current demand for superconducting materials is relatively low, continued research and development has increased the market applications for these materials and may develop a significant market for columbium alloy superconductors in the future.

Columbium occurs primarily in two mineral forms, pyrochlore $(Na, Ca)_2 Cb_2(O,OH,F)_7$ or columbite $(Fe, Mn)(Cb, Ta)_2O_6$. Mill concentrates produced from either mineral form contain approximately 60 pct columbium pentoxide (Cb_2O_5), equivalent to 42 pct Cb.

Following beneficiation, pyrochlore concentrates are processed to ferrocolumbium for use as an alloying agent in steelmaking. Steelmaking-grade ferrocolumbium can be further refined to high-purity columbium oxide for production of high-purity ferrocolumbium, nickel columbium, and columbium metal. These products are then used in the production of superalloys and other products having high-purity columbium requirements. Until recently, columbite concentrates were required for production of the more highly refined columbium end products. Advances in technology have permitted the further processing of pyrochlore concentrates to high-purity columbium oxides and high-purity ferrocolumbium, nickel-columbium, and columbium metal products, largely replacing the market importance of columbite concentrates.

Pyrochlore ores compose well over 90 pct of the known market economy country (MEC) in situ demonstrated columbium resources. Columbite ores, which at one time were the major source of columbium, presently supply only a small percentage of MEC columbium needs and are not evaluated in this study.

Greater than 80 pct of columbium concentrate production is processed to ferrocolumbium for use in steelmaking. This study evaluates the cost of mining, beneficiation, and smelting pyrochlore ores through the production of steelmaking-grade ferrocolumbium. Ferrocolumbium production costs are at the smelter site and include the cost of transporting the pyrochlore concentrates from the mill to the smelter. Additional information is available from a recently published Bureau report (*4*).

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

GEOLOGY AND RESOURCES

GEOLOGY

Columbium minerals occur in nature as oxides, multiple oxides, and hydroxides (*5*). Principal ore minerals include

Pyrochlore $(Na,Ca)_2Cb_2(O,OH,F)_7$
Pandaite (bariopyrochlore) $(Ba,Sr)(Cb,Ta)(O,OH)_7$
Columbite-tantalite $(Fe,Mn)(Cb,Ta)_2O_6$
Loparite $(Ce,Na,Ca)_2\ (Ti,Cb)_2O_6$
Ixiolite $(Ta,Nb,Sn,Fe,Mn)_4O_8$

Major economic concentrations of columbium occur predominantly as pyrochlore minerals in the carbonatite-related member of intrusive alkaline rock complexes.

Intrusive alkaline complexes commonly occur as small circular or elliptical bodies (usually less than 5 miles in diameter), arcuate ringlike structures, cone sheets, or crosscutting dikes. Carbonatites usually occur central to the alkaline complex as a plug or irregularly shaped body up to 3 mi^2 in area (*6*). Pyrochlore characteristically reaches peak abundance in the calcium-rich member of the carbonatite series, called a sovite, although it has been reported in practically all associated rock types of the alkaline granitic suite (*7*). In addition to columbium, carbonatites may contain significant concentrations of phosphate, titanium, rare earths, uranium, thorium, and fluorite ore minerals.

Deep in situ weathering of carbonatites in the tropical regions of Brazil and Africa has resulted in strong residual enrichment (on the order of 2 to 10 times) of the more resistant columbium, phosphate, and iron minerals (*8*). Under these conditions, columbium may occur in an altered mineral form of pyrochlore called pandaite (more correctly known as bariopyrochlore), in which the sodium and calcium ions are replaced in varying percentages by barium and/or strontium. The highest concentration of columbium ore is in the eluvial or laterized material that forms a cap overlying the carbonatite body.

In the more temperate climate of Canada and the United States, weathering has not played a significant role in the condition of the columbium ore deposit and columbium concentrations range from 0.5 to 0.7 pct.

RESOURCES

Demonstrated resources of the columbium deposits in Brazil, Canada, the United States, and the four African nations of Zaire, Uganda, Tanzania, and Kenya included in this study are estimated to contain 5.35 million mt of columbium. Total recoverable columbium as contained in ferrocolumbium product is presented in table 1. Resources evaluated at the inferred level from studied deposits contribute an additional 6.41 million mt of contained columbium.

Figure 1 shows total demonstrated resources of contained columbium evaluated in this study and the reserve base tonnage. Several nonproducing Canadian and African properties have been excluded from the reserve base owing to the high cost involved in developing the operations.

Producing mines in MEC's contain a total of 3.28 million mt of contained columbium from the deposits evaluated for this study. The Brazilian Araxa Mine alone contains 2.96 million mt demonstrated resources. Of all mines and nonproducing pyrochlore deposits evaluated, those in Brazil account for 60 pct of the contained columbium resources at the demonstrated level (fig. 1). To add to Brazil's dominance of world columbium resources, recent reports have claimed the discovery of a large columbium deposit, in the Amazon region, at Seis Lagos (*9-10*).

Canada contains the MEC's second largest demonstrated resource of contained columbium. The Niobec Mine is Canada's sole producer of columbium concentrate and the MEC's only source of pyrochlore concentrates. Brazil largely discontinued exporting pyrochlore concentrates in 1981 and currently exports only upgraded forms of columbium products. Several Canadian deposits (Martison Lake, Strange Lake, and Thor Lake) may produce columbium as a byproduct, depending on the market conditions for the other potential commodities contained within these deposits (*11-12*).

Table 2 lists the columbium properties included in this study.

Table 1.—Summary of MEC demonstrated columbium resources evaluated for this study, as of January 1985

Region and country	In situ, ore tonnage, 10^6 mt	In situ grade, wt pct Cb	Contained Cb, 10^3 mt	Recoverable Cb in ferrocolumbium, 10^3 mt
South America:				
Brazil	207	1.57	3,240	2,380
North America:				
Canada	439	.29	1,290	710
United States	33	.22	70	40
Total or av	472	.29	1,360	750
Africa:				
Zaire	5	.20	100	60
Uganda	186	.16	290	160
Tanzania	23	.40	90	60
Kenya	54	.50	270	50
Total or av	268	.28	750	330
Grand total or av	947	.57	5,350	3,460

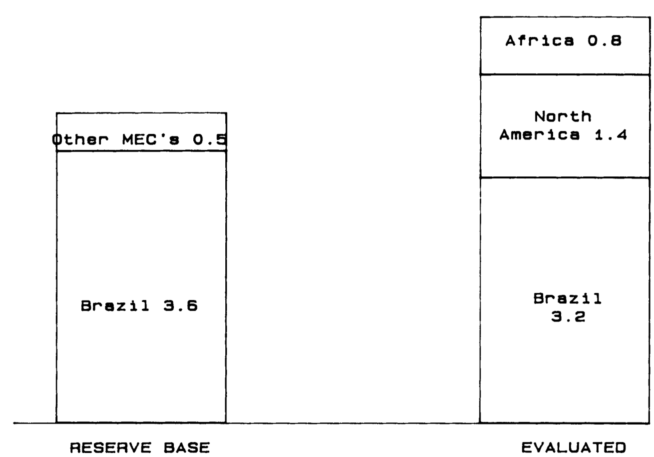

Figure 1.—Estimates of world columbium resources (million metric tons of contained columbium). Several nonproducing Canadian and African properties have been excluded from the reserve base.

Table 2.—MEC columbium properties included in this study

Country and deposit name	Ownership	Current status[1]	Mining method[2]	Milling method[3]	Feed grade, wt pct	Ore milling capacity 10^3 mt/yr	Production grade, wt pct	Production capacity, 10^3 mt/yr
United States:								
Gem Park Complex	Coca Mines, Inc.	NP	OP	F	0.21	30	60	1,066
Iron Hill	Buttes Gas and Oil	NP	OP	F	.24	990	60	1,152
Brazil:								
Araxa	CBMM	P	OP	F	1.75	1,272	66	16,910
Catalao	Mineracao Catalao de Goias S.A.	P	OP	F	.79	472	65	18,348
Catalao Ouvidor	Goias Niobio S.A.	D	OP	F	.65	230	65	708
Canada:								
Crevier	Soquem	NP	OP	G, F	.12	1,040	60	800
James Bay	Esso, Morrison, Canray, Argor	NP	OP	F	.36	577	60	1,460
Lackner Lake	Mertec Resources Develop Ltd	NP	OP	F	.16	720	60	770
Martison Lake	Camchib Mines, New Ventures Eq	NP	OP	F	.24	1,300	60	2,213
Nemegosenda	Gulf Agric Chem. Co. Ltd.	NP	OP	L	.30	520	60	1,334
Niobec	Teck, Soquem	P	UG	F	.47	725	60	2,221
Oka	St. Lawrence CB, Metals Corp	NP	OP	F	.24	1,520	60	2,620
Strange Lake	Iron Ore Co. of Canada	NP	OP	F	.50	520	60	1,556
Thor Lake	Highwood Resc, Calabros Ltd	NP	OP	F	.25	702	60	1,174
Kenya: Mrima Hill	Mrima Industrial Minerals	NP	OP	F	.50	416	60	395
Tanzania: Panda Hill	Tanzania State Mining Corp	NP	OP	F	.40	520	60	1,383
Uganda: Sukulu Hills	Sukulu Mines Limited	NP	OP	F	.15	416	60	370
Zaire:								
Bingo	Somikivu	NP	OP	F	2.40	150	60	2,052
Lueshe	..do	D	OP, UG	F	1.52	150	60	1,428

[1] P, producing; D, developing; NP, nonproducing.
[2] OP, open pit; UG; underground.
[3] F, flotation; G, gravity; L, acid leach.

U.S. AND WORLD HISTORICAL PRODUCTION

Figure 2 shows the history of world columbium concentrate production, in 5-yr intervals, from 1950 through 1985. Up to 1965, columbite concentrates from Nigeria supplied the major share of the MEC's columbium needs. In 1966, Brazil and Canada passed Nigeria as the world's largest producer of columbium concentrates. The graph also demonstrates the increase in total columbium concentrate production from 2.5 million lb in 1950 to 78 million lb in 1980. Although not shown in figure 2, because of the 5-yr interval between data points, columbium concentrate production from 1983 through 1985 showed an increase from 45.4 million lb to 69 million lb (*13*).

Brazil is the world's largest producer and processor of columbium concentrates. In 1981, Brazil all but discontinued exporting pyrochlore concentrates. Currently Brazil only exports upgraded columbium products, primarily steelmaking-grade ferrocolumbium and very minor amounts of columbium concentrates. Araxa, the largest columbium mine in the MEC's, has begun the production of high-purity columbium oxides and high-purity ferrocolumbium and nickel-columbium. Recently, Araxa announced plans to begin construction of facilities to produce 100 mt/yr of columbium metal. Production is scheduled to begin in midyear 1988 and will likely target the growing superconductor market (*14*).

Canada's Niobec Mine is the MEC's only source of pyrochlore concentrates. The Niobec Mine supplies pyrochlore concentrates to ferrocolumbium smelters in Europe, Japan, and the United States.

Another potential source of pyrochlore concentrates is the Lueshe deposit in Zaire. Development of this deposit is continuing, following the construction and successful testing of a 2-mt/h pilot plant in March 1984. Commercial production is expected to begin in the near future (*15*).

The U.S. columbium processing industry comprises seven companies with eight plants integrated through the processing of columbium concentrates (either pyrochlore

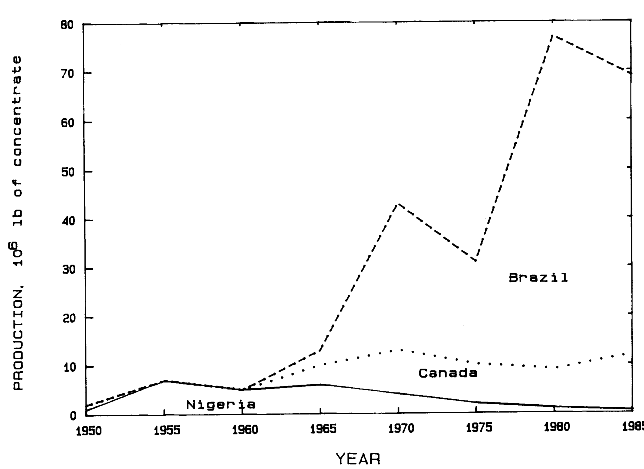

Figure 2.—World columbium concentrate mine production, 1950-85.

concentrates or columbite concentrates) to upgraded end products, including columbium metal (*16*). Two firms, KBI Div. of Cabot Corp. and Fansteel Inc., are integrated from concentrate through to the processing of columbium end products including columbium metal. Currently, the United States is the MEC's largest producer of columbium metal.

The U.S. columbium market accounted for approximately 25 pct of all MEC columbium consumption (*17*). Overall U.S. consumption of columbium decreased slightly from 7.7 million lb in 1984 to 7.6 million lb in 1985 (*16*). However, U.S. imports for consumption of ferrocolumbium from Brazil was 7.2 million lb in 1985 compared with 6.7 million lb in 1984. Imports for consumption of columbium oxides from Canada also increased in 1985 to 2.8 million lb compared with 2.5 million lb in 1984 (*13*).

It is possible that the United States may become totally dependent on foreign sources for all of its columbium product needs. Continued vertical integration of foreign columbium products through the production of upgraded columbium products could place the U.S. columbium industry at a distinct economic disadvantage.

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

Open-pit mining is employed or proposed for all but 2 of the 19 columbium properties. Of the three producing properties, Niobec is the only underground operation. Underground mining is projected for the Lueshe deposit following the sixth year of surface mining.

Brazilian pyrochlore deposits are very amenable to open-pit mining because of the highly weathered nature of the ore. Blasting is not required and mining is accomplished using crawler tractors. Front-end loaders are used for loading the ore onto either a conveyor system, such as at Araxa, or into trucks, such as at Catalao, to facilitate haulage to the mill. Mine recoveries exceed 90 pct.

Significant drilling and blasting have been proposed for all the Canadian deposits because of the hardness of the ore bodies. African deposits, although weathered, will require blasting to fracture the more resistant zones, which are estimated to occupy 10 to 20 pct of each ore body.

Underground mining at Niobec, Canada, is accomplished using a large-diameter blasthole stoping method that extends to a depth of 400 m. Primary crushing is carried out underground and trackless mining equipment is used for the extraction of the ore from the large vertical stopes.

At the Lueshe deposit in Zaire, open-pit mining would initially be used to extract pyrochlore ore from the four hills that constitute the surface expression of the carbonatite body. Mining of the ore from the lower portion of the carbonatite would be carried out using underground sublevel blasthole stoping. It was assumed that underground development work would be initiated in the sixth year of open-pit operation.

PROCESSING

Beneficiation

Beneficiation of pyrochlore ore is carried out through crushing and grinding, magnetic separation, desliming, flotation, leaching, and calcining. Mill recoveries for producing mines range from 50 to 80 pct. Anticipated mill recoveries for nonproducers range from a low of 20 pct up to 80 pct. Ranges in recoveries are based on variations in feed grade, physical characteristics of the ore, and mill recovery techniques.

Depending on the nature of the ore, one to three stages of crushing are used. Laterized ore, such as Araxa's, requires less crushing than the hard-rock ores similar to Niobec's. Crushing circuits consist of various vibrating screens that feed jaw and impact crushers in open or closed circuits. Ball and rod mills are used in the grinding circuit, usually in one or two stages to further reduce the size of the ore.

Magnetic separation may not be carried out at every operation and may serve different purposes when used. Deposits with high magnetite concentrations, such as the Brazilian ores, use magnetic separation to remove the magnetite, which can be as much as 25 pct of the ore. Magnetic separation may also be used to concentrate the slightly magnetic pyrochlore minerals.

In pyrochlore beneficiation, the desliming circuit is considered the most critical stage, as the presence of slimes may inhibit the recovery of columbium as a pyrochlore flotation concentrate. Therefore, a three-stage desliming circuit is used, consisting of various size classification and desliming cyclones along with scrubbers.

Commonly, both coarse and slime products are discharged from the desliming circuit, and a separate circuit is used for the flotation of fine particles from the desliming cyclones. Each flotation section consists of a single rougher stage followed by three to five stages of cleaners. Final pyrochlore concentrates from both sections are combined and sent to thickening and filtration prior to calcining and leaching.

A calcining and leaching stage may be necessary to reduce high concentrations of phosphate to meet market specifications. Such is the situation at the Brazilian mines where two stages of leaching are used, consisting of HCl leach and NaOH leach.

Smelting

Aluminothermy is a batch process used for producing ferrocolumbium from pyrochlore concentrates. A typical reaction charge at CBMM's Araxa ferrocolumbium operation as reported by de Souza Paraiso and de Fuccio (*8*) is as follows, in kilograms:

Pyrochlore concentrate (60 pct Cb_2O_5)	18,000
Iron oxide (as hematite) (68 pct Fe)	4,000
Aluminum powder	6,000
Fluorspar	750
Lime	500

The charge is thoroughly mixed and poured into a magnesite-brick-lined steel cylinder. This cylinder, or reactor vessel, is placed into a silica sand bed that is lined with a mixture of lime and fluorspar. A fuse mixture of aluminum powder and barium or sodium peroxide, ignited by a flame or small quantity of water, is used to start the exothermic reduction. Reaction time lasts about 15 to 20 min and reaches a maximum temperature of 2,400° C. During this reaction, gangue material, along with other impurities forming the slag, separate from the columbium-iron alloy.

When the reaction is complete and slag is drained off the top, the reaction cylinder is lifted leaving the ferrocolumbium "button" in the sand pit. After several hours for cooling and solidification, the ferrocolumbium is removed from the pit, crushed, sieved, sized (based on market specifications), and packaged for shipping. A typical charge, as previously described, will produce approximately 11 mt of ferrocolumbium containing 66 pct Cb (*8*). A similar grade of ferrocolumbium is achieved at the Catalao smelter and is expected for the developing Catalao Ouvidor operation.

A ferrocolumbium smelter model was developed for the Niobec Mine and subsequently applied to all nonproducing deposits (*18*). The model is very similar to the aluminothermic process utilized by the Brazilian operation except that it assumes a ferrocolumbium product containing 60 pct Cb.

PRODUCTION COSTS

Production costs for producers cannot be reported in order to avoid disclosing individual deposit information. As shown on table 3, total production costs for nonproducers were estimated to be $13.48/lb of Cb contained in ferrocolumbium for North American deposits and $8/lb for African deposits (this total also includes a 15-pct discounted-cash-flow rate of return [DCFROR]). Figure 3 shows a breakdown of each individual component as a percentage of the total production cost.

For Brazilian producers, operating costs are greatest in the milling and smelting area. Mining costs are substantially lower because the laterized Brazilian ores do not require drilling and blasting. Conversely, estimated mining costs for nonproducers in Canada and Africa are much higher because of the additional cost for drilling and blasting. Mining costs for underground operations compose the major share of the total production cost.

Milling costs also reflect the original condition of the ore. The more laterized ores require less crushing, possess a higher grade of columbium, and contain fewer impurities (removed by weathering) than hard-rock ores. This is demonstrated in a comparison of estimated nonproducer milling costs (on a per pound of columbium contained in ferrocolumbium product basis) of $1.61 for the weathered African ores with $3.17 for the relatively unweathered North American ores.

Taxes and royalties are generally greater for nonproducers because of the increased revenues required to cover the higher overall costs (including a 15-pct DCFROR) and the costs of recovering undepreciated capital investments over the life of the property. In other words, nonproducers would require a higher taxable income (leading to higher tax payments) to cover all operating costs and provide a 15-pct DCFROR on all investments.

Transportation costs for nonproducers in North America and Africa reflect the cost of transporting pyrochlore concentrates from the mill site to existing ferrocolumbium smelters in the United States, Europe, or Japan. Brazilian producers have a smelter on site and therefore do not incur this transportation expense.

Table 3.— Ferrocolumbium production costs for nonproducing[1] operations in selected MEC's

(All costs are in January 1985 U.S. dollars per pound of ferrocolumbium on a weighted-average basis)

	Operating costs			Transportation to smelter[2]	Net operating cost	Recovery of capital[3]	0-pct DCFROR		15-pct DCFROR		
	Mine	Mill	Smelter				Taxes and royalties[4]	Total cost[5]	Taxes and royalties[6]	Return on investment	Total cost[7]
Surface operation:											
North America . . .	2.45	3.17	0.95	0.19	2.67	2.67	0.25	8.03	1.01	3.04	13.48
Africa[8]	1.18	1.61	.76	.18	3.73	.62	.13	4.48	1.67	1.98	8.00

[1] Cost for producing operations are withheld to avoid disclosing individual deposit data.
[2] Assumed as product destination point.
[3] Includes cost of recovering remaining undepreciated investments and reinvestments required over the life of the operation.
[4] Includes property, State, Federal, and severance taxes and royalties, calculated at a 0-pct DCFROR.
[5] Includes recovery of all costs of production including capital but does not include profit.
[6] Includes property, State, Federal, and severance taxes and royalties, calculated at a 15-pct DCFROR.
[7] Includes recovery of all costs of production including capital and return on investment at a 15-pct DCFROR.
[8] Includes Lueshe, Zaire, as a surface operation.

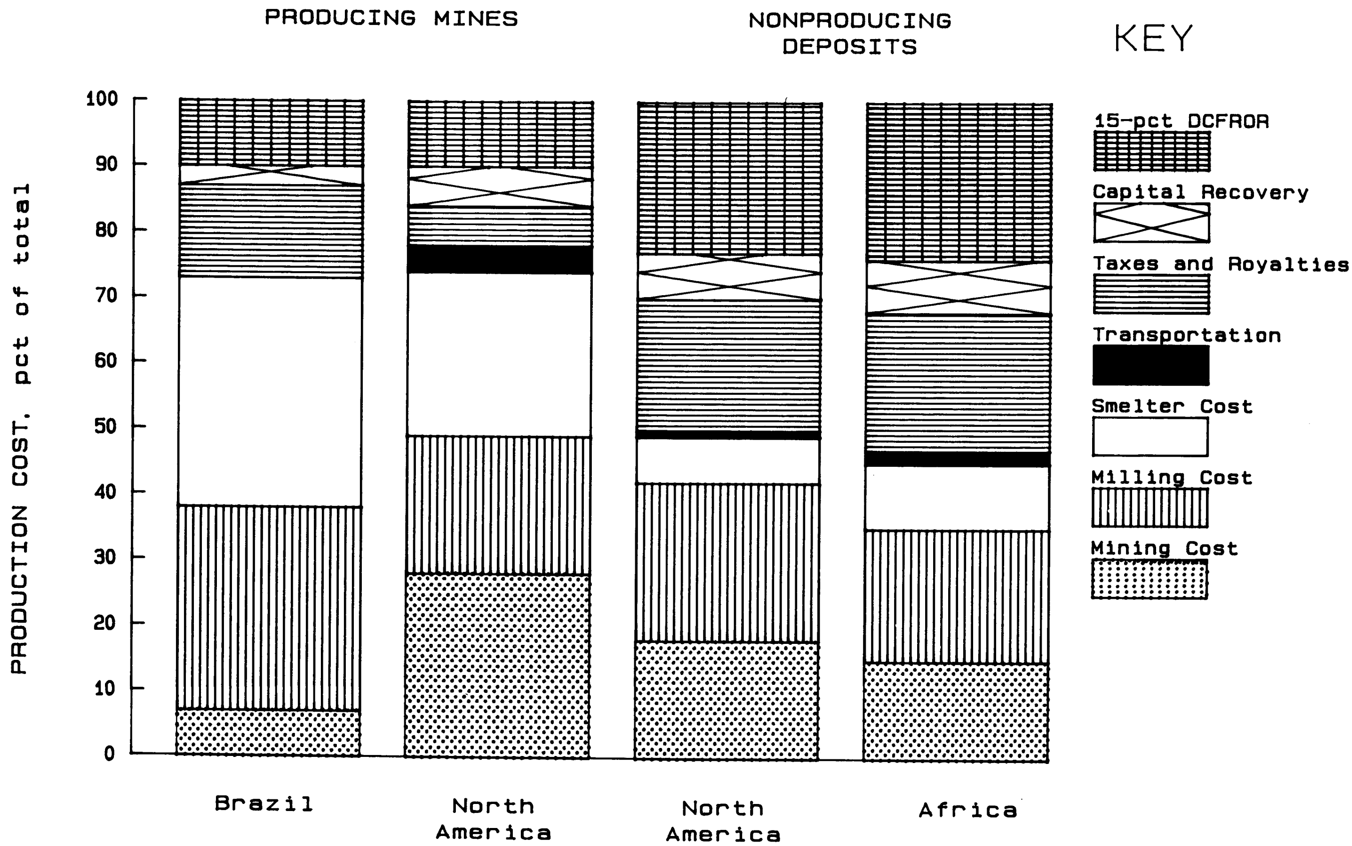

Figure 3.—Ferrocolumbium production costs for selected MEC's.

AVAILABILITY

TOTAL RECOVERABLE

Nineteen columbium deposits located in Brazil, Canada, the United States, Zaire, Uganda, Tanzania, and Kenya contain a total of 5.35 million mt of columbium contained in demonstrated resources. Using an average recovery rate of 65 pct, a total of 3.47 million mt of columbium is recoverable. Assuming all columbium is processed into ferrocolumbium, at an average grade of 64 pct Cb contained in ferrocolumbium, a total of 5.42 million mt of ferrocolumbium is potentially available from the deposits examined in this study.

Figure 4 is a graphic representation of the availability of MEC potentially recoverable columbium, as contained in ferrocolumbium, related to total costs of production including a 15-pct DCFROR. Owing to the limited number of producing mines, a separate curve is not shown for producers. The combined producer, nonproducer curve begins at a total cost of $5/lb to insure the confidentiality of data for the producing operations with production costs below $5/lb. At a cost of production comparable with the January 1985 market price of $5.66/lb for columbium contained in ferrocolumbium, 2.67 million mt of columbium is potentially available. This amounts to 77 pct of the potentially recoverable demonstrated resource.

Producing deposits contain 2.42 million mt of recoverable resources, or 70 pct of MEC total, nearly all of which is from Brazilian mines. Eleven of the nineteen MEC deposits evaluated are located in North America and account for 22 pct or 0.75 million mt of the MEC's total recoverable columbium resources. Combined with resources of Brazil, Western Hemisphere countries contain a total of 3.13 million mt of recoverable columbium, which is 91 pct of studied MEC resources.

Recoverable resources from all MEC nonproducers total 1.05 million mt of columbium. If the United States had to rely solely on columbium deposits located in North America, a total of 0.69 million mt of recoverable columbium would be available from nonproducers, in addition to that currently available from Niobec. However, a market price of greater than double the current $5.66/mt would be required to bring half of the North American recoverable columbium resources into production.

In summary, producing columbium properties have sufficient resources to supply MEC columbium needs for the foreseeable future. As a result, it is very unlikely that many nonproducing deposits will be developed in the near future.

ANNUAL CAPACITY

Current producing mines have the capacity to produce over 20,000 mt/yr of recoverable columbium contained in ferrocolumbium products (at a total cost of less than $5.66/lb), well into the next century. Estimates of actual production for 1985 amounted to 13,000 mt of columbium contained in ferrocolumbium from the three producing operations *(15)*, indicating that they have the capability to meet substantial increases in MEC columbium needs.

Cumulative world demand for columbium from the years 1983 to 2000 was estimated to be 0.39 million mt *(1)* based on an annual growth rate of 5.1 pct. From a comparison of potentially recoverable columbium resources from the producers to the estimated cumulative demand, it appears that adequate supplies of columbium from currently producing mines will be available well beyond the year 2000.

Significantly higher prices than current levels must prevail in order to cover the high costs of production associated with bringing most nonproducing operations into production. In view of the excess capacity available from producing mines, it is unlikely many nonproducing deposits will come on line in the near future.

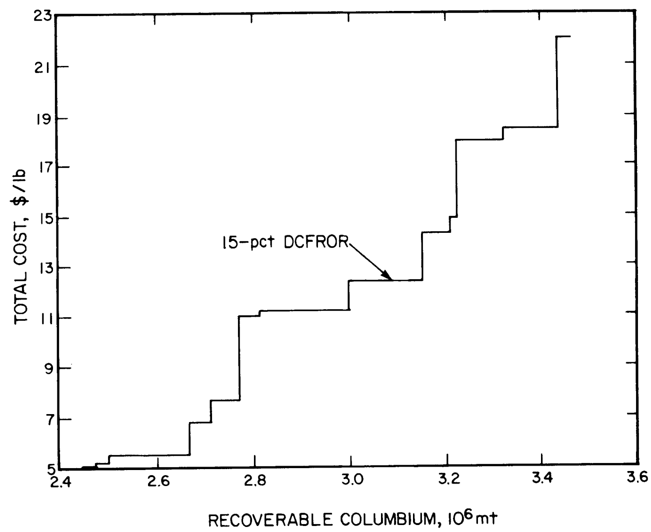

Figure 4.—Potential total MEC columbium contained in ferrocolumbium (January 1985 U.S. dollars).

CONCLUSIONS

Columbium, in the form of ferrocolumbium, is used primarily as an alloying element in high-strength, low-alloy steels and superalloys. Approximately 85 to 90 pct of the columbium consumed is in the form of ferrocolumbium for steelmaking. Other uses for columbium include further processing of ferrocolumbium to high-purity columbium products for use in cobalt-, nickel-, and iron-based superalloys and in production of columbium-titanium and columbium-tin alloys and compounds, for use in superconducting materials.

A total of 19 deposits (3 producing mines and 16 nonproducers) were examined to determine MEC availability of columbium. Total columbium contained in the demonstrated resources of these deposits amounts to approximately 5.35 million mt. This includes 3.24 million mt in Brazil, 1.36 million mt in North America (Canada and the United States), and 0.75 million mt in Africa (Kenya, Uganda, Tanzania, and Zaire). In all cases, columbium occurs in the mineral form of pyrochlore or its barium-strontium analog, pandaite, and is found in association with the carbonatite member and related members of alkalic granite complexes.

In terms of potential total availability, Brazil is the largest and lowest cost source of recoverable columbium in the MEC's. Total recoverable columbium (contained as 65 to 66 pct in ferrocolumbium) in Brazil amounts to 2.38 million mt (69 pct) of the MEC total. Recoverable columbium resources in North America (Canada and the United States) account for over 22 pct of the total MEC resources studied, or 0.75 million mt.

Current producers of columbium examined for this study include the Araxa and Catalao Mines in Brazil and the Niobec Mine in Canada. Combined, these mines contain 2.42 million mt of recoverable columbium resources that are more than sufficent to supply the estimated cumulative MEC columbium demand through the year 2000 (0.42 million mt), based on a 5.1-pct annual growth rate. Additionally, producing mines currently demonstrate excess annual capacity by operating at an average 65 pct of their total combined annual capacity in 1985.

Potentially recoverable columbium resources from MEC nonproducers total 1.05 million mt. Of this, only 8 pct or 84,000 mt is available at a cost of production comparable with the January 1985 market price of $5.66/lb, assuming a 15-pct DCFROR.

The United States is highly dependent on foreign sources of columbium, principally in the form of ferrocolumbium for steelmaking from Brazil and pyrochlore concentrates from Canada. Furthermore, Brazil has moved towards increasing its production of ferrocolumbium and other high-grade columbium products in recent years, resulting in a decrease in U.S. production. A continuation of this trend could result in increased competition for low-cost upgraded columbium products for the U.S. columbium industry. This could very likely lead to a total U.S. dependence on foreign sources for all its columbium product needs.

REFERENCES

1. Cunningham, L.D. Columbium. Ch. in Mineral Facts and Problems, 1985 Edition. BuMines B 671, pp. 185-196.

2. Manker, E.A. Columbium—An Outlook. CIM Bull., v. 74, No. 832, Aug. 1981, pp. 93-99.

3. Gregory, E. Applications and Fabrication Processes of Superconducting Composite Materials. J. Met., v. 36, No. 6, June 1984, pp. 30-34.

4. Miller, F.W., R.J. Fantel, and D.A. Buckingham. Columbium Availability—Market Economy Countries. A Minerals Availability Appraisal. BuMines IC 9085, 1986, 20 pp.

5. Parker, R.L., and J.W. Adams. Niobium (Columbium) and Tantalum. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 443-454.

6. Hyndman, D.W. Petrology of Igneous and Metamorphic Rocks. McGraw-Hill, 1972, pp. 207-222.

7. Dawson, K.R. Niobium (Columbium) and Tantalum in Canada. Geol. Surv. Canada, Econ. Geol. Rep. 29, 1974, pp.19-33.

8. de Souza Paraiso, O., and R. deFuccio, Jr. Araxa Niobium Mine. Min. Mag., Feb. 1982, pp. 134-147.

9. Mining Magazine. Brazilian Niobium Find. July 1984, p. 11.

10. Harrop, M.D. Columbium: Recovering Steel Outpost Sparks U.S. Offtake 21.6%. Eng. and Min. J., v. 186, No. 3, Mar. 1985, pp. 100-102.

11. Mining Journal. Highwood's Rare Metal Bonanza. Mar. 15, 1985, pp. 171.

12. Green Market News. Sherritt Studying Plans To Open Two Rock Properties in Ontario. V. 8, No. 28, July 16, 1984, p. 1.

13. Cunningham, L.D. Columbium and Tantalum. Ch. in BuMines Mineral Yearbook 1985, v. 1, pp. 305-316.

14. Mining Journal (London). CBMM to Produce Pure Niobium. V. 304, No. 7816, June 7, 1985, p. 407.

15. ______. Zaire Reaps Some Rewards. V. 305, No. 7822, July 19, 1985, pp. 37-39.

16. Cunningham, L.D. Columbium. Sec. in BuMines Mineral Commodity Summaries 1987, pp. 40-41.

17. Tron, A. J. Niobium and Tantalum. Sec. in Mining Annual Review—1985. Steel Industry Metals, 1985, pp. 96-97.

18. Pincock, Allen & Holt, Inc. Development of Engineering and Cost Data for Foreign Columbium, Tantalum, Mercury, and Molybdenum Properties (contract J0225022). BuMines OFR 84-85, 1985, 20 pp.; NTIS PB 85-215325.

COPPER

INTRODUCTION

Copper has been known to humankind for at least 6,000 yr and has been used perhaps longer than any other metal except gold. Several modern-day countries base a major portion of their local economy and foreign exchange on copper production and marketing.

The industrial applications of copper result from a combination of its many useful physical properties: high electrical and thermal conductivity, good resistance to corrosion, good ductility and malleability, high strength, lack of magnetism, and decorative color.

The end-use distribution of copper and copper-alloy mill products domestically in 1985 was estimated, according to the Copper Development Association Inc. (CDA), to be 40 pct in building and construction, 23 pct in electrical and electronic products, 14 pct in industrial machinery and equipment, 12 pct in transportation equipment, and 11 pct in consumer and general products *(1)*.[1] If the electrical components are extracted from the end-use categories described, electrical applications will account for approximately 60 pct of copper demand *(2)*.

This study investigates the availability of refined copper metal from selected mineral properties in market economy countries (MEC's). Transportation costs to the smelter and refinery are included. The nickel-copper resource of the Sudbury District, Canada, is not included in the study owing to nickel being the primary commodity at these properties. Operations that were nearing the depletion of resources were not included. The mines in Yugoslavia and centrally planned economy countries (CPEC's) were excluded because of a lack of reliable data to update the evaluations.

This chapter is an updated summary of Bureau of Mines Information Circular 8930 "Copper Availability—Market Economy Countries. A Minerals Availability Program Appraisal" *(3)*.

GEOLOGY AND RESOURCES

GEOLOGY

Copper occurs in at least 160 minerals in nature *(4)*. Chalcopyrite is the most abundant and economically important copper mineral. Bornite and chalcocite are major minerals in secondary enriched zones in most weathered copper deposits. Malachite is the major oxide mineral, with cuprite, azurite, and chrysocolla contributing significant copper in some deposits. The bright color associated with these oxide minerals is an important exploration guide.

Native copper is important in some deposits as an accessory mineral. The Michigan Native Copper District, not evaluated as part of this study because of its inferred resource classification, represents a major resource of copper, primarily as native copper.

Copper deposits are categorized into three major types, porphyry, sedimentary, and massive sulfide. A few deposits fall into an "other" category, which includes magmatic differentiation, copper-nickel mafic intrusives, native copper, and undetermined.

Porphyry copper deposits as defined by Cox *(4)* are deposits of disseminated copper sulfide that are in or near a felsic intrusive body. The definition is further expanded to include genetically associated vein and replacement copper deposits.

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

The Andean copper belt, extending from Panama to Chile and Argentina in the south, contains the majority of the world's known copper resource in the form of porphyry deposits. The American southwest porphyry copper district (Arizona and New Mexico in the United States and northern Mexico) constitutes the second largest concentration of world copper.

Lowell *(5)* concluded that a typical porphyry deposit measures 3,500 by 6,000 ft in plan and contains 135 million mt of material averaging 0.8 pct Cu and 0.015 pct Mo. Sulfide minerals present in this typical porphyry deposit in descending order of abundance are pyrite, chalcopyrite, molybdenite, and bornite. Where weathering and leaching have taken place in the upper portions of the deposits, a supergene-enriched zone may form and overly the primary mineralization. The principal byproduct metals recovered from porphyry deposits are molybdenum, gold, and silver.

Genetically related vein and replacement deposits are formed when metal-rich solutions, escaping from a crystallizing intrusion, deposit minerals in faults or fractures. These deposits are commonly tabular in shape and small in comparison with porphyry deposits, with an average copper content of 320,000 mt versus over a million metric tons for porphyry deposits *(4)*.

Major copper resources occur in strata-bound deposits in sedimentary rock. The Zaire and Zambia copper belt in Africa, the Kupferschiefer in Europe, and enormous deposits in the U.S.S.R. are all of sedimentary origin and play an important role in copper production. The African copper belt is a sequence of upper Precambrian sedimentary rocks that extends for about 300 miles across parts of Zaire and Zambia. The copper mineralization occurs in shales as chalcocite, bornite, and chalcopyrite with localized supergene enrichment in the form of malachite. Cobalt is the principal byproduct from these copper operations.

In the White Pine District, Michigan, copper occurs in the Nonesuch Shale of Precambrian age. The bed is 1 to 8 m thick and contains anomalous copper in the form of chalcocite over a distance of 250 km.

Massive sulfide deposits occur in volcanic rocks primarily of basalt and andesite composition. These deposits are generally stratiform or lenticular and concordant with the bedding of the surrounding rocks. The lateral extent is small in relation to thickness, and is composed largely of sulfide minerals with small proportions of silicate gangue. The deposits consist mainly of pyrite and/or pyrrhotite and varying amounts of chalcopyrite, sphalerite, and galena *(4)*. The Canadian greenstone belts including Kidd Creek and Geco deposits, "Kuroko" type deposits in Japan, and the Copper Hill deposit in the United States are examples of massive sulfide deposits.

RESOURCES

Demonstrated resources of the MEC's included in this study are presented in table 1. The properties evaluated are listed in table 2. Major changes have taken place in Chile, Australia, and Zambia since the previous study *(3)* to affect the demonstrated resources for these countries. The discovery of La Escondida in the early 1980's, and additional exploration conducted by EXXON on the Los Bronces deposit, has increased Chile's demonstrated resource approximately 24 pct to nearly 120 million mt copper contained in 11.9 billion mt material. Australia's demonstrated resource estimate was increased from 16 million mt to over 38 million mt contained copper principally as a result of further work performed at Olympic Dam, South Australia. Zambia had a downward revision in the demonstrated resource estimate from 34 million mt to 16 million mt copper. This may be the result of conservative reporting practices by Zambia Consolidated Copper Mines Ltd. in light of the depressed copper market.

Table 1.—Summary of MEC demonstrated copper resources evaluated for this study as of January 1985[1]

(Million metric tons unless otherwise specified)

Country	Number of deposits	In situ resource: Tonnage	In situ resource: In situ grade, pct Cu	In situ resource: Contained copper	Minable resource: Tonnage	Minable resource: Feed grade, pct Cu	Minable resource: Contained copper	Recoverable copper
North America:								
Canada	35	4,195	0.50	20.9	4,069	0.50	20.4	17.6
Mexico	6	4,468	.68	29.1	4,,103	.65	26.5	20.5
United States	62	[2]16,204	[2].53	85.7	[2]15,753	[2].51	80.8	57.4
Total or av[3]	103	24,867	.55	135.7	23,924	.53	127.7	95.5
Central and South America:								
Chile	16	11,969	1.00	119.9	11,877	1.00	118.1	97.0
Peru	12	3,082	.83	25.7	3,022	.82	24.9	21.2
Other	8	3,098	.66	20.6	3,014	.66	19.9	17.3
Total or av[3]	36	18,149	.92	166.2	17,913	.91	162.9	135.5
Europe	12	929	.84	7.8	837	.79	6.6	5.5
Middle East	10	586	1.33	7.8	551	1.30	7.1	6.3
Asia:								
India	6	458	1.28	5.8	439	1.23	5.4	4.7
Philippines	22	3,211	.47	15.0	2,660	.47	12.5	10.4
Other	6	522	.67	3.5	524	.65	3.4	2.9
Total or av[3]	34	4,191	.58	24.3	3,624	.59	21.3	18.0
Africa:								
South Africa, Republic of	5	423	.65	2.7	425	.64	2.7	2.2
Zaire	8	677	4.04	27.3	614	4.09	25.1	20.7
Zambia	9	[4]621	[4]2.58	16.0	[4]635	[4]1.97	12.5	10.2
Other	10	116	1.98	2.3	116	1.87	2.2	1.7
Total or av[3]	32	1,837	2.63	48.4	1,790	2.38	42.5	34.7
Oceania:								
Australia	9	2,261	1.69	38.3	1,746	1.70	29.6	25.5
Papua New Guinea[5]	5	3,639	.47	17.2	3,122	.47	14.7	12.4
Grand total or av[3]	241	56,462	.79	445.7	53,511	.77	412.6	333.4

[1] Includes oxide and leach material.
[2] Includes 4,989 million mt oxide and silicate material at 0.29 pct Cu. Sulfide resources and mill feed averaged 0.65 and 0.62 pct Cu, respectively.
[3] Data may not add to totals shown because of independent rounding.
[4] Includes 135 million mt oxide material at 0.70 pct Cu in Nchanga Stage III leach project. Primary resources and mill feed averaged 3.10 and 2.32 pct Cu, respectively.
[5] Includes Namosi deposit in Fiji.

Table 2.—MEC copper properties included in this study

Country and property name	Ownership	Current status[1]	Mining method[2]	Milling method[3]	Deposit type[4]	Year of initial production[5]	Recoverable copper,[6] 10^3 mt/yr
Argentina:							
Bajo la Alumbrera	Yacimientos Agua del Dionisio	E	S	F	S	NAP	52.4
El Pachon	Cia Minera Aguilar S.A.	E	S	F	S	NAP	97.4
Paramillo Sur	Fabricaciónces Militares	E	S	F	S	NAP	50.8
Australia:							
Benambra	Western Mining, BP Australia	E	U	F	O	NAP	15.0
CSA	Cobar Mines, CRA	P	U	F	O	1907	10.6
Cadia	Pacific Copper Ltd, Homestake Australia	E	S	F	O	NAP	3.7
Chesney	Cobar Mines, CRA	E	U	F	O	NAP	7.2
Golden Grove	North Broken Hill Holdings	E	S	F	M	NAP	10.1
Lady Annie	Traiko Mines, Mount Isa Mines	E	S	L	S	NAP	6.1
Mount Isa	Mount Isa Mines Ltd	P	U	F	M	1931	164.6
Mount Lyell	Mount Lyell Mining & Railway	P	U	F	M	1935	19.5
Olympic Dam	Western Mining, BP Australia	D	U	F	S	1988	53.4
Botswana: Selebi, Phikwe	BCL Ltd	P	U	F	O	1974	9.2
Brazil:							
Camaqua	Cia Brazileira de Cobre	P	U	F	P	1981	11.6
Jaguari (Caraiba)	Caraibas Metais S.A.	P	C	F	P	1982	63.0
Pedra Verde	Promisa, Caraibas Metais S.A.	E	U	F	P	NAP	7.4
Burma: Monywa	Burmese Government	P	S	F	P	1984	15.5
Canada:							
Afton	Teck Corp., Iso Mines Ltd	P	S	F	P	1977	17.0
Bell	Bell Copper (Noranda)	T	S	F	P	1972	22.1
Berg	Kennco Explorations (Western)	E	S	F	P	NAP	55.0
Bethlehem	Bethlehem Copper Ltd	T	S	F	P	1962	23.0
Brenda	Brenda Mines (Noranda)	P	S	F	P	1970	12.6
Casino	Casino Mining Co.	E	S	F	P	NAP	26.4
Catface	Falconbridge, Catface Copper	E	S	F	P	NAP	30.7
Copper and Needle Mountains	Mines Gaspe (Noranda)	T	U	F	P	1955	49.3
Copper Rand	Northgate Exploration Ltd	P	U	F, L	O	1959	10.6
Coppermine River	Coppermine River Ltd	E	S	F	M	NAP	6.5
Detour project	Selco Mining Corp. Ltd	P	C	F	O	1981	14.1
Galore Creek	Stikine Copper Ltd	E	S	F	P	NAP	47.4
Geco	Noranda Mines Ltd	P	U	F	M	1957	18.9
Gibraltar	Gibraltar Mines Ltd	P	S	F, L	P	1972	36.4
Granisle	Noranda Mines Ltd	T	S	F	P	1966	16.9
Great Lakes Nickel	Boliden Canada Ltd	E	U	F	O	NAP	10.3
Heath Steel	Heath Steel, Noranda, ASARCO	P	U	F	M	1957	7.9
High Lake	Kennarctic Explorations	E	U	F	P	NAP	8.7
Highmont	Highmont Mining Corp	T	S	F	P	1981	20.5
Huckleberry Mountain	Kennco Explorations Ltd	E	S	F	P	NAP	13.6
Island Copper	Utah Mines Ltd	P	S	F	P	1971	61.8
Izok Lake	Texasgulf, Inc.	E	S	F	M	NAP	17.9
JA Zone	Bethlehem Copper Ltd	E	U	F	P	NAP	24.8
Kidd Creek	Kidd Creek Mines Ltd	P	U	F	M	1966	129.7
Lornex	Lornex Mining Corp. Ltd	P	S	F	P	1972	91.2
Maggie	Bethlehem Copper Corp	E	S	F	P	NAP	13.7
Poison Mountain	Long Lac, Copper Giant	E	S	F	P	NAP	14.1
Ruttan	Sherritt Gordon Mines Ltd	P	U	F	M	1973	20.5
Sam Goosly	Equity Mining, Placer Development	P	S	F	P	1980	8.6
Schaft Creek	Teck Corp., Liard Copper Mines	E	S	F	P	NAP	29.8
Similkameen	Newmont Mining Corp	P	S	F	P	1925	23.5
Summers Creek	Global Energy Corp	E	S	F		NAP	15.9
Sustut	Falconbridge Ltd	E	S	F	P	NAP	39.7
Thierry	Union Miniere S.A.	T	U	F	O	1976	13.5
Valley Copper	Valley Copper Mines Ltd	P	S	F	P	1983	127.3
Chile:							
Andacollo	Noranda Mines Ltd	E	S	F	P	NAP	90.8
Andina	CODELCO-Chile	P	U	F	P	1970	102.7
Cerro Colorado	Cerro Colorado Mine Development Co.	E	U	F	P	NAP	54.5
Chuquicamata	CODELCO-Chile	P	S	F	P	1915	462.5
El Abra	..do	E	S	F, L	P	NAP	47.4
El Salvador	..do	P	U	F, L	P	1959	106.3
El Soldado	Exxon Minerals Co.	P	U	F	P	1800's	52.7
El Teniente	CODELCO-Chile	P	U	F	P	1906	293.3
La Cascadas	Sociedad Minera Pudahuel (SMP)	P	S	L	P	1970	19.4
La Escondida	Utah International, Getty Oil	E	S	F	P	NAP	210.0
Lo Aguirre	Sociedad Minera Pudahuel (SMP)	P	S	L	P	1980	14.0
Los Bronces	Exxon Minerals Co.	P	S	F	P	1962	36.3
Los Pelambres	Anaconda Co. (ARCO)	E	F	F	P	NAP	102.2
Mantos Blancos	Empresas Sudamericana Cons	P	S	L	P	1961	58.4
Portrerillos	CODELCO-Chile	E	U	F	P	NAP	14.2
Quebrada Blanca	ENAMI, Falconbridge Ltd	E	S	F	P	NAP	99.8
Fiji: Namosi	Viti Copper Ltd.	E	S	F	P	NAP	58.4
Finland:							
Keretti	Outokumpu Oy	P	U	F	M	1957	14.8
Pyhasalmi	..do	P	U	F	M	1962	6.0
India:							
Ambaji	Gujarat Mineral Development	E	U	F	M	NAP	3.4
Khetri, Kolihan	Hindustan Copper Ltd.	P	U	F	O	1973	27.9
Malanjkhand	..do	P	S	F, L	O	1982	23.5
Mosaboni	..do	P	U	F	O	1965	12.0
Rakha	..do	P	U	F	O	1919	2.8
Surda, Pathagora, Kendadi	..do	P	U	F	O	1975	7.4

See explanatory notes at end of table.

Table 2.—MEC copper properties included in this study—Continued

Country and property name	Ownership	Current status[1]	Mining method[2]	Milling method[3]	Deposit type[4]	Year of initial production[5]	Recoverable copper,[6] 10^3 mt/yr
Indonesia: Ertsberg	Freeport Indonesia Inc.	P	U	F	P	1972	70.2
Iran: Sar Cheshmeh	Iranian Government	P	S	F	P	1983	137.5
Israel: Timna	Israel Chem. Ltd. (Government)	P	U	F	S	1959	4.2
Japan:							
Hanaoka	Dowa Mining Co. Ltd.	P	U	F	M	1973	17.1
Kosaka	..do	P	U	F	M	1898	6.6
Jordan: Wadi Dana	Jordanian Government	E	S	L	S	NAP	31.7
Malaysia: Mamut	Overseas Mineral Resource Development.	P	S	F	P	1975	28.7
Mauritania: Akjoujt	Societe Miniere de Mauritania	E	S	F	S	NAP	24.0
Mexico:							
Arroyos Azules	Comision de Fomento Minero	E	U	F	O	NAP	9.1
Cananea	Industrial Minera Mexico	P	S	F, L	P	1963	165.9
El Arco	Industrial Minera Mexico, ASARCO	E	S	F	P	NAP	155.3
La Caridad	Mexicana del Cobre	P	S	F	P	1980	174.0
La Verde	Compania Cuprifera la Verde	E	S	F	P	NAP	27.6
Santo Tomas	Industria Minera Penoles	E	S	F	P	NAP	79.5
Morocco: El Bleida	Societe Minerie de Bou Gaffer	P	U	F	O	1977	15.3
Nambia:							
Klein Aub	Klein Aub Kooermaatskappy	P	U	F	S	1966	4.2
Kombat, Asis West	Tsumeb Corp. Ltd.	P	U	F	O	1965	9.0
Otjihase	Tsumeb Corp. Ltd., Otjihase Ltd.	P	U	F	O	1982	12.6
Tsumeb	Tsumeb Corp. Ltd.	P	U	F	M	1900	7.6
Norway: Tverrejellet	Folldal Verk A/S	P	U	F	M	1968	5.9
Oman: Sohar Project	Oman Mining Company	P	U	F	M	1982	17.8
Pakistan: Saindak	Resource Development Corp. (Government).	E	S	F	P	NAP	12.3
Panama:							
Cerro Colorado	Empresa de Cobre Cerro Colo	E	S	F	P	NAP	172.1
Cerro Petaquilla	Panamanian Government	E	S	F	P	NAP	32.4
Papua New Guinea:							
Bougainville	PNG Government, CRA, Public, Panguna.	P	S	F	P	1972	172.1
Freida River	Conzinc Rio Algom	E	S	F	P	NAP	90.4
OK Tedi	Dampier, My Fugilan, Kupfer	P	S	F	P	1984	92.3
Yandera	Trialco, Buka, Broken Hill	E	S	F	P	NAP	74.2
Peru:							
Antamina	Minero Peru	E	S	F	P	NAP	123.2
Berenguela	..do	E	S	F	S	NAP	7.9
Cerro Verde	..do	P	S	F, L	P	1977	62.2
Cobriza	Centromin	P	U	F	S	1967	38.2
Coroccohuayco	Minero Peru	E	U	F	O	NAP	17.6
Cuajone	Southern Peru Copper Corp	P	S	F	P	1976	126.7
El Aguila	Empressa Minera el Aguila	P	S	F	P	1978	11.2
Michiquillay	Minero Preu, Michiquillay Copper Co.	E	S	F	P	NAP	83.1
Quellaveco	Minero Peru	E	S	F	P	NAP	85.0
Tintaya	Minero Peru, Centromin, Cofide	P	S	F, L	P	1985	60.4
Toquepala	Southern Peru Copper Corp.	P	S	F	P	1960	111.6
Toromocho	Centromin	E	S	F	P	NAP	60.3
Philippines:							
Amacan (North Davao)	North Davo, Private	P	S	F	P	1983	32.5
Basay	Southern Star Mining, Ind. Corp.	T	S	F	P	1979	13.4
Batong-Buhay	Development Bank of Philippines	T	U	F	P	1983	16.0
Biga (Atlas)	Atlas Consolidated Mining & Development	P	S	F	P	1962	39.3
Boneng-Lobo	Western Minolco Corp.	T	S	F	P	1974	23.4
Carmen (Atlas)	Atlas Consolidated Mining & Development	P	S	F	P	1977	65.0
Copper Shield Project	Various claim holders	T	U	F	P	1969	3.2
Dizon	Benguet Consolidated Ltd.	P	S	F	P	1980	21.5
Hinobaan	Negros Copperfield Mines Inc.	D	S	F	P	1986	28.9
Inayawan	Denmag (Philippines) Inc.	E	S	F	P	NAP	16.2
Ino-Capayang	Consolidated Mines Inc.	T	S	F	P	1978	22.5
Lutopan (Atlas)	Atlas Consolidated Mining & Development	P	U	F	P	1966	66.2
Mapula, Masara	Apex Mining Co.	E	S	F	P	NAP	6.9
Sabena	Sabena Mining Co.	T	S	F	P	1979	20.2
San Antonio	Marcopper Mining Corp.	E	S	F	P	NAP	68.7
Santo Nino	Baguio Gold Mining Co.	T	U	F	P	1972	5.6
Santo Tomas	Philex Mining Corp.	P	U	F	P	1957	30.8
Sipalay	Maricalum Mining Corp.	T	S	F	P	1957	51.1
Tapian	Marcopper Mining Corp.	P	S	F, L	P	1969	33.8
Tawi-Tawi	Benguet Consolidated Inc.	E	S	L	P	NAP	5.1
Taysan	..do	E	S	F	P	NAP	22.5
Trident (Sulat)	Trident Mining & Industrial Co.	E	S	F	P	NAP	10.5
Portugal:							
Aljustrel	Empressa Minera Mines	P	U	F	S	1982	9.0
Neves-Corvo	RTZ Metals Ltd. Edma	D	U	F	S	1987	59.0
Saudi Arabia: Jabal Sayid	Saudi Arabian Government	E	U	F	M	NAP	15.6
South Africa, Rep. of:							
Black Mountain	Phelps, Dodge, G.F.S.A.	E	U	F	M	NAP	18.0
Broken Hill	..do	P	U	F	M	1980	5.6
Messina	Messina (Transvaal) Development Ltd.	P	U	F	O	1906	7.1
O'Okiep	O'Okiep Copper Co. Ltd.	P	U	F	O	1965	24.6
Palabora	Palabora Mining Co.	P	S	F	P	1965	122.3
Spain:							
Aznalcollar	Andaluza de Pititas, S.A.	P	S	F	M	1979	13.9
Cerro Colorado	Rio Tinto Patino S.A.	P	U	F	M	1873	33.7
Santiago	..do	P	S	F	M	1975	8.3
Sotiel	Cia Brazileira de Cobre	P	U	F	O	1983	2.6

See explanatory notes at end of table.

Table 2.—MEC copper properties included in this study—Continued

Country and property name	Ownership	Current status[1]	Mining method[2]	Milling method[3]	Deposit type[4]	Year of initial production[5]	Recoverable copper,[6] 10³ mt/yr
Sudan: Hofrat en Nahas	Sudan Government	E	E	F	O	NAP	24.3
Sweden:							
Aitik	Boliden Metall AB	P	S	F	O	1968	38.2
Stekenjokk	..do	P	U	F	M	1975	6.8
Viscaria	Luossavaara-Kirunauaara AB	P	U	F	S	1983	12.0
Turkey:							
Cayeli	Etibank, Phelps Dodge	E	U	F	O	NAP	16.5
Ergani-Madeni	Etibank	P	S	F, L	P	1980	11.0
Espiye	Etibank, KBI	D	C	F	M	1986	7.5
Murgul	Etibank, Black Sea Copper	P	S	F	M	1972	41.8
Siirt	Etibank, Preussag Metall	E	U	F	M	NAP	13.4
Uganda: Kilembe	Ugandan Government	E	U	F	O	NAP	6.9
United States:							
Alaska:							
Arctic Camp	Bear Creek Mining Co. (Kennecott)	E	S	F	P	NAP	109.5
Bornite	..do	E	U	F	P	NAP	22.4
Brady Glacier	Newmont Mining Corp.	E	U	F	P	NAP	12.5
Orange Hill, Bond Creek	Bear Creek Mining Co. (Kennecott)	E	S	F	P	NAP	36.7
Arizona:							
Bagdad	Cyprus Minerals Co.	P	S	F, L	P	1940	52.6
Casa Grande	Casa Grande Copper Co.	E	U	F	P	NAP	88.3
Christmas	Inspiration Resources Corp.	T	C	F	P	1962	6.3
Copper Basin	Phelps Dodge Corp.	E	S	F	P	NAP	25.8
Dubacher Canyon	Occidental Minerals Corp.	E	S	L	P	NAP	8.6
Florence (Conoco)	Continental Oil (Min. Div.)	E	S	F, L	P	NAP	23.6
Helvetia East	Anamax Mining Co.	E	S	F, L	P	NAP	38.4
Helvetia West	Inspiration Resources Corp.	E	S	F	P	NAP	5.5
Inspiration	..do	P	S	F, L	P	1915	71.1
Lakeshore	Noranda Mines Ltd.	P	U	L	P	1976	10.9
Miami East	Newmont Mining Corp.	D	U	F	P	1977	9.7
Miami Leach	Cities Services Co.	T	S	L	P	1954	5.3
Mission, San Xavier	ASARCO	T	S	F	P	1961	35.7
Morenci, Metcalf	Phelps Dodge Corp.	P	S	F, L	P	1942	203.9
New Cornelia	..do	T	S	F, L	P	1917	46.3
Oracle Ridge	Continental Copper Co. (subsidiary of Union Oil).	E	U	F	P	NAP	10.2
Ox Hide	Inspiration Resources Corp.	T	S	L	P	1968	4.4
Palo Verde	Anamax	T	S	F	P	1979	39.7
Peacock	Producers Mineral Corp.	E	S	F, L	P	NAP	7.0
Pinto Valley	Newmont Mining Corp.	P	S	F, L	P	1943	71.9
Ray	Kennecott Minerals (SOHIO)	P	S	F	P	1955	62.4
Red Mountain	Kerr-McGee Corp.	E	U	F	P	NAP	37.2
Sacaton	ASARCO	T	S	F	P	1974	25.3
Safford (Kennecott)	Kennecott Minerals (SOHIO)	E	S	L	P	NAP	21.9
Safford Phelps Dodge	Phelps Dodge Corp.	E	U	F	P	NAP	127.9
San Manuel-Kalamazoo	Newmont Mining Corp.	P	U	F	P	1955	108.3
Sanchez	Inspiration Resources Corp.	E	S	F, L	P	NAP	17.1
Sierrita-Esperanza	Cyprus Minerals Co.	P	S	F, L	P	1959	94.8
Silver Bell	ASARCO	T	S	F, L	P	1954	20.5
Twin Buttes	Anamax	T	S	F, L	P	1969	78.7
Van Dyke	Van Dyke Copper, Sho-Me Copper	E	S	L	P	NAP	8.0
Vekol Hills	Newmont Mining Corp.	E	S	F	P	NAP	28.3
California:							
Lights Creek	Placer Amax	E	S	F	P	NAP	27.7
Walker	Calicopia Corp.	E	U	F	P	NAP	3.6
Maine: Bald Mountain	Superior Oil, Louisiana Land	E	S	F	M	NAP	14.3
Michigan:							
Presque Isle Syncline	AMAX	E	U	F	P	NAP	30.9
White Pine	Copper Range Co.	P	U	F	S	1953	51.1
Minnesota:							
Ely Spruce	International Nickel Co.	E	S	F	P	NAP	49.3
Minnamax	Bear Creek Min. Co. (Kennecott)	E	U	F	P	NAP	49.1
Montana:							
Butte Copper	Washington Corp.	P	S	F	P	1952	88.5
Heddleston	ASARCO	E	S	F	P	NAP	16.4
Troy	..do	P	U	F	O	1982	17.9
Neveda:							
New Ruth	Kennecott Minerals (SOHIO)	T	S	F, L	P	1970	27.2
Yerrington	Anaconda Co. (ARCO)	E	S	F, L	P	NAP	11.7
New Mexico							
Chino	Kennecott (SOHIO)/, Mitsubishi	P	S	F, L	P	1912	107.5
Continental Surface	U.V. Industries Inc.	T	S	F	P	1968	11.6
Continental Underground	..do	T	U	F	P	1968	13.0
Copper Flat	Quintana Minerals, Philbro	E	S	F	P	(7)	15.2
Nacimiento	Earth Resources Co.	E	S	L	S	NAP	1.4
Pinos Altos	Exxon Min. Co., Boliden Mining	E	S	F	P	NAP	10.6
Tyrone	Phelps Dodge Corp.	P	S	F, L	P	1970	127.2
Utah:							
Bingham Canyon	Kennecott Minerals (SOHIO)	P	S	F	P	1906	155.0
Carr Fork	Anaconda Co. (ARCO)	D	U	F	P	(8)	51.0
Washington: Sunrise	International Brenmac Dev. Corp.	E	U	F	O	NAP	8.1
Wisconsin:							
Crandon	Exxon Minerals Co.	E	U	F	P	NAP	40.0
Flambeau	Flambeau Min. Corp. (Kennecott)	E	S	F	M	NAP	10.2
Pelican River	Noranda Mines Ltd.	E	U	F	P	NAP	1.3
Wyoming: Kirwin	AMAX Inc.	E	S	F	O	NAP	26.5

See explanatory notes at end of table.

Table 2.—MEC copper properties included in this study—Continued

Country and property name	Ownership	Current status[1]	Mining method[2]	Milling method[3]	Deposit type[4]	Year of initial production[5]	Recoverable copper,[6] 10^3 mt/yr
Zaire:							
Dikuluwe, Mashamba	Gecamines	P	S	F	S	1975	146.1
Kakanda, Diselle	..do	P	U	F	S	1930	22.2
Kambove	..do	P	U	F	S	1926	37.2
Kamoto	..do	P	U	F	S	1972	97.4
Kipushi	..do	P	U	F	O	1926	42.5
Kov	..do	P	S	F	S	1956	139.4
Musoshi, Kinsenda	Sodimiza	P	U	F	S	1972	34.6
Tenke Fungurume	Gecamines	E	S	F	S	NAP	9.7
Zambia:							
Baluba	Zambia Consolidated Copper Mines Ltd.	P	U	F	S	1973	60.7
Chambishi	..do	P	U	F	S	1965	30.5
Chibuluma	..do	P	U	F	S	1965	16.4
Kalulushi East	..do	E	U	F	S	NAP	14.0
Konkola Division	..do	P	U	F	S	1957	42.1
Luanshya	..do	P	U	F	S	1931	46.0
Mufilira	..do	P	U	F	S	1933	102.3
Nchanga Division	..do	P	C	F, L	S	1965	317.3
Nkana Division	..do	P	C	F	S	1919	60.2
Zimbabwe: Mangula (Miriam)	MTD (Mangula) Ltd.	P	U	F	O	1958	11.0

NAP, Not applicable
[1] P, producing; T, temporarily shut down; D, developing; E, explored.
[2] S, surface; U, underground; C, combined surface and underground.
[3] F, flotation; L, leach.
[4] P, porphyry; S, sedimentary; M, massive; O, other.
[5] Initial year of significant production leading up to current operation or expected startup date for properties under development.
[6] Annual recoverable copper including mining and processing losses through refined copper.
[7] Copper Flat mill has been sold and shipped to OK Tedi, Papua New Guinea. Property has been downgraded to explored status.
[8] Carr Fork mill has been sold and shipped to OK Tedi, Papua New Guinea. Deposit has been sold to Kennecott Minerals Co. Development. Status is unknown.

As shown in table 1, Chile maintains the lead in demonstrated copper resources with 29 pct of the recoverable copper, followed by the United States with 17 pct, Australia 8 pct, Mexico 6 pct, Zaire 6 pct, Peru 6 pct, and Canada 5 pct. These seven countries account for nearly 78 pct of all MEC demonstrated copper resources.

Figure 1 shows the relationship of the demonstrated resources included in this study to the reserve base estimate *(6)*. Total land-based copper resources are estimated to be 1,600 million mt. Of the properties evaluated in this study, mines in production as of January 1, 1985, are estimated to contain 194 million mt, 58 pct of potentially recoverable copper, compared with 140 million mt copper potentially recoverable from nonproducing properties. Individual countries vary widely with respect to the proportion of recoverable resource associated with producing mines. Zambia and Zaire have nearly their entire resources from the producing properties while Canada, United States, Peru, and Philippines have 50 pct or less, and Mexico and Chile are approximately 75 pct. Australia, with the discovery of Olympic Dam, has only 15 pct of recoverable copper resource occurring in producing mines.

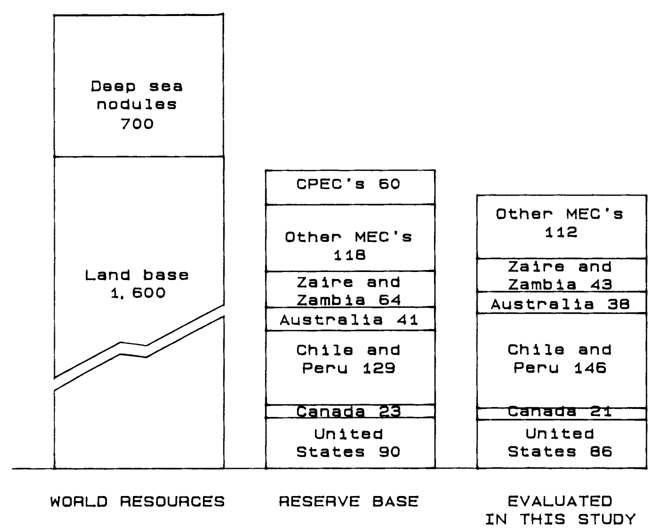

Figure 1.—Estimates of world copper resources (million metric tons of contained copper).

U.S. AND WORLD HISTORICAL PRODUCTION

Total copper contained in world mine production in 1985 was estimated to be 8.1 million mt, of which 82 pct was from MEC's *(6)*. Figure 2 shows the historical mine production of copper in 5-yr intervals from 1950 through 1985 *(7-13)*. The post-World War II period experienced a large growth in copper production averaging 6 pct/yr from 1950 through 1960, dropping to 3.6 pct/yr from 1961 through 1975. The last decade (1976 through 1985) has averaged 1.2 pct/yr, due in large part to the excess supply and general world economic recession.

The United States has experienced the greatest decline in production during recent years, with the loss of approximately one-third of its production since 1970. Zaire and Zambia have been on a slow decline since 1975, with a 13-pct reduction in total production. Chile and Peru have shown the largest gain in mine production, with a combined increase of over 80 pct from 1970 through 1985. The Third World countries overall have witnessed a major increase of 90 pct from 1970 through 1985.

The relative ranking of copper-producing countries has changed over the last 10 yr. The United States has been losing ground on the world market since 1970, with market share falling from 26 pct to less than 14 pct of total world production. Zaire and Zambia market shares have also dropped from 19 pct to 13 pct over the same period. Canada and the centrally planned economy countries (CPEC's) are holding their 10-pct and 18-pct shares, respectively. Chile and Peru have cumulatively increased from approximately 16 to 22 pct, but the largest gains are in smaller Third World countries such as Mexico, Papua New Guinea, and Iran, which as a group have increased from 17 to 25 pct of world copper production.

Government involvement in direct ownership of copper operation other than CPEC's is 48 pct of the evaluated production capacity and 65 pct of the demonstrated resource. The two largest copper producing companies in MEC's, CODELCO-Chile and Zambia Consolidated Copper Mines Ltd. (ZCCM), are 100 pct government owned.

EXTRACTION AND PROCESSING TECHNOLOGY

Mining and beneficiation methods used are listed in table 2 along with ownership, current status, deposit type, initial year of production, and annual copper production.

MINING

Open pit mining is the primary method used in the United States, and it accounts for over 80 pct of recoverable copper. In MEC's, nearly 60 pct of the copper is recovered by open pit. Open pit mining requires a relatively shallow deposit with stripping ratios (tons of waste rock to tons of ore) normally between 1:1 and 2:1. A typical surface copper mine uses rotary blasthole drills and a shovel-truck combination for the loading and hauling operation.

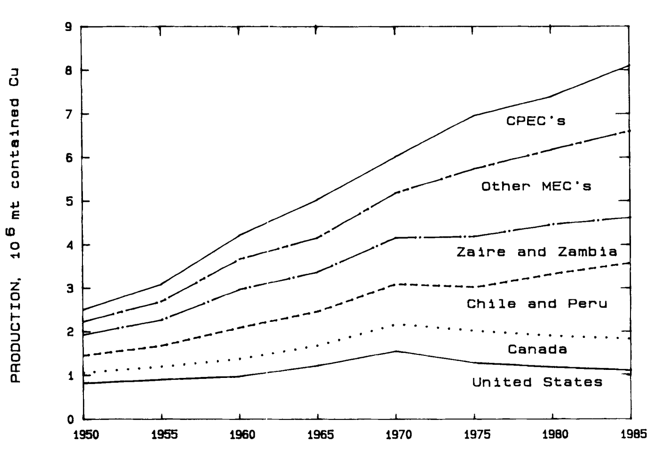

Figure 2.—World copper mine production, 1950-85.

Underground mining methods vary depending on the orientation and depth of the ore zone, structure, topography, and relative strengths of wall rock and ore. The principal underground mining method used in the copper industry is block caving: San Manuel, United States, and El Teniente, Chile, are prime examples. Sublevel stoping is second in order of importance with Kidd Creek, Canada, and Mt. Isa, Australia, as examples.

PROCESSING

Beneficiation

Beneficiation of the ore is performed by either flotation or leaching. In flotation, the ore is crushed and ground to a fine sand size with sulfides recovered in flotation cells. Sulfide particles that have been treated with reagents attach to air bubbles, float to the top of the cell as a froth, and are recovered as a concentrate. After filtering and drying, the concentrate is shipped to a smelter for further processing.

Leaching is the dissolving of copper from oxide and sometimes sulfide minerals with dilute sulfuric acid and recovering the leach solution for further processing. The copper-bearing material to be leached can be in the form of dumps, specially prepared heaps, vats, or broken material still essentially in place in the ground. As the acid percolates through the material, it dissolves and transports the copper to a collection system. The copper is then recovered from leach solutions by solvent extraction-electrowinning (SX-EW) to directly produce refined copper or precipitate copper on scrap iron (cementation). Precipitates are usually processed through a smelter.

Flotation is by far the most widely used method to recover copper, and it accounts for over 91 pct of the copper produced in the United States and 96 pct in MEC's. Leaching accounts for the remaining production in the United States and MECs. The current trend is increased employment of leaching technology to recover copper from low-grade dump material, with SX-EW being used to supplement the flotation operation and reduce smelting requirements.

Smelting and Refining

Copper concentrates are shipped to smelters for reduction to metallic copper. Conventional roasting and reverberatory furnaces oxidize and melt the sulfide concentrate. The melt is converted to copper metal primarily in Pierce-Smith converters producing a blister copper product. Blister copper is reduced to remove oxygen and is cast into anode shapes for shipment to an electrolytic refinery where impurities are removed and cathode copper is produced. Cathodes are cast into shapes for use in manufacturing copper products.

NEW TECHNOLOGY

The copper industry is faced with the challenge of recovering copper from lower grade material. The United States has been a leader in applying innovative technology to increase efficiency and reduce costs in many areas of the copper recovery cycle. The trend over the last few decades has been to larger capacity equipment in the mine loading and hauling cycles and crushing and flotation cycles in beneficiation. This trend has contributed significantly to the domestic copper industry's ability to stay competitive on the world copper market in spite of the relatively low copper grades.

New technology applied to mining in recent years, and having potential for widespread applications, are portable in-pit crushing-overland conveyor systems, on-line (automated) production monitoring and control (dispatching), and in situ solution mining. In addition, trolly-assist truck haulage out of the pit saves fuel and increases haul speeds.

Autogenous and semiautogenous grinding eliminates or reduces the use of grinding media such as steel balls or rods. Automated process control of concentrates has become popular with recent advances in instrumentation and minicomputers *(14)*.

Smelting technology has progressed in areas of energy savings and improved efficiency in ambient air quality control. Continuous smelting and converting technology, such as the Noranda, Mitsubishi, and Outokumpu processes are versions of this concept. The Outokumpu flash smelting process has been adopted in about two-thirds of the new smelters constructed in the world since 1970, and is now often considered the "conventional" smelting process *(15)*. The adoption of solvent extraction-ion exchange-electrowinning for treating pregnant leach solutions instead of the cement copper route is increasing in use.

PRODUCTION COSTS

Production costs were estimated for 241 copper properties in MEC's, 110 were producing and 131 were either developing, temporarily shut down, or explored as of January 1985. Table 3 and figures 3 and 4 show the estimated production costs for major producing countries at 0- and 15-pct discounted-cash-flow rates of return (DCFROR's).

Included in the nonproducing properties are 6 developing and 27 temporarily shut down operations. All of the temporarily shut down operations are located in Canada, Philippines, and United States. They account for 20, 35, and 26 pct of total installed mine production capacity in their respective countries, or 11 pct of MEC production capacity. Currently developing properties can potentially increase production capacity by 3 pct.

Five major copper producing countries, Australia, Chile, Papua New Guinea, Zaire, and Zambia, had average breakeven production costs (0-pct DCFROR) at producing properties below the average 1985 copper U.S. producer price of \$0.67/lb. Average breakeven production costs for Mexico, the Philippines, and the United States were marginally above the January 1985 copper producer price at \$0.70/lb. India is a high-cost source for copper at an average cost of \$1.10/lb because of higher than normal mining, smelting, and refining costs and low byproduct credits.

Mining and milling costs are somewhat uniform on a dollar per pound copper basis. Australia is lower than average as a result of efficient operations and high-grade deposits. Canadian and Philippine milling costs are high because of low mill feed grades. Canada also has a greater proportion of multimetallic concentrators. Smelting costs vary from \$0.05/lb to \$0.50/lb. Canadian and the Republic of South Africa smelting and refining costs are high because smelting of coproduct lead and zinc is included. India has high energy costs and poor availability of electrical power, which contribute to high smelting and refining costs. Zambia has low hydroelectric power costs resulting in low smelting and refining costs.

Byproduct credit has a major impact on costs in some countries. Canada, the Republic of South Africa, and the "other" category on table 3 have from \$0.30/lb to \$0.49/lb copper byproduct credits because of a high proportion of polymetallic, massive sulfide operations. Zaire has cobalt as its main byproduct. Operations in the Philippines derive most of their byproduct credit from the high gold content in the ores. India, as already mentioned, suffers from low byproduct credits.

Copper mines in recent years have had to face numerous challenges in order to remain competitive in a depressed market. Some mines, either because of low grades or other adverse operating conditions, have been forced to close, some never to open again. Operations that are surviving the current crisis are doing so by implementing severe cost-cutting measures. Areas that have seen the greatest changes are mine planning, labor, energy, smelter contracts, tax relief, corporate reorganizations, and new technology.

Changes in the mine plan, such as selective high grading, postponing development, and steepening pit wall slopes, are short-term means of cost reduction and cannot be sustained over the mid- to long-term range. Long-term changes in mine plans, such as expansions to reduce unit costs and modernizations to upgrade deteriorating and obsolete equipment, can reduce operating costs significantly.

Labor costs constituted approximately 45 pct of the total mining cost and 30 pct of total milling cost in the United States in 1984. In addition, the labor portion of smelting and refining was 41 pct and 53 pct, respectively. This becomes significant to reducing production costs in the United States in the future in light of the 1986 copper industry labor contract negotiations. The new contracts reduce wages and benefits by 20 to 30 pct, which are not reflected in this evaluation.

Current shortage of concentrates has prompted smelters to offer low-cost contracts to producers in order to secure adequate feed to keep smelters at full capacity. Design capacity is the optimum operating rate for most smelters because it maximizes efficiency and minimizes unit costs.

Tax relief has played an important role in the Philippines and Canada, with tax holidays and special tax rates for the depressed copper industry. Subsidies to promote continued production or reopen operations have also been used to support the local industries.

Corporate reorganizations, such as happened in Zambia with the formation of the Zambia Consolidated Copper Mines Ltd., can lead to more efficient use of resources. Effective use of equipment and labor in the overall mine plan is the major objective. Domestically, a number of spinoffs and reorganizations have taken place in an attempt to improve efficiencies and lower costs.

New technology has the highest potential for long-term cost reductions. The use of in-pit crushing and conveyor haulage out of the pit is employed at operations in the United States and Canada. Trolley-assisted truck haulage on the steep grades out of the pit is being used in Palabora, Republic of South Africa, and is being considered in Zaire. Semiautogenous grinding mills are being installed in more operations every year because of lower energy and grinding medium requirements. The trend to larger capacity trucks is continuing. Smelter technology using oxygen enrichment saves fuel and increases the efficiency of downstream SO_2 recovery for pollution control.

Table 3.—Copper production costs for producing and nonproducing operations in selected MEC's

(All costs are in January 1985 U.S. dollars per pound of refined copper)

Country	Number of mines	Operating cost: Mine	Operating cost: Mill	Smelter-refinery cost[1]	Byproduct credit	Net operating cost[2]	Recovery of capital	0-pct DCFROR: Taxes and royalties[3]	0-pct DCFROR: Total cost[4]	15-pct DCFROR: Taxes and royalties[5]	15-pct DCFROR: Return on investment	15-pct DCFROR: Total cost[6]
PRODUCING												
Australia	3	0.19	0.10	0.32	0.07	0.55	0.08	0.01	0.64	0.06	0.07	0.76
Canada	14	.27	.35	.45	.32	.76	.12	.01	.89	.07	.06	1.01
Chile	9	.29	.16	.12	.06	.51	.06	.02	.59	.04	.03	.65
India	5	.34	.21	.42	.03	.93	.12	.05	1.10	.20	.11	1.36
Peru	6	.19	.20	.32	.05	.65	.14	.02	.81	.08	.07	.94
Philippines	7	.31	.27	.22	.21	.58	.07	.05	.70	.07	.04	.76
South Africa, Republic of	4	.41	.25	.50	.49	.67	.13	.00	.80	.04	.04	.88
United States	13	.23	.20	.23	.09	.56	.12	.02	.70	.06	.08	.82
Zaire	7	.39	.19	.25	.41	.42	.07	.08	.57	.15	.05	.69
Zambia	8	.31	.14	.05	.09	.42	.12	.06	.60	.14	.04	.72
Other	34	.25	.27	.29	.30	.51	.10	.05	.66	.11	.09	.81
Total or av.	110	.28	.21	.22	.16	.55	.09	.03	.67	.08	.05	.77
DEVELOPING												
Total or av.[7]	5	0.24	0.11	0.35	0.03	0.66	0.12	0.03	0.80	0.11	0.14	1.03
TEMPORARILY SHUT DOWN												
Canada	6	0.38	0.37	0.33	0.16	0.91	0.16	0.02	1.10	0.17	0.13	1.37
Philippines	8	.35	.36	.23	.08	.86	.12	.08	1.06	.15	.13	1.26
United States	13	.34	.31	.37	.07	.95	.07	.03	1.05	.06	.08	1.16
Total or av.	27	.34	.33	.33	.08	.92	.09	.04	1.06	.10	.10	1.21
EXPLORED												
Australia	5	0.37	0.20	0.78	0.99	0.35	0.30	0.03	0.68	0.79	1.28	2.42
Canada	15	.56	.49	.50	.24	1.30	.26	.06	1.63	.34	.49	2.13
Chile	7	.22	.21	.17	.05	.56	.15	.05	.76	.25	.39	1.20
Peru	6	.25	.22	.40	.16	.71	.22	.06	.99	.21	.39	1.31
Philippines	6	.19	.38	.24	.10	.71	.28	.08	1.07	.19	.49	1.39
United States	34	.44	.42	.52	.30	1.07	.23	.04	1.35	.23	.63	1.94
Other	25	.29	.30	.34	.16	.77	.30	.10	1.17	.46	.65	1.88
Total or av.	98	.32	.30	.35	.17	.80	.23	.07	1.10	.32	.54	1.67

[1] Includes smelting and refining charges and transportation costs to smelter and refinery (but not to market) for copper and post mill (noncopper) commodities.
[2] Total cost minus byproduct credit.
[3] Includes property, State, Federal, and severance taxes and royalties, calculated at a 0-pct DCFROR.
[4] Includes recovery of all costs of production including capital but does not include a profit.
[5] Includes property, State, Federal, and severance taxes and royalties, calculated at a 15-pct DCFROR.
[6] Includes recovery of all costs of production including capital and return on investment at a 15-pct DCFROR.
[7] Olympic Dam, Australia, not included to avoid disclosing company proprietary data.

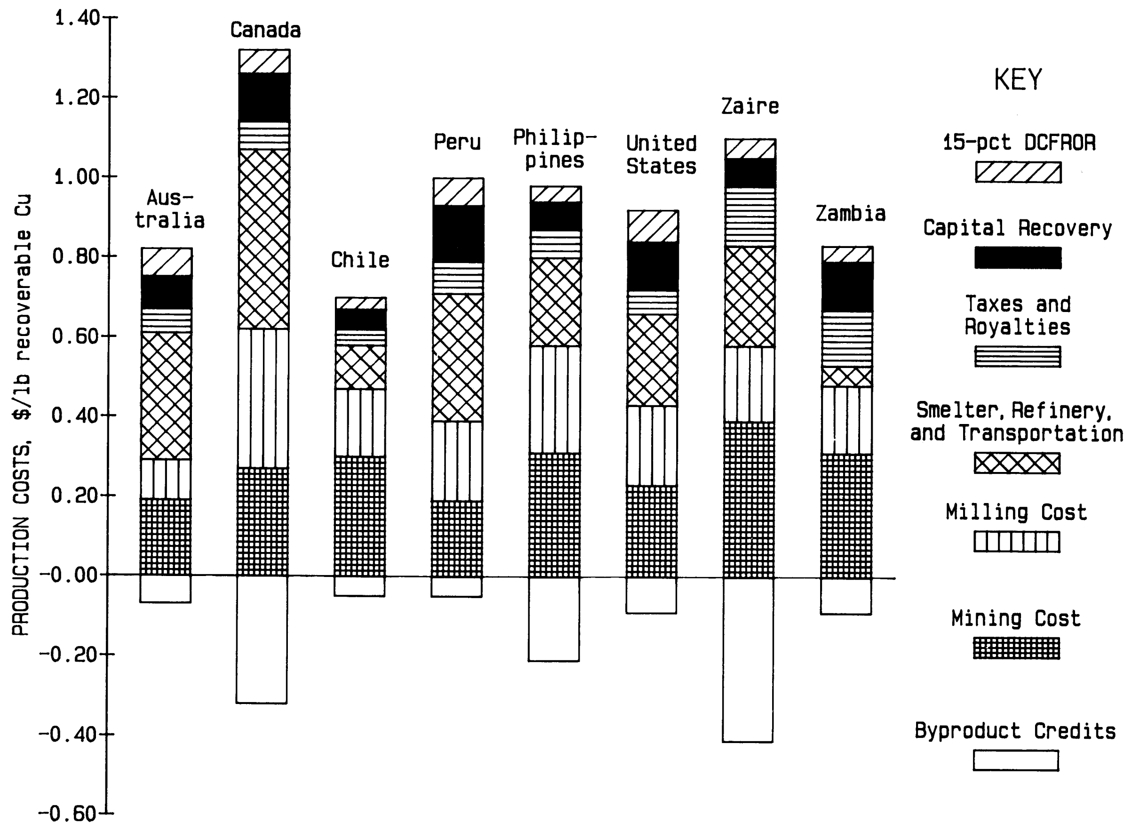

Figure 3.—Copper production costs for producing mines in selected MEC's (January 1985 U.S. dollars).

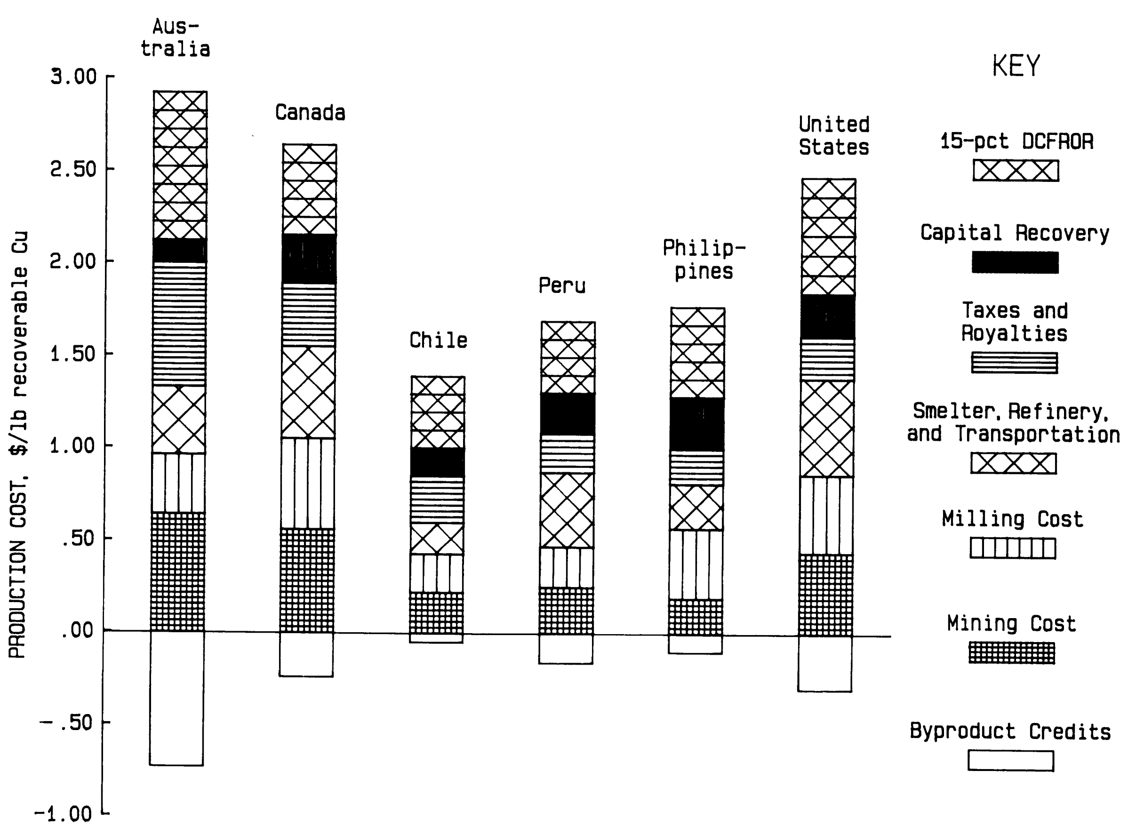

Figure 4.—Copper production costs for nonproducing deposits in selected MEC's (January 1985 U.S. dollars).

AVAILABILITY

TOTAL RECOVERABLE

A total of 241 mines and deposits in MEC's were analyzed, 62 domestic plus 179 foreign (see table 3). These mines and deposits account for over 90 pct of the Bureau's reserve base for copper *(6)*. As of January, 1985, 110 of the mines evaluated were producing (13 in the United States) and another 27 mines were temporarily shut down (13 in the United States) as a result of the depressed copper market. A total of 104 deposits evaluated did not have an installed capacity as of the study date. Six were currently being developed and the remaining 98 deposits were still being explored.

Total refined copper potentially available from producing mines and nonproducing deposits analyzed is shown in figure 5. Production costs for operating mines are estimated at 0- and 15-pct-DCFROR rates, while nonproducing deposits are estimated at the 15-pct rate DCFROR only.

A total of 333 million mt of copper is potentially recoverable from these deposits (60 pct from producers and 40 pct from nonproducers). Chile is the largest potential source for copper in the MEC's with 97 million mt recoverable copper, followed by the United States with 57 million mt and Australia with 26 million mt.

In 1985, the average copper price listed on the London Metal Exchange was \$0.65/lb and Metals Week U.S. producer cathode was \$0.67/lb *(15)*. At a longrun total production cost (including a 15-pct DCFROR) of \$0.67/lb copper (equal to the average 1985 domestic producer price), 123 million mt copper could be recovered from currently producing mines, primarily in Chile, Zaire, and Zambia, and an additional 19 million mt from nonproducing deposits in MEC's. Of this, only 10 million mt would be available from producing mines in the United States and less than 1 million mt from nonproducing deposits.

At a total production cost (including a 15-pct DCFROR) of \$0.90/lb, currently operating mines located primarily in Chile, United States, Zaire, and Zambia, could produce 177 million mt and nonproducing deposits could produce 59 million mt over the life of the operations. In the United States, the available copper from producing mines would increase to over 28 million mt copper and 2 million mt copper from nonproducing deposits.

ANNUAL CAPACITY

Potential 1985 production at full capacity from producing mines in MEC's is shown in figure 6. The lower curve indicates operating cost while the upper curve indicates costs at a 0-pct DCFROR. Total annual capacity potential in 1985 from currently producing operations analyzed in this study was nearly 6 million mt copper. Actual production for 1985 was 6.4 million mt copper from MEC's. Installed capacity at properties analyzed, including temporarily shut down operations, is 6.7 million mt copper per year.

At a production cost equal to the January 1985 copper price of \$0.67/lb, 4.7 million mt of copper could potentially be produced annually but only 3.5 million mt of this annual production would also be able to cover recovery of capital. At a \$0.90/lb production cost, operations representing approximately 5.5 million mt copper capacity would be able to cover operating costs and 5.0 million mt of this capacity would also be able to recover capital costs and achieve a 0-pct DCFROR.

Regional curves are shown in figure 7 for major copper-producing countries and regions. Annual production capacity for 13 mines evaluated in the United States was approximately 1.1 million mt in 1985. Properties evaluated in Canada have a cumulative annual capacity of 0.6 million mt copper or 85 pct of 1985 production. INCO's and Falconbridge's Sudbury operations along with Huson Bay's Flin Flon production, were not evaluated but constitute an annual capacity of 250,000 mt copper. Capacity from evaluated mines in Chile and Peru totals 1.4 million mt copper. Nearly all are able to cover direct operating cost at 1985 copper price levels and 1.1 million mt of the total would be able to achieve at least a 0-pct DCFROR. Zaire and Zambia have operations totaling almost 1.2 million mt copper capacity of which over 1 million mt capacity is able to cover direct operating costs and 0.8 million mt can achieve at least a 0-pct DCFROR.

Potential annual production of copper from producing copper mines in MEC's from 1985 through 1995 is shown in figures 8 and 9. The curves reflect the production capacity

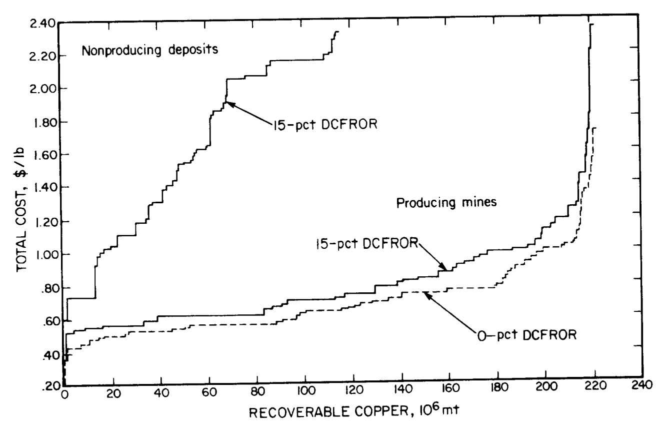

Figure 5.—Potential total copper available from MEC's (January 1985 U.S. dollars).

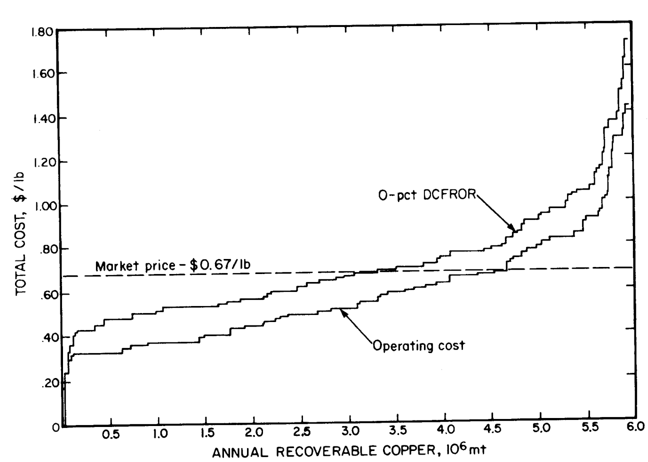

Figure 6.—1985 copper capacity from producing mines in MEC's (January 1985 U.S. dollars).

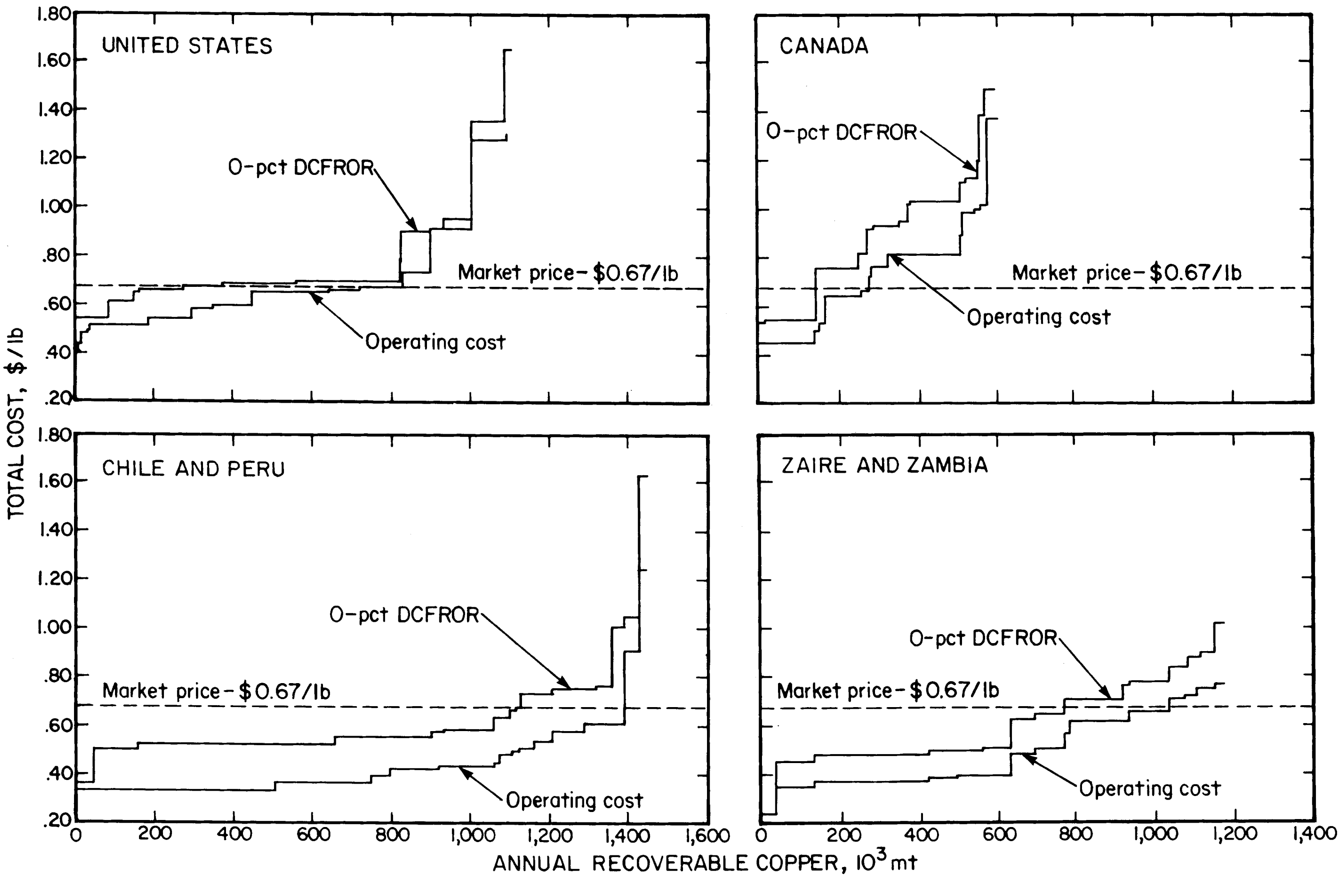

Figure 7.—1985 copper capacity from producing mines in selected MEC's (January 1985 U.S. dollars).

of existing mines, including planned expansions, but exclude new mines scheduled to open before 1995, at selected production cost ranges including a 15-pct DCFROR.

Allowances have been made to bring the temporarily shut down mines back on line. The added capacity supplements current producers and replaces lost production as reserves are exhausted in some deposits. Projected production for MEC's is stable and large reductions in capacity are not anticipated during this time period.

Canada and the Philippines have been the hardest hit because of their low grades and relatively high production costs. The United States will be able to survive at a reduced capacity. High-cost producers have closed during the current market recession. Lower labor costs and corporate reorganizations have helped the remaining operations to reduce production costs and return to profitability. Chile still maintains the lead in low-cost producers because of the high-grade resources, high level of technology, and low labor costs. Zambia and Zaire appear as low-cost producers owing in part to high byproduct credits in Zaire and low energy costs in Zambia. All three countries have experienced major currency devaluations in recent years.

The United States has about 400,000 mt copper capacity on temporary shut down status that could be brought back into production in just a few years. As shown in figure 9, this could bring domestic production up to over 1.5 million mt copper annually.

Zambia is projected to have major cuts in production capacity after 1997 because of exhaustion of reserves. However, as stated earlier, Zambia has additional lower grade

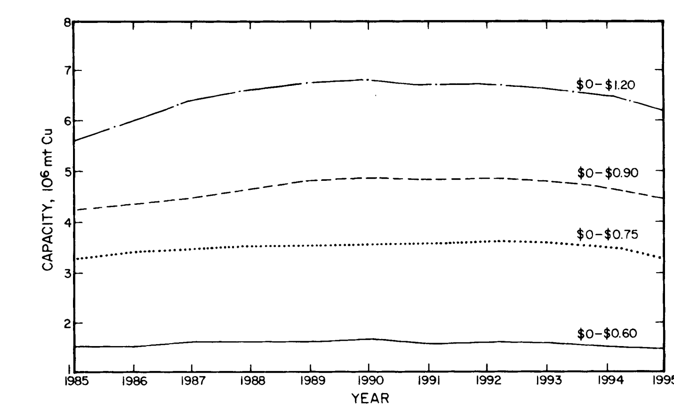

Figure 8.—Potential annual copper production from producing mines in MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

resources that may be mined in the future. Zaire appears to have sufficient resources to extend production at current levels beyond the year 2000.

Canada's production is projected to peak in 1987 if temporarily inactive mines representing 145,000 mt copper annually are brought back on line. A slow decline after that is expected to level off in the early 1990's.

Chile and Peru show significant gains in capacity to the year 1990, with a leveling off at approximately 1.8 million mt copper, over 70 pct of which is in Chile.

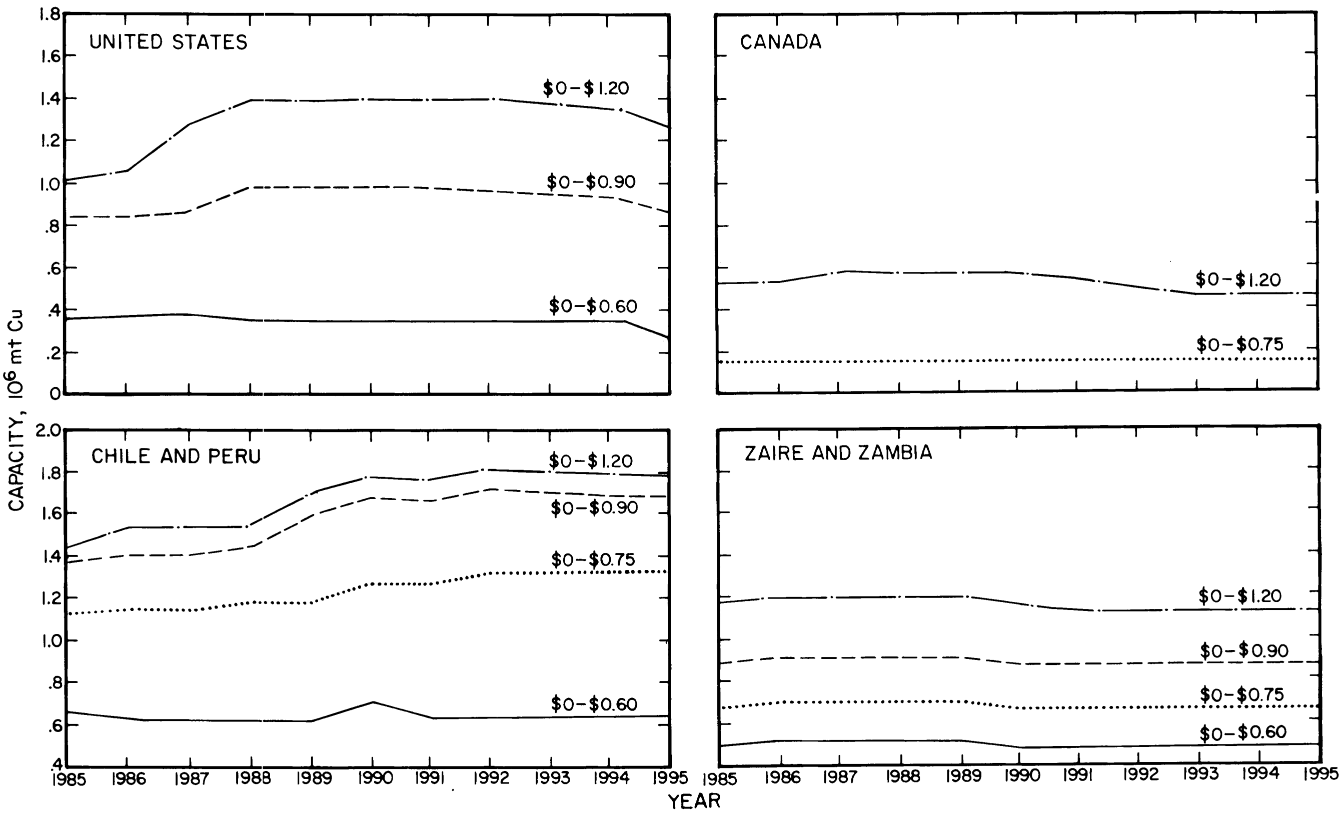

Figure 9.—Potential annual copper production from producing mines in selected MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

CONCLUSIONS

A total of 241 mines and deposits were evaluated for this summary; 110 were producing, 27 were temporarily shut down, and 6 were being developed as of January 1985. The remaining 98 deposits were explored with no definite plans for development. Total demonstrated copper resources evaluated for MEC's contain 465 million mt of in situ copper, an estimated 349 million mt of which is recoverable to refined copper. The two countries with the largest copper resource are Chile with 139 million mt of contained copper and the United States with 86 million mt. Recoverable copper totals are 112 million mt for Chile and 57 million mt for the United States. Average resource grade for producing mines in Chile and the United States is 1.02 and 0.65 pct Cu, respectively, not including leach material that provides a considerable cost advantage to Chile.

The United States has historically been the leading copper-producing nation in the world until in 1982 when uneconomic operations started shutting down. In 1982, Chile took over first place in mine production and is continuing to expand.

The domestic copper industry has been faced with low copper grades and high labor, energy, and environmental costs. Compounding the effect of these factors on production cost is the recent depression in the copper market that has lasted from 1981 through 1985. High copper inventories, excess production capacity, and a strong U.S. dollar, combined with foreign government subsidies of copper production, have all contributed to low copper prices.

REFERENCES

1. Copper Development Association, Inc. Copper and Copper Alloy Mill Products to U.S. Markets—1985, CDA Market Data. May 28, 1986, 1 p.

2. Jolly, J.L.W. Copper. Ch. in Minerals Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 197-221.

3. Rosenkranz, R.D., E.H. Boyle, Jr., and K.E. Porter. Copper Availability—Market Economy Countries. A Minerals Availabiliy Program Appraisal. BuMines IC 8930, 1983, 30 pp.

4. Cox, D.P., R.G. Schmidt, J.D. Vine, H. Kirkemo, E.B. Tourtelot, and M. Fleischer. Copper. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 163-189.

5. Lowell, J.D., and J.M. Guilbert. Lateral and Vertical Alteration-Mineralization Zoning in Porphyry Ore Deposits. Econ. Geol., v. 65, 1970, pp. 373-408.

6. Jolly, J.L.W. Copper. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 42-43.

7. Cole, J.W. Copper. Ch. in BuMines Minerals Yearbook 1968, v. 1, pp.463-493.

8. Jolly, J.L.W. and D.L. Edelstein. Copper. Ch. in BuMines Mineral Yearbook 1984, v. 1, pp. 291-340.

9. McMahon, A.D., and G.N. Greenspoon. Copper. Ch. in BuMines Minerals Yearbook 1958, v. 1, pp. 371-420.

10. Meyer, H.M., and G.N. Greenspoon. Copper. Ch. in BuMines Minerals Yearbook 1953, v. 1, pp. 381-436.

11. Schroeder, H.J. Copper. Ch. in BuMines Minerals Yearbook 1972, v. 1, pp. 473-509.

12. Schroeder, H.J., and G.J. Coakley. Copper. Ch. in BuMines Minerals Yearbook 1976, v. 1, pp. 475-520.

13. Wideman, F.L. Copper. Ch. in BuMines Minerals Yearbook 1963, v. 1, pp.443-488.

14. Weiss, A., N. Themelis, and N. Guernsey. Technological Innovation in the Copper Industry. Volume I, Methodology; Volume II, Technology Forecast; Volume III, Cost/Benefit Analysis (contract J0100063, Mineral Systems Inc.). BuMines OFR 125(1)-83, Mar. 1983, 58 pp., NTIS PB 83-227066; OFR 125(2)-83, Mar. 1983, 137 pp., NTIS PB 83-227074; OFR 125(3)-83, Mar. 1983, 91 pp., NTIS PB 83-227082.

15. Strauss, S.D. Copper. Eng. and Min. J., v. 187, No. 3, Mar. 1986, pp. 29-32.

FLUORSPAR

INTRODUCTION

Fluorspar is the commercial name for fluorite, a mineral composed of calcium fluoride (CaF_2). The United States is one of the largest consumers of fluorspar with a total apparent consumption in 1985 of about 620,000 mt. U.S. production for 1985 was estimated at 60,000 mt from the one remaining major producer and three other minor operations. As a result of low production levels and the largest import volume of all market economy countries (MEC's), the United States has an import reliance greater than 90 pct. Mexico and the Republic of South Africa are the largest U.S. suppliers, accounting for approximately 80 pct of total imports.

Fluorspar is marketed in three major grade classifications—acid, ceramic, and metallurgical. Acid-grade fluorspar contains not less than 97 pct CaF_2 and is consumed primarily in the production of anhydrous and aqueous (70 pct) hydrofluoric acid (HF). The largest uses of HF are in the production of fluorocarbons and aluminum, which together account for approximately 70 pct of the total U.S. HF consumption. Other uses include petroleum alkylation, stainless steel production, uranium processing, rare metal production, glass etching, fluoride salts, and herbicides. HF is also used to make elemental fluorine (*1*).[1]

There are two standard forms of ceramic-grade fluorspar, No. 1 ceramic, containing 95 to 96 pct CaF_2, and No. 2 ceramic, containing 85 to 90 pct CaF_2. Uses for ceramic-grade products include fiberglass insulations, fluxes, flint glass, white or colored opal glasses, enamels, and welding rod coatings (*2*).

Metallurgical-grade fluorspar traded on the world market contains a minimum of 60 pct CaF_2, and is primarily consumed by the steel industry. In the United States, a minimum of 70 pct CaF_2 (effective percent), with not over 0.1 pct S, and not over 0.25 pct Pb, is specified (*3*). The effective percentage is calculated by multiplying the silica content by a factor of 2.5 and subtracting that amount from the percentage of CaF_2.

In addition to metallurgical grade, an increasing percentage of the fluorspar used in the steel industry is in the form of briquettes made from flotation concentrates of either acid or ceramic grade. The grades of the briquettes range widely and depend upon consumer preference. High grades (90 pct CaF_2) usually have no additive fluxes, but low grades (25 pct CaF_2) are diluted by other fluxing minerals or scrap ore materials. Briquettes are normally manufactured near consuming centers in order to blend in materials otherwise lost as waste.

Most of the information presented in this chapter is an updated (from 1984 to 1985) summary of Bureau of Mines Information Circular 9060 "Fluorspar Availability—Market Economy Countries and China. A Minerals Availability Appraisal" (*4*).

This chapter examines the availability of acid-, ceramic-, and metallurgical-grade fluorspar from 13 countries and the United States. The average total cost of production for the concentrates was estimated using local prices and price proportions for the three grades of fluorspar. Additional costs for drying or further processing, such as to HF, were not included. Final products were delivered to the port of export or point of local consumption within each country. The cost of transportation to importing countries would therefore need to be added to the costs estimated for this study.

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

GEOLOGY AND RESOURCES

GEOLOGY

Fluorite occurs in a wide variety of geological environments, which indicates that deposition takes place in a number of different ways. The following are 10 different modes of occurrence of fluorspar *(2)*.

1. Fissure vein deposits commonly occur along faults or shear zones and are the most readily recognized form of fluorspar occurrence in the world. Although the vein structure may be persistent, the fluorspar mineralization commonly occurs as lenses or ore shoots separated by barren zones. Fissure veins occur in igneous, metamorphic, and sedimentary rocks.
2. Stratiform, manto, or bedded deposits occur as replacements in carbonate rocks. Some beds are replaced adjacent to structural features such as joints and faults. In frequent instances, there is a capping of sandstone, shale, or clay.
3. Replacement deposits in carbonate rocks along the contact with acidic igneous intrusives are another common type of deposit. Deposits do not have to be the result of contact metamorphism, but may be introduced later, following the contact zone as a conduit and replacing the limestone.
4. Stockworks and fillings in shear and breccia zones are another form in which fluorspar occurs. The Buffalo deposit in the Transvaal consists of a network of fluorspar veinlets in sill-like bodies that are inclusions in the granite of the Bushveld complex.
5. Carbonatite and alkalic rock complexes may have fluorspar at their margins. Fluorspar grades are not usually sufficient to be economic, but the Okorusu deposit in Namibia is made up of a number of bodies of fluorspar in limestones, quartzites, and related rocks which have been intruded and metamorphosed by an alkaline igneous rock complex.
6. Residual deposits of fluorspar are formed in clayey and sandy residuum that results from surficial weathering of fluorspar veins and replacement deposits. These deposits may be the sources of metallurgical-grade fluorspar. They include detrital deposits blanketing the apex of veins and the upper portions of the veins themselves that have been deeply weathered to depths of 100 ft or more.
7. Fluorspar may also occur as a major gangue mineral in lead and zinc veins. Two operations in the Parral area of Mexico are treating the tailings of lead-zinc mines to recover fluorspar from previously discarded gangue minerals.
8. Breccia pipes may contain fluorspar in economic quantities.
9. Fillings in open spaces of veins or stratiform deposits may be filled with fluorspar.
10. Flurospar may occur in unconsolidated clayey and sandy pyroclastic sediments in the beds of former lakes. The Pianciano deposit near Rome, Italy, is an example of this type. Fluorine from igneous sources permeated the lake sediments and is present as finely disseminated crystals.

Deposits containing economic concentrations of fluorspar are located on every continent, but the major operations are in Mexico, the Republic of South Africa, China, Spain, Italy, Kenya, Morocco, Thailand, United Kingdom, U.S.S.R., Mongolia, and the United States. Additional information on fluorspar deposits can be found in U.S. Geological Survey Professional Paper 820 *(5)*.

RESOURCES

Fluorspar resources for 34 U.S. and 50 non-U.S. properties evaluated total 152 million mt of contained CaF_2 at the demonstrated level. The total world reserve base is estimated to be 328 million mt contained CaF_2, and identified world flurospar resources include 408 million mt *(6)*. This evaluation analyzes the availability of 121 million mt contained fluorspar from 52 properties. Figure 1 compares the evaluated resources to the reserve base and identified world resources. Demonstrated fluorspar resources are listed by country in table 1; table 2 lists the deposits evaluated in this study.

U.S. resources include approximately 32 million mt from 34 properties. The average total cost of production at 30 of the U.S. properties was determined to be above (by at least 10 pct) the 1985 published U.S. producer prices of $171/mt acid grade, $165/mt ceramic grade, and $125/mt metallurgical grade *(7)*. Although the resources for these properties are included in table 1, they were not included in the production cost and availability analyses. Resources from 14 of these properties were found to be marginally economic to subeconomic, with average total costs between 10 to 50 pct higher than the 1985 published prices. The remaining 16 properties were found to have total costs greater than 1.5 times the current published prices. High costs hamper development plans for the majority of domestic fluorspar

Table 1.—Summary of demonstrated fluorspar resources as of January 1985

Country	In situ ore, 10^3 mt	CaF_2 grade, pct	CaF_2, 10^3 mt: Contained	CaF_2, 10^3 mt: Recoverable
China	35,686	63	22,471	16,887
France	20,722	42	8,703	6,549
Germany, Federal Republic of	W	W	W	W
Italy	17,422	41	7,105	6,171
Kenya	5,700	41	2,565	2,166
Mexico[1]	31,091	64	19,865	17,134
Morocco	W	W	W	W
Namibia	7,879	50	3,940	2,963
South Africa, Republic of[2]	165,038	22	36,619	29,095
Spain	25,710	33	8,493	6,530
Thailand	2,703	47	1,260	1,010
Tunisia	W	W	W	W
United Kingdom	6,069	40	2,412	1,918
United States[3]	219,666	14	31,666	W
Total or av.	554,572	27	151,984	94,780

W Withheld to protect confidentiality, included in totals.

[1] Includes Rio Colorado, operation closed 1984, not included in availability analyses.

[2] Includes Marico, operation closed 1983, not included in availability analyses.

[3] Includes fluorspar reserves (economic) for 4 properties in Illinois, Nevada, and Texas plus other resources from 30 operations determined to be uneconomical at a breakeven (0-pct) discounted-cash-flow rate of return (DCFROR). These 30 are located in Alaska, Arizona, Colorado, Idaho, Illinois, Kentucky, New Mexico, Nevada, Tennessee, and Utah. Recoverable tonnage calculated for reserves only. Other resources not included.

Table 2.—World fluorspar properties included in this study

Country and property name	Current status[1]	Mining method	Beneficiation method	Products recovered
China:				
Da Gai Tang mines, Hua De mill	P	Shrinkage	Flotation	Acid, metallurgical.
De An District	P	Open pit	..do	Acid, metallurgical, ceramic.
Hei Shao Tou Mine, Bai Yun He Pe mill	P	Shrinkage	..do	Acid, metallurgical.
Hong An District	P	Descending crosscuts	..do	Do.
Pong Lai mines, Fu Shan mill	P	Shrinkage	..do	Do.
Pong Lai mines, Xian Shan mill	P	..do	..do	Do.
Wu Yi District	P	..do	..do	Do.
France:				
Escaro	P	Open pit	..do	Acid.
Fontsante	P	Shrinkage	..do	Do.
Le Burc	P	Top slicing	Heavy media	Acid, metallurgical.
Montroc	P	Open pit	Flotation	Acid, ceramic.
Morvan District	N	..do	..do	Acid.
Rossignol	P	Shrinkage	Heavy media	Metallurgical, barite.
Germany, Federal Republic of:				
Clara	P	Withheld (proprietary)	Flotation	Acid, barite products.
Kaefersteige	P	..do	..do	Acid.
Italy:				
Domusnovas	N	Open pit	..do	Do.
Mineraria Silius	P	Sublevel	..do	Acid, metallurgical, lead, barite products.
Pianciano	N	Open pit	Gravity	Metallurgical.
Kenya: Kenya Flurospar Co	P	..do	Flotation	Acid.
Mexico:				
El Realito (Rio Verde)	P	Stoping with fill	..do	Acid, metallurgical.
El Refugio (Rio Colorado)[2]	PP	..do	..do	Acid, metallurgical, ceramic.
Fluorita De Mexico	P	Room and pillar	..do	Acid.
La Domincia	P	..do	..do	Do.
Las Cuevas	P	Shrinkage	..do	Acid, metallurgical.
Minas De Navidad	P	..do	..do	Do.
San Francisco del Oro	P	..do	..do	Acid.
Zinc de Mexico	P	Hydraulicking	..do	Do.
Morocco: El Hammam	P	Shrinkage	..do	Do.
Namibia: Okorusu	N	Open pit	..do	Do.
South Africa, Republic of:				
Buffalo Fluorspar	P	..do	..do	Acid, metallurgical, ceramic.
Kruidfontein	N	..do	..do	Do.
Marico Fluorspar[3]	PP	..do	..do	Acid.
Transvaal Flurospar	N	Room and pillar	..do	Do.
Vergenoeg	P	Open pit	..do	Do.
Witkop	P	..do	..do	Do.
Spain:				
Fluoruros	P	Open pit, room and pillar.	..do	Acid, metallurgical, ceramic.
Gijon Area	P	Room and pillar	..do	Acid, ceramic.
Mina Ana, Torre mill	P	Room and pillar, open pit.	..do	Do.
Minas de Orgiva	P	Room and pillar	..do	Acid, ceramic, lead.
Thailand:				
Ban Lard mill	P	Shrinkage, open pit	..do	Acid.
Mae La Luang	P	Open stope	Sizing	Metallurgical.
Mae Tha District	P	Open pit	Heavy media	Do.
Phanom Thuan District	P	..do	Sizing	Acid feed[4], metallurgical.
Salak Pra	P	Open pit, open stope	Hand sorting	Do.
SK Minerals	P	Open pit	Flotation	Acid.
Takien Ngam	P	..do	Hand sorting	Metallurgical.
Tunisia: Hammam Zriba	P	Room and pillar	Flotation	Acid.
United Kingdom:				
Blackdene mill and mines	P	Shrinkage	..do	Acid, lead.
Broadwood mill and mines	P	..do	..do	Acid, metallurgical, lead.
Derbyshire deposits (Laporte)	P	Sublevel stoping	..do	Acid, barite, aggregate.
United States:				
Annabell Lee	P	Shrinkage	..do	Acid, zinc.
Denton	P	Room and pillar	..do	Do.
Nye Crowell	P	Sublevel	Sizing	Metallurgical.
Piasano	P	Open pit	..do	Do.

[1] N, nonproducer; P, producer; PP, past producer.
[2] Closed in 1984, not included in availability analyses.
[3] Closed in 1983, not included in availability analyses.
[4] Acid feed at approximately 55 pct CaF_2 is sold to Thai Fluorite Processing Co. for its Ban Lard mill.

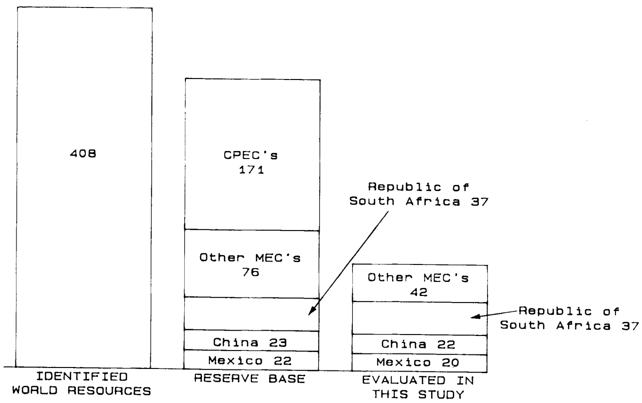

Figure 1.—Estimates of world fluorspar resources (million metric tons).

properties. However, plans are being discussed to reopen the Babb-Barnes fluorspar mill complex and several small local mines in Kentucky. This small operation was not included in this analysis because of the tentative nature of the proposed operation.

Two other properties included in the resource table were not included in the analyses. These past producers include Marico Flurospar in the Republic of South Africa, and Rio Colorado in Mexico, both closed for economic reasons. Although the potential exists for future production from these properties, it is unlikely in the next few years because of the downturn in demand for fluorspar on world markets. Current producing operations can easily supply present needs.

FLUORINE FROM PHOSPHATE

In addition to primary fluorspar resources, fluorine contained in phosphate rock is another important resource. Fluosilicic acid resulting from the processing of phosphate rock is used in the production of aluminum fluoride, in water fluoridation, and in a number of other chemical applications. Fourteen U.S. plants processing phosphate rock for the production of phosphoric acid and two plants producing HF sold or used about 75,000 mt of byproduct fluosilicic acid in 1985 (*1*).

According to the Bureau's availability appraisal of phosphate rock, approximately 6.6 billion mt of potentially recoverable phosphate rock was available from currently producing mines at total costs ranging up to \$30/mt in January 1985 dollars (see the phosphate chapter). Based on an assumed average fluorine content of 3 pct and a processing recovery of 30 pct, this represents approximately 59.4 million mt of CaF_2 equivalent potentially available. At costs up to \$50/mt, a total of 11.9 billion mt phosphate rock (107 million mt CaF_2 equivalent) was potentially available. Although technology exists to recover fluosilicic acid, not all operations find it economically viable to do so. Therefore, the 59.4 million mt of CaF_2 equivalent is a high estimate of the potential resources available from currently producing phosphate operations. However, fluorine from phosphate was not evaluated in this analysis.

U.S. AND WORLD HISTORICAL PRODUCTION

World fluorspar production for the 1950-85 period, in 5-yr intervals, is shown in figure 2. The 27 countries reporting production in 1985 produced an estimated 4.8 million mt of fluorspar. The 1985 estimated production from the evaluated MEC's and China was about 65 pct (3.1 million mt) of the world total, and production from these countries has averaged about 67 pct of the world total over the last 5 yr. Annual production from the other (not evaluated) MEC's has averaged 2 pct, and the remaining 31 pct was produced primarily by the U.S.S.R. and Mongolia, with lesser amounts from other centrally planned economy countries (CPEC's). Since 1950, U.S. production has declined, but all other countries experienced increases up through the late 1970's.

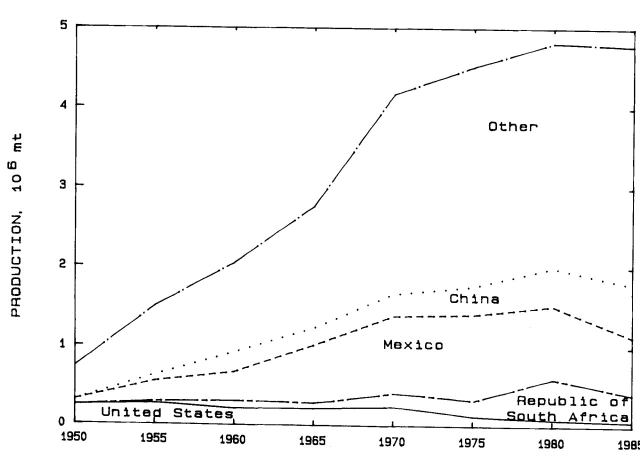

Figure 2.—World fluorspar production, 1950-85.

The world recession of the early 1980's was accompanied by steel mill closures, and cutbacks in production of aluminum and HF. Environmental concerns (fluorine contamination) and cost-saving measures in the steel and aluminum industries are also taking their toll on the fluorspar industry as consumers are developing processes that require less fluorspar. The decrease in demand from the steel and aluminum industries is reflected best in the drop in production in Mexico and the Republic of South Africa. Production of acid and ceramic grades is expected to pick up slightly in the late 1980's, but metallurgical grades may never recover their previous high levels.

The apparent increasing percentage of world production from the "other" countries is somewhat misleading. CPEC's make up an increasing share of the "other" category and many of these countries do not report annual production figures to the Bureau, thus the production data may be estimates based upon historical trends and assumed capacities. The assumed growth in these countries does not take into account the overall world decline in fluorspar production, which may have affected the CPEC's as well. The countries in this evaluation that are sheltered from world markets (i.e., they produce primarily for domestic consumption) have not been immune to recession and declines in consumptive industries such as steel and aluminum. They may have suffered less severe cutbacks than exporting countries, but none the less, they have suffered cutbacks.

Recent information from Czechoslovakia indicates that the historic reported production and capacity figures for that country are almost double current actual levels. The acid-grade concentrates produced in Czechoslovakia are all targeted for consumption by the U.S.S.R. This information, and exporting information obtained during site visits to other countries, indicates that the U.S.S.R. is not self-sufficient in fluorspar. Operations evaluated in Thailand and China are known to export fluorspar to the U.S.S.R.

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

Mining methods at the deposits evaluated vary according to geologic conditions. Table 2 lists the evaluated deposits and the mining and beneficiation processes used at each. Deep deposits usually require underground techniques, while wide, shallow deposits employ open pit methods. If ground conditions are unable to support underground mining, open pit methods may be used even though overburden removal might be substantial. In some cases, open pit methods are used until the depth or costs require moving underground.

Narrow vein mining is often done by shrinkage stoping and open stoping methods where strong walls are present, while stratiform or bedded deposits use room-and-pillar systems. Replacement and fissure vein deposits are mined with shrinkage stoping or cut-and-fill methods if they are deep, narrow occurrences. They may also be mined by open pit or strip mining techniques where they are near the surface and have competent sidewalls. The replacement and stockwork deposits in the Republic of South Africa are all mined by open pit methods, as are most of the fissure veins in Thailand. However, replacement deposits in Mexico are extracted by stoping or cut-and-fill methods.

PROCESSING

Most fluorspar must be upgraded to meet market requirements, however, high-grade deposits in China, Mexico, and Thailand, and even the United States, produce a metallurgical-grade gravel by hand sorting and sizing only. Lower grade ore must be processed by gravity, heavy media separation, or flotation methods to become marketable.

Gravity separation is used for ores with relatively coarse interlocking minerals. Heavy media cone and drum separators are used on finer ores to produce metallurgical gravel or for preconcentrating ore prior to flotation. Heavy media processing can preconcentrate ore as low as 14 pct CaF_2 to yield a flotation feed of 40 pct CaF_2, or more *(2)*. Other minerals, such as lead, zinc, and barite, are concentrated with the fluorspar through preconcentration and can then be recovered through flotation.

Acid and ceramic grades of fluorspar are produced by froth flotation processes. The ore is first crushed and ground to size. Then, if present in recoverable quantities, lead and zinc sulfides are floated off, with the lead first, followed by the zinc. All the easy-floating fluorspar is removed in the next step through a rougher flotation circuit, and then sent to the cleaner circuit; the rougher tailing may be further treated in scavenger cells or discarded. The middling product may be reground and sent to one or more cleaning circuits to recover more finely interlocked grains of fluorspar and gangue.

Acid-grade fluorspar is normally sold with a moisture content of 8 to 10 pct, because it is easier to load and unload and dust losses are minimized. Some products may be dried in a furnace and either shipped in covered hopper cars or tank trucks or bagged for shipment, depending upon the needs of the customer. Ceramic grades are normally marketed in bags, but dried products may be shipped to large consumers in railcars. Metallurgical-grade fluorspar is usually shipped as lump or gravel products, and generally is transported in barges, ships, or railcars.

NEW TECHNOLOGY

The Bureau has been testing new methods of flotation processing on ore from several of the western U.S. fluorspar deposits. A new column method of flotation appears to be successful in producing acid-grade concentrates and by-product beryllium from ores of the Fish Creek deposit in Nevada, and possibly could be used to produce a metallurgical-grade product from the Bayhorse ores *(8)*. Transportation costs could keep these particular deposits from being exploited, but perhaps the new flotation technology will benefit other operations.

PRODUCTION COSTS

A weighted-average total cost of fluorspar production was determined for each operation. Total fluorspar revenues were calculated by taking the total property revenues determined and subtracting all byproduct revenues (from nonfluorspar products). The remaining revenues (total fluorspar revenues) for each operation were then divided by the total tonnage of recoverable acid-, metallurgical-, and/or ceramic-grade fluorspar to provide a weighted-average total cost of fluorspar production, by operation. This evaluation allows for comparisons of fluorspar production costs to be made between selected operations and weighted-averages to be compiled for country comparisons.

Table 3 lists production costs for selected producing operations by country. Weighted-average production costs at both a 0- and 15-pct DCFROR are represented. The United States and several other countries are not included in order to protect individual company proprietary data, or by request. Figure 3 illustrates the production cost breakdown. The taxes and royalties shown in the figure are the amounts determined at the 15-pct DCFROR.

Table 3.—Fluorspar production costs for producing mines in selected countries

(All costs are in January 1985 U.S. dollars per metric ton of fluorspar concentrate)

Country	Operating cost		Transportation,	Net	Recovery	0-pct DCFROR		15-pct DCFROR		
	Mining	Milling	mill to port	operating cost	of capital[1]	Taxes and royalties[2]	Total cost[3]	Taxes and royalties[4]	Return on investment[5]	Total cost[6]
China	6.92	8.46	9.39	24.77	1.16	1.23	27.16	4.29	2.50	32.72
France[7]	38.90	22.97	12.31	74.18	7.54	0.61	82.33	4.31	4.37	90.40
Mexico	21.87	13.53	17.18	52.58	6.49	3.31	62.38	8.24	5.00	72.31
South Africa, Republic of	17.82	27.50	17.66	62.98	3.97	0.01	66.96	1.68	2.33	70.96
Spain	52.41	27.69	4.52	84.62	5.58	0.23	90.43	1.54	11.40	103.14
Thailand[8]	24.12	1.36	15.58	41.06	4.53	3.64	49.05	8.11	7.28	60.98

[1] Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure as of January 1985, and investments required over the life of the operation.
[2] Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 0-pct DCFROR.
[3] Equal to the sum of net operating costs, taxation, and capital recovery determined at a 0-pct DCFROR.
[4] Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 15-pct DCFROR.
[5] The revenue increase necessary per metric ton to obtain a 15-pct DCFROR.
[6] Equal to the sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery *plus* the per metric ton increase necessary to provide a 15-pct DCFROR after taxation.
[7] Costs represent only acid-grade operations, not Rossignol (metallurgical only).
[8] Costs represent metallurgical-grade production only.

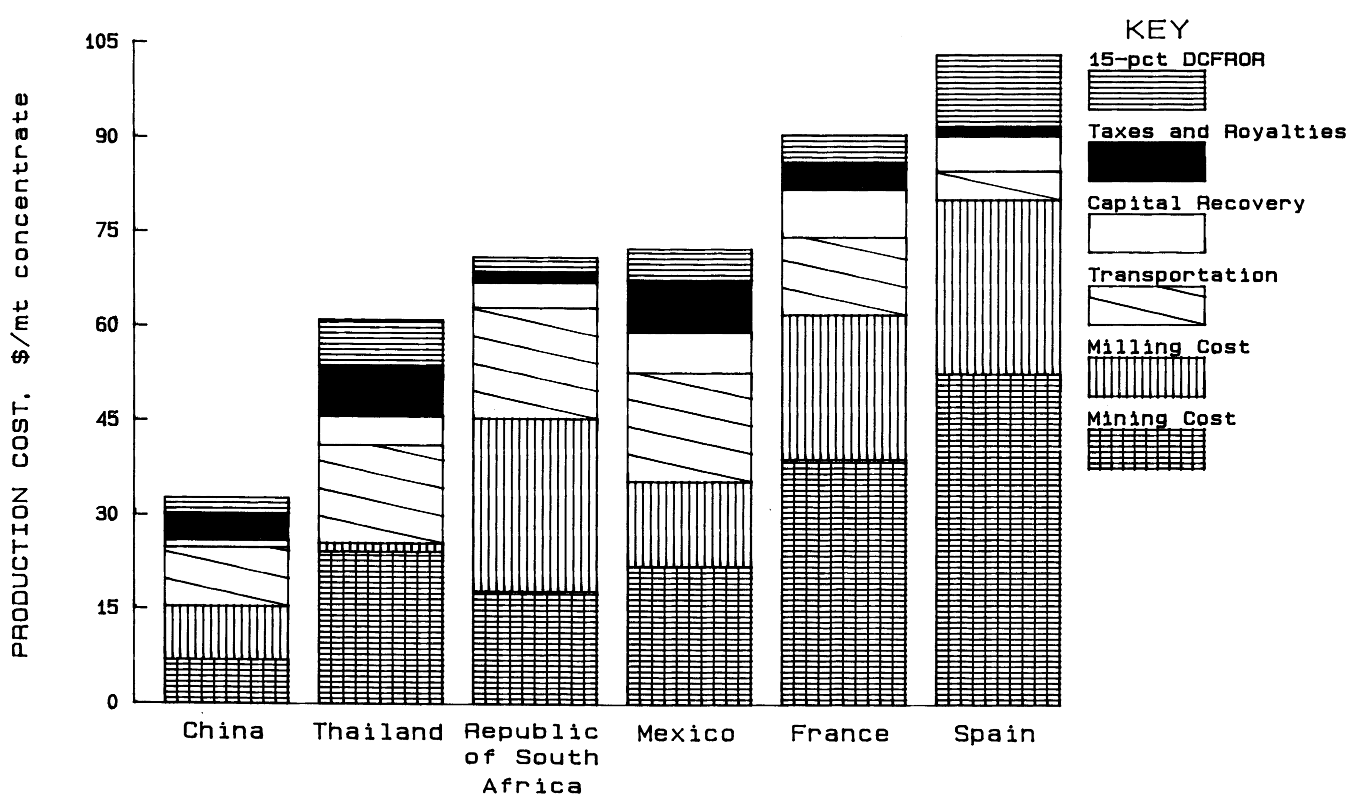

Figure 3.—Fluorspar production costs for producing mines in selected countries (January 1985 U.S. dollars).

Mining costs vary widely among mining methods and deposit types, and make up 20 to 60 pct of total costs. A room-and-pillar mine in a stratiform deposit may have costs as low as $6.50/mt to as high as $16/mt of ore mined. However, room-and-pillar mining costs generally run between 30 and 45 pct of total costs per metric ton of fluorspar concentrate. Economies of scale give an advantage to the large, low-grade open pit mines of the Republic of South Africa. Stripping ratios (waste to ore) also impact mining costs, particularly for Europe's surface mines. Operations moving large tonnage of waste per metric ton of ore are at an obvious disadvantage, no matter what the final ore grade may be.

Milling costs are fairly uniform for flotation methods, while costs for heavy media, gravity separation, and screening methods are much lower. Processing costs make up less than 20 pct of the total costs at operations not employing flotation methods, but 20 to 40 pct of the total cost if flotation is utilized. As shown in table 3, milling costs for China and Thailand, and Mexico to a lesser degree, reflect the large percentage of hand sorting and screening methods utilized to produce metallurgical-grade fluorspar.

Transportation costs are usually less than 30 pct of total costs. However, transportation costs may be as high as 40 pct for operations utilizing truck and/or rail for haulage over long distances to market or port areas. Spain, with its operations located in coastal areas, was the only country where transportation costs were not a signficant portion of the total cost.

Recovery of capital for exploration, acquisition, development, mine plant and mine equipment, constructing and equipping the mill, and infrastructure is shown separately on table 3. The large surface operations in the Republic of South Africa and China benefit from lower per-ton investment amounts than their underground counterparts in Europe and Mexico.

All Federal, State, property, and severance taxes and all royalties were calculated for each property. Revenue-generated taxes and royalties are noticeably higher at the 15-pct-DCFROR level because of the higher revenues generated to achieve this rate of return.

China had the lowest average total cost of production of the countries evaluated. This was directly related to low capital and operating costs associated with the high-grade deposits. Thailand had the next lowest cost operations, but the costs represent only the metallurgical-grade production, which is primarily hand sorted and crushed, so milling costs are negligible. France and Spain experienced the highest average total cost of production.

The per-ton amount of return on investment needed to achieve a 15-pct DCFROR is lowest for China and the Republic of South Africa where costs can be expensed immediately rather than amortized over time. Operations with higher return on investment requirements not only experienced higher capital costs, but often had shorter life spans over which to recover the costs.

AVAILABILITY

The 1985 published market prices for fluorspar products are listed in table 4. These prices can be used for comparison with the average total cost of production shown in this chapter. It should be kept in mind, however, that these are current published prices, and the production costs represent the longrun total cost over the life of the operations.

Acid-grade fluorspar was recovered from 45 of the 52 fluorspar mines and deposits included in this analysis. A total of 30 operations recovered a single fluorspar product; 23 of these recovered only acid grades, and 7 recovered only metallurgical grades. The remaining 22 properties were divided as follows: 14 recovered acid and metallurgical grades; 4 recovered acid and ceramic grades; and 4 recovered all three grades.

The availability of ceramic-grade fluorspar is highly variable, and none of the operations evaluated produce only ceramic-grade products. Ceramic-grade fluorspar is normally produced according to consumer needs and specifications, and because of the closeness in CaF_2 grades, is nearly always produced along with acid-grade concentrates. For this evaluation, ceramic-grade fluorspar was recovered from nine operations that regularly produced ceramic-grade fluorspar at the time of the site visits and were expected to do so on a continuing basis in the foreseeable future. Many other acid-grade operations can and do produce small amounts of ceramic grades as consumers request.

Table 4.—Fluorspar grades and 1985 market prices (*7, 9*)

Country and Product	CaF_2 grade, pct	Comments	Price, $/mt
China:[1]			
Acid, dry basis	97	f.o.b. China	110
Metallurgical	[2]70	..do	55
Mexico:			
Acid, dry basis, filter cake	+97	Tampico, f.o.b. vessel.	108
Ceramic	94-95	Mexican border, f.o.b. cars.	103
Metallurgical	[2]72.5	Tampico, f.o.b. vessel.	80
Northern Europe:			
Acid, dry bulk	97	f.o.b. northern Europe.	120
South Africa, Republic of:			
Acid, dry basis	97	f.o.b. Durban	110
United Kingdom:			
Acid, dry basis	97	Dried, bulk delivered tankers.	143
Metallurgical	[2]70	Ex-UK mine	70
United States:			
Acid, dry basis	97	Carload	171
		Pellets	170
Ceramic, No. 1	95-96	Calcite and silica variable.	165
Metallurgical	[2]70	Pellets	125

[1] Data supplied by R.B. Fulton III.
[2] Effective percentage rating for metallurgical-grade fluorspar defined as percent CaF_2 minus 2.5 times percent SiO_2.

TOTAL RECOVERABLE

Figure 4 shows the total availability of acid-, metallurgical-, and ceramic-grade fluorspar for the properties evaluated. The solid line represents the average total cost required over the life of the operation to meet all costs at breakeven, or 0-pct DCFROR. The broken line represents the average total cost of production and includes a 15-pct DCFROR on invested capital. As illustrated, the costs depicting the 15-pct DCFROR increase more towards the high end of the curve, where undeveloped and capital intensive operations are more prevalent.

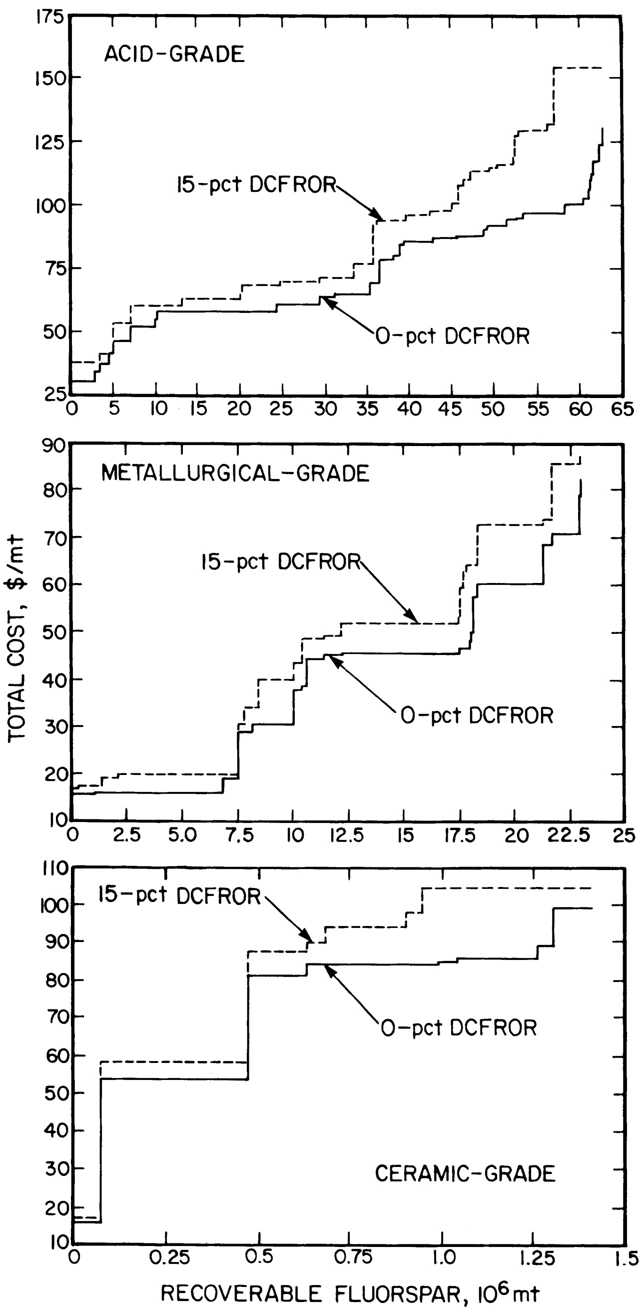

Figure 4.—Potential total acid-, metallurgical-, and ceramic-grade fluorspar availability (January 1985 U.S. dollars).

During 1985, the average market price for acid-grade fluorspar was around \$110/mt. Approximately 75 pct (47 million mt) of acid-grade fluorspar was potentially available at an average total cost below \$110/mt, including a 15-pct DCFROR.

Market prices for metallurgical-grades were averaging between \$60/mt and \$80/mt during 1985, and almost 94 pct of the evaluated tonnage recovered was potentially available below \$75/mt, including a 15-pct DCFROR. About 77 pct (17.9 million mt) could potentially be produced for less than \$60/mt.

The 1985 prices for ceramic-grade fluorspar were reported at \$103/mt from Mexico and \$165/mt in the United States. At a 15-pct DCFROR, virtually all of the evaluated ceramic-grade fluorspar was potentially available for \$103/mt or less. About 67 pct was potentially available for less than \$100/mt.

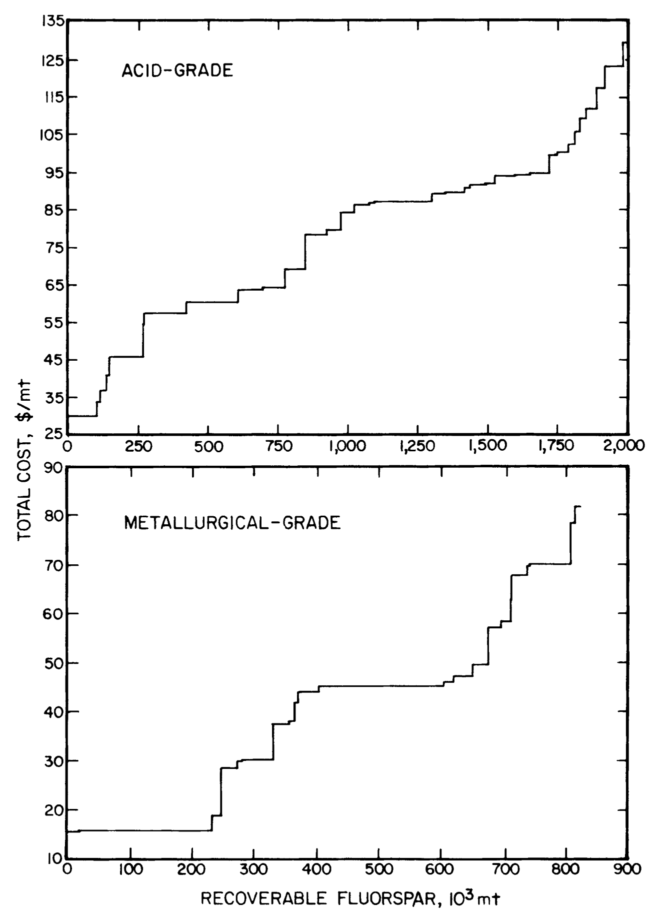

Figure 5.—1985 acid- and metallurgical-grade fluorspar capacities from producing operations in MEC's (January 1985 U.S. dollars).

ANNUAL CAPACITY

The 1985 production potential of acid- and metallurgical-grade fluorspar from producing mines is illustrated in figure 5. Approximately 1.8 million mt (93 pct) of acid-grade fluorspar was potentially available below $110/mt, indicating that more producers were able to meet the break-even (0 pct) cost of production at current prices, but not all had much in the way of a profit margin. The same can be said for the metallurgical-grade producers, nearly 87 pct (710,000 mt) was potentially available below $60/mt. Ceramic-grade fluorspar was not represented in these curves in order to protect confidentiality.

Figure 6 compares the potential annual availability of acid-grade fluorspar from producing mines in various countries through the year 1995. The majority of 1985 potential production comes from three countries, including Mexico with 27 pct, the Republic of South Africa with 20 pct, and China with 10 pct of the 2 million mt of potential production. The "other MEC's" production curve includes the remaining 43 pct of production potential. Production drops off as a result of poorly defined resources in Thailand, and small vein operations in Europe and the United States. The drop in these curves should not be taken as an indication of a decline in the availability of acid-grade fluorspar; it simply represents a static view from 1985. The short-term

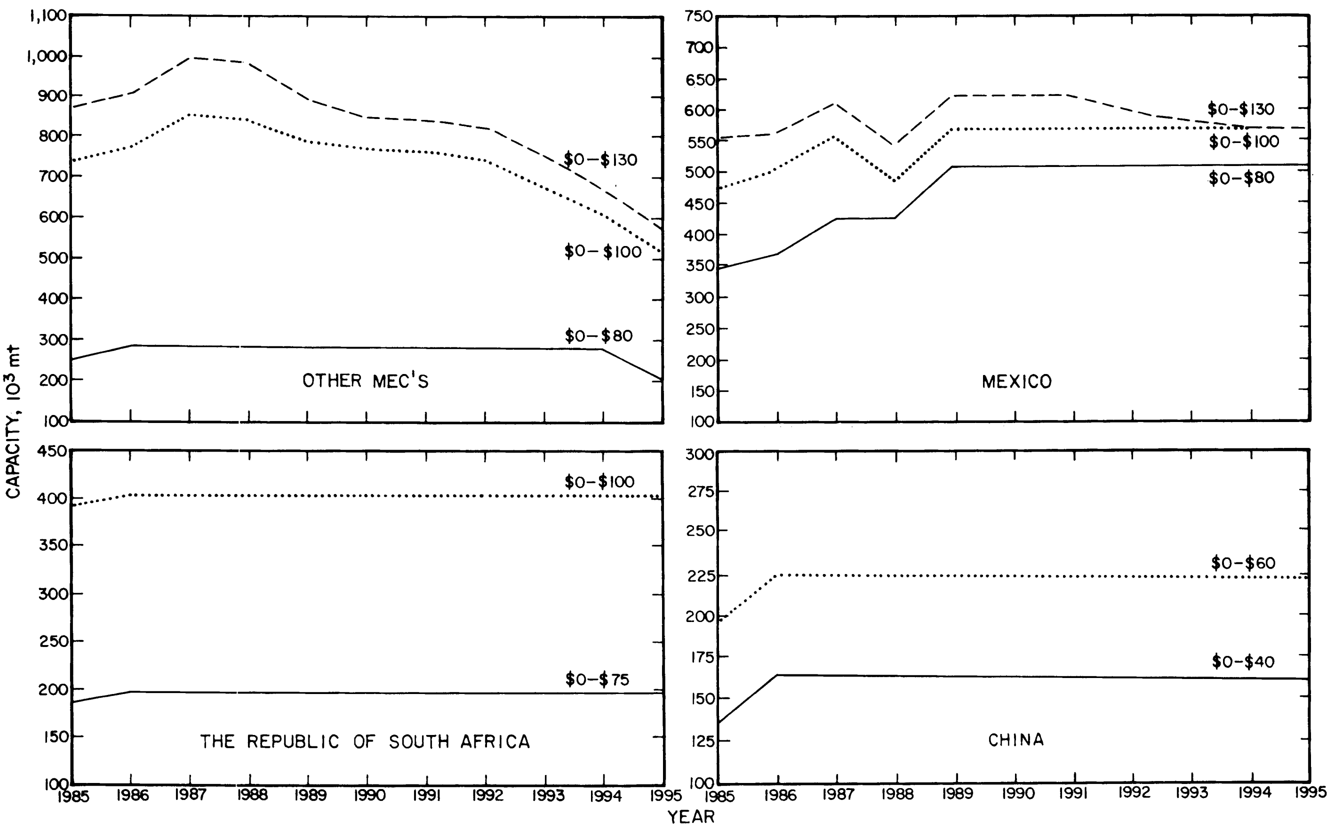

Figure 6.—Potential annual acid-grade fluorspar production from producing mines in various countries at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

demonstrated reserves presently defined in several of the countries could be replaced by currently known undeveloped resources.

Mexico's production drops slightly about 1988, as presently defined resources for the Zinc de Mexico tailings reprocessing operation become depleted. The increase occurring in 1989 represents achievement of full proposed capacity at several of the operations, most notably, Las Cuevas. This optimistic view of potential production levels assumes world consumption levels will return to higher levels.

Production through 1995 remains fairly constant in the Republic of South Africa. Operators have been producing at reduced levels over the last few years, and are just now returning to full capacity. All three producers are in a position to increase production capacities in the future, should demand warrant it.

In China, the production increase of 1986 indicates the assumed resumption of higher production levels at several of the operations. Production levels remain constant through 1995, but these levels are not necessarily maximum capacity. Many of the operations could increase production with minor expense or delay. The Fu Shan and Xian Shan operations, for example, have converted old copper and iron flotation mills for fluorspar production and utilize only partial capacity at the present time.

Potential annual availability of metallurgical-grade fluorspar from 1985 to 1995 is shown for various regions in figure 7. China represents 41 pct of the total 817,000 mt produced in 1985, with Mexico accounting for 33 pct. Increases in Mexican production are the result of assumed resumption of production levels and achievement of proposed expansions. The present outlook for the steel industry, the primary consumer of metallurgical-grade fluorspar, is not good. Thus, the proposed increases may take much longer to occur, if at all.

Most of the Chinese production is recovered through hand sorting and crushing, so changes in production needs are easily accommodated. The decline in "other MEC" production is strongly influenced by the small resources defined for Thailand. The definition of additional resources could stabilize production levels for these countries.

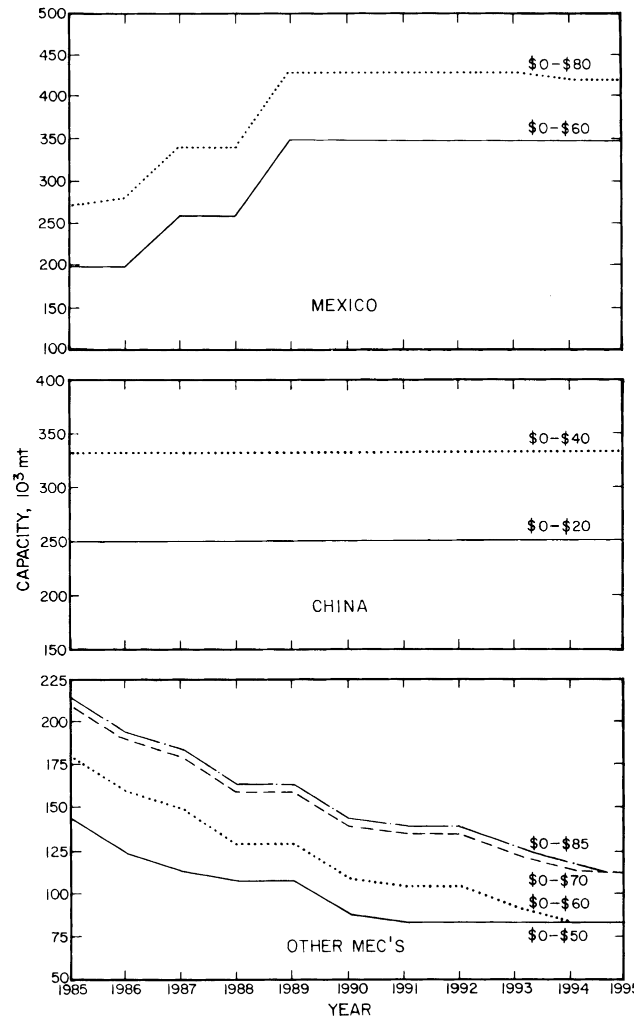

Figure 7.—Potential annual metallurgical-grade fluorspar production from producing mines in various countries at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

CONCLUSIONS

Fluorspar prices peaked in 1982 and then dropped significantly as reduced demand and oversupply slowed the market. Consumption dropped drastically in 1982 and 1983, but recovered slightly in 1984-85.

The threat of supply interruptions, although inconvenient, would not put many fluorspar consumers at much of a disadvantage because fluorspar is available from a number of different countries. Mexico, the Republic of South Africa, and China are the three largest producers, with additional sources from other African and European countries. As Mexico found out when it kept its prices high, other nations eagerly filled orders for Mexico's previous customers. It could be assumed that similar arrangements would be made if fluorspar was suddenly not available from any one country. Production capacities at some operations may need to be increased to meet additional demands, but since many operations are currently producing at less than capacity, this would not necessarily be a lengthy or costly task.

Most operations are producing at greatly reduced levels as a result of the worldwide slowdown in the industries consuming fluorspar, principally in the steel and aluminum industries and chemical manufacturing. Although the aluminum and chemical (primarily HF) industries are showing signs of recovery, the demand for fluorspar in the steel industry does not look promising. The steel industry has been slow to recover, and steel producers are consuming less and less fluorspar per metric ton of steel. If these trends continue, the fluorspar industry will need to find alternative markets for its products currently targeted for metallurgical consumption. In addition, the industry needs to improve and expand its markets for higher grades of fluorspar if higher production levels and improved earnings are to be achieved.

REFERENCES

1. Pelham, L. Fluorspar. Ch. in BuMines Minerals Yearbook 1985, v. 1, pp. 419-428.

2. Fulton, R.B. III, and G. Montgomery. Fluorspar and Cryolite. Ch. in Industrial Minerals and Rocks (Nonmetallics Other Than Fuels). Soc. Min. Eng. AIME, 5th ed., 1983, pp. 723-744.

3. U.S. Department of Commerce, Federal Emergency Management Agency. Stockpile Specifications, Fluorspar, Metallurgical-Grade. May 1984, 1 p.

4. Kilgore, C.C., S.R. Kraemer, and J.A. Bekkala. Fluorspar Availability—Market Economy Countries and China. A Minerals Availability Appraisal. BuMines IC 9060, 1985, 57 pp.

5. Worl, R.G., R.E. Van Alstine, and D.R. Shawe. Fluorine. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 223-235.

6. Pelham, L. Fluorspar. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 52-53.

7. Engineering and Mining Journal. Markets, Nonmetallic Minerals. v. 186, No. 2, Feb. 1985, p. 13

8. Foot, D.G. Jr., F.W. Benn, and J.L. Huiatt. Recovery of Fluorite and Byproducts From the Fish Creek Deposit, Eureka County, Nevada. Paper in Physical Chemistry of Extractive Metallurgy, ed. by V. Kudryk and Y.K. Rao. Metall. Soc. AIME, Feb. 1985, pp. 251-262.

9. Industrial Minerals. Prices. No. 209, Feb. 1985, p. 74.

GOLD

INTRODUCTION

Gold is a unique element in that it is, above all, an alternative store of wealth to fiat currencies, most notably the U.S. dollar. The U.S. dollar serves as the primary medium of exchange in international transactions and is the currency in which world gold prices are denominated. The great majority of gold is held either as an investment medium or as a form of insurance to hedge against uncertainty about the value of fiat currency or potential disaster. Only a small percentage of the world's total gold production has been consumed in industrial applications, and much of that metal will eventually be recycled.

In this chapter, gold is defined as a metal product of at least 99.6 pct purity. Following the most prevalent world industry practice, gold product quantities are reported in troy ounces (31.1035 g/tr oz) and gold grades are reported in grams per metric ton. Quantities of gold-bearing ore and contained gold are reported on a mill feed basis (which includes allowances for mining recovery and dilution) or as recoverable gold (i.e., the amount of gold available on a postrefining basis).

Most of the information presented in this chapter is an updated summary of Bureau of Mines Information Circular 9070 "Gold Availability—World. A Minerals Availability Appraisal" (1).[1] This chapter represents evaluations of 110 of the most significant producing operations (circa 1982) in the market economy countries (MEC's). It is indicated that these properties still form the core of the MEC's gold mining industry as of 1985.

GEOLOGY AND RESOURCES

GEOLOGY

The world's primary gold deposits can be classified into four separate categories: placer deposits, lode deposits, blanket (or reef-type) deposits, and disseminated deposits.

Placer deposits are usually defined as flat-lying deposits composed of unconsolidated loose materials, such as gravels and sands, in which the gold particles occur as free particles ranging in size from coarse nuggets to grains to fine flakes. An important distinction is that a placer deposit represents material derived from the erosion and transport of the original country rock. Lode gold deposits, by contrast, consist of gold particles contained in the *in-place* quartz veins or country rock, and by definition, should be inclined from the horizontal. The blanket or reef-type deposits are deposits where the gold occurs in quartz conglomerates that have resulted from the consolidation of placer deposits and are inclined from the horizontal.

The newest classification (introduced since the mid- to late-1960's) is the disseminated gold deposits of which the Carlin, NV, deposit was the first example, and are found at present mostly in the southwestern United States (Nevada, Utah, New Mexico, and the southern portion of California) and in Western Australia. Three characteristics of the disseminated gold deposits serve to distinguish them from the other three categories. First, and of most significance, is that their gold mineralization is fairly evenly distributed throughout the deposit rather than being concentrated in veins (as in lode deposits) or in pay-streaks (as in placer deposits). Second, the deposit consists of *in-place* materials rather than transported materials and third, the disseminated deposits are flat lying.

Prior to the 1860's and 1870's, nearly all of the world's gold production came from the mining of placer deposits. Since that period, the reef-type deposits of the Witwatersrand Basin in the Republic of South Africa have dominated, with the result that these deposits have been responsible for an estimated 40 pct of all of the world's gold production through the year 1980. The disseminated deposits have seen production only since the 1965 startup of the Carlin operation. However, the disseminated deposits presently constitute the vast majority of production in the State of Nevada and the State of Western Australia, which are the regions with the largest production within their respective nations.

Table 1 summarizes the breakdown of these four classifications into geographic location in the major gold-producing countries and provinces, mill feed or recoverable grade data, and predominant type of mining presently used for extraction. Additional geologic information on world gold deposits can be found in U.S. Geological Survey Professional Paper 820 (*2*).

[1]Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

Table 1.—Geographic locations and classification of primary gold deposits

Classification, primary gold deposits	Major producing countries with production and/or explored deposits, circa 1985	Mill feed grade, tr oz	Type of mining
Placer	Colombia, Brazil, Ghana, Bolivia, States of Alaska and California in the United States.	[1]0.07– 0.75	Water-based surface methods: hydraulicking, dredging, and open pit mining, some significant underground mining in the U.S.S.R. (past and present), and an insignificant amount in California (past).
	U.S.S.R.	[2]1.8	
Lode	Canada, United States, U.S.S.R., Australia, Brazil, Zimbabwe, Chile, Ghana, Republic of South Africa.	1.9 –21.1	Vast majority underground with many different methods. Some surface mining.
Blanket or reef-type	Republic of South Africa, Ghana, Brazil	3.5 –16.1	Underground.
Disseminated	United States, Australia, U.S.S.R.	.5 – 8.0	Surface.

[1]Large and small dredging operations, estimated recoverable grade, Alaska not represented.
[2]Underground placers; estimated recoverable grade.

RESOURCES

The world gold reserve base is estimated to be 1.45 billion tr oz of in situ gold. Of this total, 18 pct (0.26 billion tr oz) is estimated to be present in centrally planned economy countries (CPEC's), primarily in the U.S.S.R., China, and the Eastern Bloc countries, and 1.19 billion tr oz is contained in MEC's. In comparison, the total gold estimated to be recoverable from the demonstrated resources of 110 evaluated MEC properties that were in production as of January 1985 amounts to 0.792 billion tr oz, with the Republic of South Africa accounting for 87.5 pct of the total and the United States, Canada, and Australia for 3.9, 4.4, and 0.7 pct, respectively. The evaluated properties are listed in table 2 and include only MEC producers. Producing mines in the U.S.S.R., China, and the Eastern Bloc were not included owing to a lack of data. Demonstrated resource data by country are listed in table 3 and depicted graphically in figure 1.

Because the evaluated demonstrated resources shown in table 3 are composed only of the tonnage present in large-scale producers (greater than 20,000 tr oz/yr in all but a few cases), the January 1985 recoverable estimate should be considered conservative. Similarly, the January 1985 estimate reflects subtraction of estimated 1982, 1983, and 1984 production without adjusting for new resources that may have been discovered since the time of the original resource estimate (1982), which also adds to the conservative nature of the evaluated tonnage. In addition, the Republic of South Africa underground resource estimate includes only reef material grading in excess of 2.5 g/mt. Furthermore, the high level of gold exploration and development activity in Australia and the United States since 1982 results, respectively, in at least 18 and 29 mines or deposits of comparable size not being included in the January 1985 estimate. The majority of these 47 mines or deposits that were not included are, or will be, surface mining operations.

The estimate of recoverable gold in demonstrated resources for Australia and the United States is composed of about 25 pct surface minable material in Australia and about 75 pct surface minable material in the United States. By comparison, the vast majority of the resources of the Republic of South Africa and Canada are contained in large, underground operations. This distinction between surface minable and underground minable resources is important from a long-term availability standpoint since large underground operations tend to be longer lived. This difference primarily occurs because (1) surface operations are generally limited in terms of the depth of economic mining, and (2) heap leaching operations, which have increased their percentage contribution to production in the United States and Australia, are limited to treating oxide ores.

As noted, South African and Canadian gold mining industries are almost entirely dependent upon underground resources. In the Republic of South Africa, although the tonnage of surface waste material is large, the extremely low grade renders it a relatively insignificant resource. In Zimbabwe and Ghana, all evaluated resources are underground, as is roughly two-thirds of the evaluated resource in the Philippines.

The preceding discussion points out three basic caveats to the resource analysis that must be made to place the evaluated resources in proper perspective. They concern the following:

The dynamic nature of the world's gold mining industry.
The type of resource data that are reported.
The type of resource occurrence that accounts for production in an individual country.

First, the world's primary gold mining industry is highly dynamic. The rapidly rising, and still relatively high prices, of gold by comparison to the early 1970's has continued to generate tremendous exploration and development activity. This activity has been especially pronounced in the United States, Canada, and Australia. Deposit discoveries and new mine developments have been extremely numerous by historical standards, especially since the early 1980's. In addition to the 47 large-scale mines and deposits already

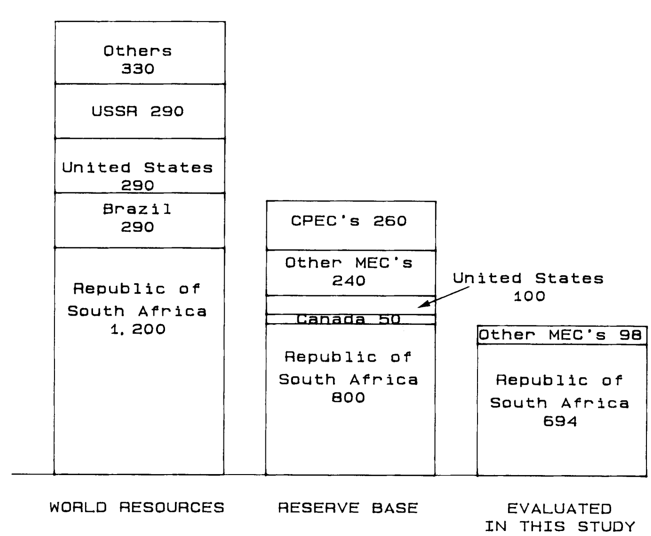

Figure 1.—Estimates of world gold resources (million troy ounces).

Table 2.—MEC gold properties included in this study

Country and property name	Ownership	Mining method[1]
Australia:		
Central Norseman	Western Mining and general public.	C/U
Fimiston Leases	Kalgoorlie Mining Associates	U
Hill 50, Morning Star	Hill 50 and Western Mining	U
Mount Morgan Tailings	Peko Wallsend Ltd	S/D
Mount Charlotte	Kalgoorlie Mining Associates	U
North Kalgurli	North Kalgurli Mines and Metals.	U
Telfer	Newmont Holdings and Dampier.	S
Bolivia: Teoponti	Compania Minera del Sur S.A.	S/D
Brazil:		
Cerra Pelada	CVRD, Brazilian Government	S
Morro Velho	Bozzano, Simonsen, Anglo American.	U
Canada:		
Agnico-Eagle (Gold)	Agnico-Eagle Mines Ltd	U
Camflo, Malarctic Hygrade	Camflo Mines Ltd	U
Campbell Red Lake	Campbell Red Lake, Dome Mines.	U
Con-Rycon	Cominco	U
Detour Lake	Various owners	U
Dickenson	Dickenson Mines Ltd	U
Dome	Dome Mines Ltd	U
Giant Yellowknife	Giant Yellowknife	U
Golden Giant (Hemlo)	Norado, Goliath, Golden Sceptre.	U
Kerr Addison	Kerr Addison Mines Ltd	U
Lac Minerals (Hemlo)	Lac Minerals	U
Ladner Creek	Carolin Mines, Aquarius Group	U
Lupin Project	Echo Bay Mines Ltd	U
Macassa (Willroy)	Willroy Mines Ltd	U
Pamour Porcupine	Pamour Porcupine and Noranda.	U
Sigma	Sigma Mines, Dome Mines	U
Teck, Corona (Hemlo)	International Corona, Teck Corp.	U
Chile: El Indio	St. Joe Minerals and private	U
Columbia:		
El Bagre	Mineros de Antioquia	S/D
La Salada	Sindico	U
Dominican Republic: Pueblo Viejo.	Rosario Dominicana S.A.	S
Ghana:		
Ashanti	Ghana Government and Lonrho.	U
Prestea	State Gold Mining Corp.	U
Tarlwa	..do	U
Philippines:		
Benguet (Gold)	Benguet Corp.	U
Masbate	Atlas Consolidated Mining & Development Corp.	S
Paracale	Philippine Eagle Mines Inc.	U
South Africa, Republic of:		
Beatrix	Gencor	U
Blyvooruitzicht	Barlow Rand	U
Bracken	Gencor	U
Buffelsfontein	General Mining Group	U
Consolidated Modderfontein	Consolidated Modderfontein	U
Deelkraal	Gold Fields	U
Doornfontein	..do	U
Dreifontein Consolidated	Dreifontein Consolidated	U
Durban Roodepoort Deep	Barlow Rand	U
E.T. Consolidated	Eastern Transvaal Consolidated.	U
East Rand Proprietary	East Rand Proprietary Mines	U
Egoli Consolidated (East)	Egoli Consolidated Mines	S
Egoli Consolidated (West)	..do	S
Elandsrand	Anglo American	U
Ergo (East Rand Gold and Uranium) Co.	..do	U
Fairview (Barberton)	General Mining	U
Free State Geduld	Anglo American	U
Grootvlei	Gencor	U
Harmony	Barlow Rand Group	U
Hartebeestfontein	Anglo-Transvaal Consolidated	U
Joint Metallurgical	Various mines	S
Kinross	Gencor	U
Kloof	Gold Fields	U
Leslie	Gencor	U
Libanon	Gold Fields	U
Loraine (Allanridge)	Anglo-Transvaal Group	U
President Brand	Anglo American	U
President Steyn, Video	..do	U
Randfontein Estates	Johannesburg Consolidated	U
RMMM Slimes Project	Rand Mines	S
Simmer and Jack	Anglo American	C/U
St. Helena	Gencor	U
South Africa, Republic of—Con.:		
Stilfontein	Gencor	C/U
Unisel	Union Corp.	U
Vaal Reefs	Vaal Reefs Exploration & Mining South Vaal Holdings.	U
Venterspost	Gold Fields	U
Village Main Reef	Village Main Reef Gold Mining	S
West Rand Consolidated	Gencor	U
Western Areas	Johannesburg Consolidated	U
Western Deep Levels	Western Deep Levels	U
Western Holdings Complex	Four combined owner-operations.	U
Winkelhaak	Gencor	U
Witwatersrand Nigel	African Exploration	U
Taiwan: Chin-Qua-Shih	Taiwan Metal Mining Corp.	U
United States:		
California:		
McLaughlin	Homestake Mining Co.	S
Yuba Placer	Yuba Placer Gold Co.	S/D
Colorado: Victor Project	Silver State Mining Corp.	S
Idaho: West End, Garnet Creek.	Pioneer Metals Corp., TRV Minerals.	S
Montana:		
Golden Sunlight	Placer U.S. Inc.	S
Zortman-Landusky	Pegasus Gold Inc.	S
Nevada:		
Alligator Ridge	Nerco Minerals, Amselco	S
Battle Mountain	Battle Mountain Gold Co.	S
Borealis	Tenneco Minerals	S
Carlin Operations	Newmont Gold Co.	S
Jerritt Canyon	Freeport McMoRan Gold Co., FMC Gold Inc.	S
Pinson, Preble, Ogee	Lacana, Rayrock, Siscoe Metals.	S
Round Mountain	Echo Bay Inc., Homestake, Case.	S
Windfall	Western-Windfall Inc.	S
New Mexico: Ortiz	Gold Fields Mining Corp.	S
South Dakota: Homestake	Homestake Mining Co.	U
Utah: Mercur	Barrick Mercur Gold Miner Inc.	S
Zimbabwe:		
Arcturus	Lonrho	U
Athens	..do	U
Blanket	Falconbridge Nickel	U
Dalny	Falcon Mines Ltd.	U
How	Lonrho	U
Mazoe	..do	U
Muriel	..do	U
Old West, Redwing	..do	U
Patchway, Brompton	Rio Tinto (Zimbabwe)	U
Renco	..do	U
Shamva	Lonrho	U
Venice	Falcon Mines Ltd.	U

[1]S, surface; S/D, surface dredge; U, underground; C/U, combined, primarily underground.

Table 3.—Summary of MEC demonstrated gold resources evaluated for this study as of January 1985

Country	Recoverable ore, 10^6 mt	Gold grade, g/mt mill feed basis[1]	Gold, 10^3 tr oz Contained	Recoverable
Australia	62	[2]5.3	6,997	5,910
Canada	178	6.8	38,808	35,445
Ghana	20	10.0	6,556	5,628
Philippines	31	3.9	3,875	3,231
South Africa, Rep. of:				
Underground mines	2,879	7.7	712,727	680,257
Surface reprocessing	2,008	.3	18,722	13,759
United States[3]	686	2.0	43,220	30,844
Zimbabwe	14	7.2	3,139	2,736
Other MEC's[4]	NM	NM	NM	14,501
Total	NM	NM	NM	792,312

NM Not meaningful.
[1]Weighted-average basis and rounded.
[2]Excludes 1 large, low-grade tailings reprocessing operation that would bias the estimate because it is not reflective of demonstrated resources in general.
[3]Includes 1 large underground operation along with the 16 surface operations to avoid possible disclosure of company proprietary data.
[4]Includes Brazil, Bolivia, Colombia, Dominican Republic, Chile, and Taiwan. Total ore tonnage, contained gold, and weighted-average grade data are not meaningful owing to the wide variation in these data between countries and the significant differences in gold recovery.

mentioned for Australia and the United States, an additional 100 or so new mines or explored deposits in MEC's could be added to the analysis, although many of these are not sufficiently explored to warrant classifying their resource tonnages at the demonstrated level. Thus, any overall resource total for a country can become quickly outdated, especially in this period of expanded exploration and development of new gold deposits.

Second, the type of resource data that are reported often vary owing not only to government policies but also to a reluctance on the part of many mining companies to divulge data on their gold operations. This reluctance is due to a number of factors. Among them is the desire to minimize or avoid taxation in localities that tax the value of unmined gold reserves. Also, most publicly held mining companies are required to report their reserves (as opposed to resources) annually to various agencies and institutions. For valid reasons, these annual reported reserves are usually defined as that ore which has been developed on at least three sides and assayed as thoroughly as required. These proven reserves are redefined each year based on production, new development, new assays, changes in prices and costs and, possibly, changes in technology. These proven reserves are justifiably conservative because of the preceding reasons as well as the fact that many vein-type gold mines are geologically erratic, which makes reasonable inferences of resources difficult or perhaps impossible. Also, exploration work to delineate resources is a costly and time-consuming endeavor and cannot be justified by small operators that have been in production for a long time. For example, some mines in Canada have annually reported proven reserves sufficient for only 1 to 5 yr of production and have been doing so for 20 yr or longer. Most gold mining operations will not estimate beyond the proven reserve level without a very good reason, such as when formulating plans for major capital investments.

The third caveat deals with the type of resource occurrence in an area or country. Some countries, such as Brazil, Colombia, and Bolivia, and the State of Alaska in North America, have the great majority of their gold production emanate from placer deposits. These operations are generally small scale, intermittent, and nearly impossible in most cases to estimate for contained resources, much less costs of production. In the case of Brazil especially, most production emanates from tens of thousands of individuals or small groups mining placer deposits in the Amazon Basin. The government of Brazil does not really know how much gold is produced or smuggled out of the country. Brazil undoubtedly has large gold resources but measuring them is virtually impossible.

There are thus two forces at work in terms of demonstrated resource estimation. First, the largest and best established areas generally have the best estimation and reporting. Second, those countries or areas with geological occurrences that lend themselves relatively easily to estimation, such as the Witwatersrand Basin of the Republic of South Africa, have the best available data in terms of quality and quantity. In the case of the Republic of South Africa mining industry, the quality and quantity of reported data is unsurpassed. One key, therefore, to interpretation of *world* demonstrated gold resources is to know where it has *not* been measured or reported, in addition to knowing where it has been measured and reported. This study, by necessity, deals only with countries and areas where enough basic information is collected and reported so that it is possible to estimate demonstrated resources with some reasonable degree of confidence.

U.S. AND WORLD HISTORICAL PRODUCTION

Figure 2 shows gold production for the Republic of South Africa, the U.S.S.R., and the rest of the world in 5-yr intervals over the 1950-85 period. Current world production remains dominated by the Republic of South Africa, which produced an estimated 22.2 million tr oz in 1985 (*3*). The Republic of South Africa's production represented 62 pct of total MEC production and 47 pct of world production. The Soviet Union is the world's second largest producer with an estimated 1985 production level of 8.7 million tr oz, which represented 18.5 pct of world production in that year. All other gold-producing countries remain insignificant by comparison to the Republic of South Africa and the Soviet Union.

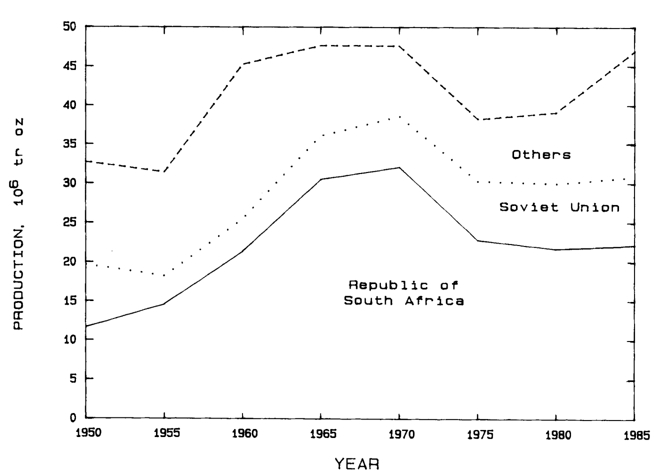

Figure 2.—World gold production, 1950-85.

In the aggregate, estimated 1985 world production is approximately 1.1 pct (0.5 million tr oz of annual production) less than 1970. The Republic of South Africa's 1985 production level represents a decline of 30.9 pct, or very close to 10 million tr oz of annual production, since 1970. This decline represents a lowering of the pay limits (lowest economically minable grade) to which the companies must mine by law and a comparable increase in tonnage milled in reaction to the higher gold prices that resulted from demonetization. In fact, by 1975, grade and tonnage adjustments had been made such that the last 11 yr have shown a steady level of production between about 21 and 23 million tr oz/yr.

By comparison, three major MEC producers (the United States, Canada, and Australia) had their lowest levels of annual production for the 1950-85 period occur during the years of 1979 and 1980. As of 1985, all three countries had dramatically reversed the 1970's production declines with the result that, combined in total, they are producing about 2 million tr oz more, on an annual basis, than in 1970. The U.S.S.R., China, and Brazil have also increased their combined annual production levels by close to 5.5 million tr oz/yr. Thus, these six countries have compensated for about 75 pct of the Republic of South Africa's production decline since 1970.

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

Table 4 presents a breakdown by country of surface and underground mining utilized to extract the demonstrated resources of the 110 evaluated producing MEC operations. As shown, entirely underground or predominantly underground methods are used at 81 of the 110 operations. Of the 29 surface mining operations, 4 are dredging operations, 6 are surface dump and tailings reprocessing operations in the Republic of South Africa, and 19 are conventional open-pit mines, 15 of which are located in the United States.

Table 4.—Country distribution of underground and surface minable resources

Country	100 underground or predominantly underground	100 pct surface: Open pit	Surface reprocess	Dredge	Total
Australia	5	1	0	1	7
Canada	17	0	0	0	17
South Africa, Republic of	37	0	6	0	43
United States	1	15	0	1	17
Others	21	3	0	2	26
Total	81	19	6	4	110

The surface mining operations contain only 46.7 million tr oz (5.9 pct) of the 792.3 million tr oz of total recoverable gold. The 15 U.S. surface operations account for 49.7 pct of this total and the 6 Republic of South Africa tailings and dump reprocessing operations account for another 29.7 pct. Only 7 of the 29 operations (all located in the United States) represent entirely heap leaching operations, and two others (both in the United States) were treating a portion of their total mill tonnage using heap leaching methods.

The 81 underground producers represent 745.6 million tr oz (94.1 pct) of total recoverable gold. Relative country percentages of total demonstrated resources contained in underground operations are 98 pct for the Republic of South Africa, 100 pct for Canada, 77.8 pct for Australia, 85.4 pct for the Philippines, 100 pct for Zimbabwe, 100 pct for Ghana, and approximately 25 pct for the United States. This underground resource distribution is reflected in the 1985 production share attributable to underground production in all countries except in Australia, where many open pit operations have been developed over the past few years, and in the Philippines, where much of the gold production is a byproduct of copper production or, more recently, from small-scale placer operations.

Of the 37 underground producers in the Republic of South Africa, 35 produce from Witwatersrand Basin paleoplacer gold deposits, and 2 produce from lode gold deposits in the archean greenstone gold district of the Eastern Transvaal. The Witwatersrand Basin gold mining operations are unique in comparison with other gold mining operations around the world. The mines are extremely large tonnage, multishaft operations that are highly labor intensive, yet highly mechanized. They mine to great vertical depths (over 3,500 m at some of the operations), have very high capital cost requirements, and some of the operations face technological hurdles related primarily to the requirement to mine from stopes ranging from only 0.9 to 2.3 m in height (an average of slightly over 1.4 m) at vertical depths of over 2 miles.

In Canada, the 17 predominantly underground producers are 11 mines that have been in production since before 1975 and 6 that have been brought into production since 1979. The six new operations include the three large Hemlo District operations that were not technically in full production until sometime in 1986. The mining methods in use at the 17 operations show a wide variation from high-cost, cut-and-fill or narrow-vein shrinkage methods to low-cost, long-hole open-stoping methods.

Most of the six Canadian operations that have been brought into production since 1979 or were in development as of 1985, either utilize or plan to utilize low-cost underground mining methods such as blasthole or long-hole open stoping. This is partly a reflection of the increase in the price of gold that has lead to the economic extraction of lower grade ore bodies, and partly a reflection of the discovery of a new type of ore body, the Hemlo type, which has very large thicknesses (for the gold grades obtained) that range from 3 to 30 m, and that can be mined utilizing low-cost, high-tonnage underground methods. Currently, the depth at which the 17 underground mines are operating does not appear to have a major influence on their overall cost of production.

PROCESSING

There are a variety of beneficiation methods represented by the analyzed properties. This section briefly summarizes beneficiation methods at the 84 South African, Canadian, Australian, and U.S. producers included in the analysis.

In the Republic of South Africa, the 43 operations represent a total of 79 individual mills, 19 of which were being utilized in the mid-1980's for U_3O_8 leaching and 60 for gold vat leaching with cyanide. As of 1983-84, six of these gold leaching mills were utilizing the relatively new carbon-in-pulp method for extracting gold from the cyanide leach solution and 54 were still the older, more labor-intensive Merrill-Crowe or zinc dust precipitation method. It is expected that in the future, all new gold leaching mills in the

Republic of South Africa will utilize the carbon-in-pulp method. Overall, the Witwatersrand Basin operations can be grouped according to the various commodities produced as follows:

Those producing only gold bullion.

Those producing gold bullion and pyrite concentrates.

Those producing gold bullion and U_3O_8 (in the form of uranium hexafloride) along with pyrite concentrates and/or sulfuric acid either for their own use or for sale to other operations.

By comparison, the Eastern Transvaal operations utilize gravity methods (to extract the high proportion of free gold) and flotation-roasting-leaching to extract the remaining recoverable gold.

The 17 Canadian operations represent 18 separate mills. All 18 of these mills utilize vat leaching with 14 utilizing the Merrill-Crowe method for gold extraction and 4 (including all 3 of the Hemlo District operations) using the carbon-in-pulp method. Only 4 of the 18 mills do not use preaeration or flotation methods prior to the cyanide leach stage and only 5 have jigs installed to extract free gold. There are no heap leaching operations represented by the 17 Canadian operations.

The seven Australian operations represent six separate mills. Four of these mills are utilizing the Merrill-Crowe method for gold extraction and two are utilizing the carbon-in-pulp method. All five of the primary gold ore mills have a gravity separation circuit included to effect recovery of free gold. Of the nine ore types being beneficiated by the six mills, two are refractory ores (requiring roasting) and two require flotation and cyanide leaching of the concentrate. No heap leaching operations are represented by these seven Australian operations.

The 17 U.S. producers include 8 heap leaching operations, 6 vat leaching operations, 2 operations utilizing both heap leaching and vat leaching, and 1 dredging operation using conventional gravity methods for gold recovery. Both heap and vat leach operations are characterized by a preponderance of carbon-in-pulp gold extraction circuits. For example, only three of the heap leaching operations and only two of the major circuits at the vat leaching operations utilize the old Merrill-Crowe gold extraction methods. Of particular interest is that all of the newest vat leach mills use autoclaves to roast their sulfidic ore prior to cyanide leaching and two of the vat leaching mills use the preoxidation-chlorinoxidation method to treat that portion of their mill feed that is carbonaceous.

PRODUCTION COSTS

Table 5 and figure 3 illustrate average production costs for the four most important MEC gold-producing countries. These four MEC countries, representing 61.7 pct of total 1985 world mine production and 80.5 pct of 1985 MEC mine production, are felt to be indicative of the basic technical, geologic, and economic factors that govern the cost structure of the world gold mining industry. The other evaluated countries are not included in the table owing to the small sample size in each country, which is not conducive for portraying an indicative national average, or to avoid disclosing possible company proprietary data. The production costs shown in table 5 are expressed in constant U.S. dollar terms and are based on current extractive technology.

Because the cost estimates are expressed in a single base currency (the U.S. dollar), the changing value of the dollar relative to the currencies of other countries and the differential in rates of inflation have a significant impact upon the absolute and relative values of the production cost estimates. To quantify this effect, it is necessary to compare the January 1985 average total cost estimates for the 0-pct discounted-cash-flow rate of return (DCFROR) level with the comparable January 1984 estimates presented in reference 1.

The January 1984 average total cost estimate for the Republic of South Africa underground mines was \$285/tr oz. By comparison, the January 1985 estimate is \$204/tr oz, a decline of 28.4 pct over a 1-yr period. Similarly, Canada shows a 9.6 pct decline, Australia a 5.9 pct decline, and the U.S. weighted-average shows (by definition) no change. Essentially, all of these percentage changes reflect the degree to which the value of the U.S. dollar (in percentage terms) has increased or decreased in value relative to the currencies of these countries after accounting for the differential in rates of inflation. These economic factors must be kept in mind when making cross-country comparisons of the current January 1985 cost estimates.

As previous mining and beneficiation sections have indicated, cross comparisons of the individual cost structures for these major MEC gold producers are technically very difficult because of widely varying gold grades, ore types, size of operations, plus many other geologic, engineering, and economic factors. The Republic of South Africa underground mines, which contain the vast majority of the evaluated recoverable gold summarized in table 3, have the lowest average total cost estimates at both the 0- and 15-pct DCFROR levels. The Republic of South Africa underground mines on a weighted-average basis have three advantages: (1) the presence of a number of very large "supermines" with excellent scale economies, (2) the presence of a relatively low-cost labor structure, especially in relation to the other three countries, and (3) relatively high ore grades, at 7.7 g/mt. The total U.S. cost structure is the highest of the four countries, owing primarily to the low average grade of the operations (2.0 g/mt). It is the only one of the four countries with a mill operating cost estimate that is higher than the mine operating cost estimate; this is primarily due to the predominance of lower cost (but also lower grade) surface minable tonnage.

Byproduct revenues are insignificant, overall, for all four countries. However, two of the surface waste reprocessors in the Republic of South Africa do accrue significant byproduct revenue from the sale of recoverable uranium (hence the high byproduct revenue estimate for these operations as a group) as do a few of the underground mines.

Table 5.—Gold production costs for selected MEC's

(All costs are in January 1985 U.S. dollars per troy ounce of recoverable gold on a weighted-average basis)

	Operating cost			Byproduct credit	Net operating cost	Recovery of capital[2]	0-pct DCFROR		15-pct DCFROR		
	Mine	Mill	Other[1]				Taxes and royalties[3]	Total cost[4]	Taxes and royalties[5]	Return on investment[6]	Total cost[7]
Austrialia[8]	143.68	77.32	4.39	1.01	224.38	76.07	0.00	330.45	0.00	43.78	344.23
Canada	131.69	61.45	3.10	2.55	193.69	62.52	7.6	263.81	81.09	57.62	394.92
South Africa, Republic of:											
Underground	137.17	22.83	.12	11.85	148.27	41.22	14.77	204.26	29.97	15.99	235.45
Surface	85.49	181.72	.08	[9]107.52	159.77	47.43	18.91	226.11	46.17	37.49	290.86
United States[10]	129.09	150.59	2.75	2.22	280.21	49.86	23.67	353.74	44.51	48.53	423.11

[1]Includes cost of smelting, refining, and transportation of gold dore.
[2]Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure as of January 1985, and investments required over the life of the operation.
[3]Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 0-pct DCFROR.
[4]Equal to the sum of net operating costs, taxation, and capital recovery determined at a 0-pct DCFROR.
[5]Includes property, State, federal, and severance taxes and royalties, where applicable, calculated at a 15-pct DCFROR.
[6]The revenue increase necessary per troy ounce to obtain a 15-pct DCFROR.
[7]Equal to the sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery *plus* the per troy ounce increase necessary to provide a 15-pct DCFROR after taxation.
[8]Average of surface and underground producers, including a large tailings reprocessing operation.
[9]Includes 2 properties with significant uranium revenues, which increases the overall average for this category of resource.
[10]Includes 1 large underground producer to preserve confidentiality.

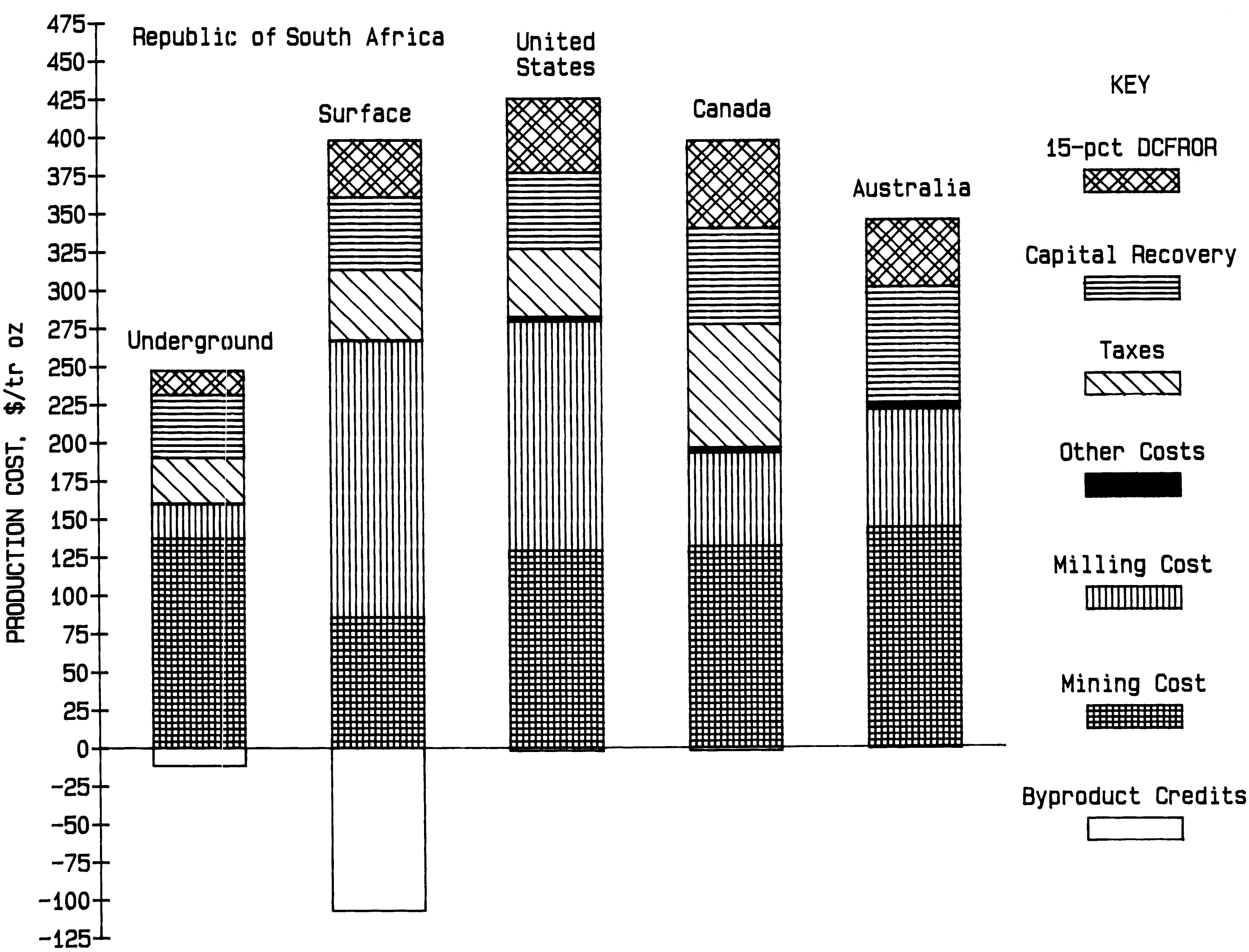

Figure 3.—Gold production costs for producing mines in selected MEC's (January 1985 U.S. dollars).

For all four countries, capital costs per troy ounce of recoverable gold are relatively similar, ranging from $41.22 to $76.07. As a percentage of total production cost, capital costs range from 14.1 to 25.3 pct. This similarity most likely reflects the fact that each operation adjusts to the particular ore and ore grade that constitutes their demonstrated resource. For example, the underground mines obviously have much higher total capital cost requirements owing to the neeed for investment in items such as shaft systems, etc., but these costs are generally offset by the higher grade of their underground ore.

For all countries, taxation expenses average less than 10 pct of total cost at the 0-pct-DCFROR level and, since taxation expense is primarily revenue determined, it rises in proportion to the prespecified DCFROR. The Canadian taxation estimate at the 15-pct-DCFROR level is heavily influenced by the high total capital cost requirements of several operations that require significantly higher revenues to cover these costs *and* obtain the required DCFROR. Australia does not tax revenue accruing to the production of gold.

AVAILABILITY

TOTAL RECOVERABLE

A total of 110 primary gold producing operations were evaluated to determine total gold availability and gold production cost at the 0- and 15-pct-DCFROR levels. The results are shown in figure 4. A total of 792.3 million tr oz of recoverable gold is available from all evaluated operations; the Republic of South Africa accounts for 87.5 pct of the total.

The January 1985 gold price was $302/tr oz. At the 0-pct-DCFROR level, 96 pct of total gold is available at a longrun total cost of $400/tr oz or less and 82 pct is available at $300/tr oz or less. At the 15-pct-DCFROR level, 90 pct of total gold is available at a cost of $400/tr oz or less (with the Republic of South Africa accounting for 92 pct of the total) and 72 pct of total gold is available at $300/tr oz or less (with the Republic of South Africa accounting for 94 pct of the total). In terms of total production cost and total resource inventory, the Republic of South Africa thoroughly dominates the gold industry of the MEC's. However, longrun gold prices exceeding $400/tr oz are required to ensure that all current producers will be able to cover their total production costs over the current estimate of mine life.

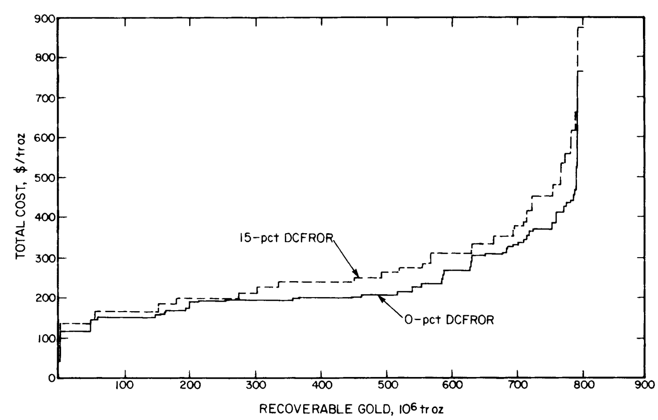

Figure 4.—Potential total gold available from producing mines in MEC's (January 1985 U.S. dollars).

ANNUAL CAPACITY

Annual output is also dominated by the Republic of South Africa. Figure 5 shows potential production (at capacity levels) in 1985 from all 110 producing mines. The curves show potential production available at a price level that covers net operating costs and at a price level that covers total production costs at the breakdown level (0-pct DCFROR). As shown, the 110 analyzed mines represent an estimated total capacity production level of some 26.3 million tr oz as of 1985, with the 43 Republic of South Africa operations accounting for 22.1 million tr oz (84 pct) of the total. During 1985, these operations accounted for the great majority of primary gold production and approximately 73 pct of total mine production of gold (including byproduct production) in the MEC's. If the price of gold is held constant at the January 1985 level of $302/tr oz, approximately

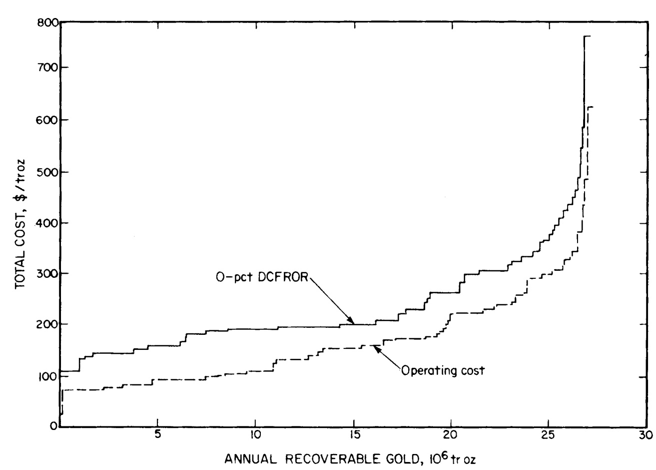

Figure 5.—1985 gold capacity from producing mines in MEC's (Janauary 1985 U.S. dollars).

80 pct of total production in that year could cover longrun average total cost at the breakeven level. This again indicates that some operations may not be able to continue producing over the long term unless prices (in constant January 1985 U.S. dollars) rise above the $302/tr oz level.

Figure 6 shows annual production potential for all operations from 1985 to 1995. Annual capacity in 1995 from these mines (estimated at 26.7 million tr oz) may not be significantly different from the 1985 level even with no new additions to demonstrated resources. This assumes that market forces are the only factors at play and technology remains constant. A serious worsening of the political situation in the Republic of South Africa or falling gold prices could significantly alter this outlook. Technological improvements could lower costs and/or make lower grade material economic. It is important to note that only reef material in the Republic of South Africa grading over 2.5 g/mt was included in the resource availability estimates.

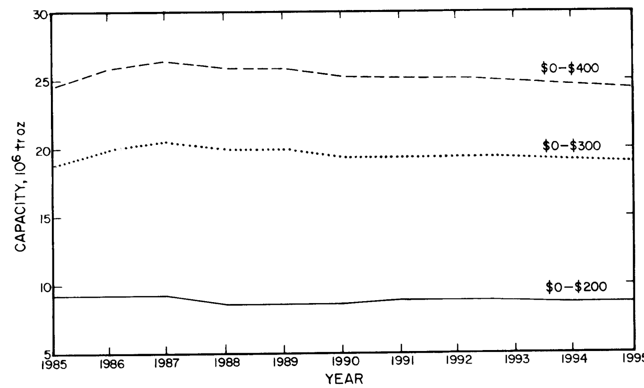

Figure 6.—Potential annual gold production from producing mines in MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

CONCLUSIONS

The purpose of this chapter has been to outline some general information and analyses concerning 110 of the MEC's most significant primary gold producers. Most of the information has been updated from the original, in-depth study published as Bureau of Mines Information Circular 9070.

Some of the important results of this updated summary follow:

1. The Republic of South Africa should remain by far the world's largest gold producer through the remainder of this century (barring non-market problems) and contains the world's largest demonstrated resource of recoverable gold.
2. A large majority of the evaluated demonstrated resources are contained in operations that will employ underground mining methods. The majority of the analyzed surface minable resource is contained in conventional surface mining operations in the United States and waste reprocessing operations in the Republic of South Africa. Many of the newer operations and explored deposits in the United States and Australia, which are not represented in the analyzed resource, are or will be surface mining operations, but the long-term availability of newly mined gold will continue to be dominated by underground operations.
3. The United States accounts for approximately 4 pct of the total recoverable gold. The United States possesses a viable gold industry that continues to expand. The major long-term issue for the United States is production maintenance through replacement of surface minable resources that have relatively short lives by comparison to underground producers in the Republic of South Africa or Canada. Of the total recoverable gold evaluated in the United States, approximately 75 pct is available from surface mines.
4. The changing value of the U.S. dollar relative to the currencies of other gold-producing nations significantly affects the estimated cost structure and determined profitability of gold mining operations in foreign countries when production costs are incurred in local currency but expressed in terms of the U.S. dollar.

REFERENCES

1. Thomas, P. R., and E. H. Boyle, Jr. Gold Availability—World. A Minerals Availability Appraisal. BuMines IC 9070, 1986, 87 pp.
2. Simons, F. S., and W. C. Prinz. Gold. Ch. in United States Minerals Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 263-275.
3. Lucas, J. M. Gold. Sec in BuMines Mineral Commodity Summaries 1986, pp. 62-63.

GRAPHITE

INTRODUCTION

The major applications of flake graphite are in refractories, foundries, facings, and crucible manufacturing for industries. Graphite as refractories depends upon its high heat conductivity and its ability to retain strength at high temperatures. Other uses are lubricants, brake linings, and pencils.

Natural graphite occurs as three basic types based on varying degrees of crystallinity: amorphous, crystalline flake, and high crystalline (vein type). Amorphous graphite resources are abundant worldwide, and most of the U.S. supply is provided by Mexico. Amorphous graphite has not been classified as a strategic and critical material and is not analyzed in this chapter. One of the major uses of flake graphite is in crucible manufacturing for the foundry melting of steel and nonferrous industries. In fabrication of the silicon carbide-graphite crucibles, about 30 pct is composed of flake graphite having a carbon (C) content of 80 pct. At present the crucible mix relies on blending of graphite from many sources.

The refractories industry also uses a variety of blended graphites with minimum of 85 pct C and flake size ranges of minus 30 mesh to plus 100 mesh, equivalent to graphite A in this study. A recent development in the graphite refractories industry has been the introduction of magnesia-carbon bricks for use in steel furnace linings and the development of alumina-carbon (graphitized aluminas) refractories. The thermal shock and erosion resistances of the refractories are improved by the addition of graphite. The carbon content of the magnesia-carbon bricks in United Kingdom and U.S. furnaces may be as high as 15 to 20 pct. The use of graphitized aluminas in the process of continuous casting in steel production improves the corrosion resistance and thermal shock resistance of alumina refractories, which are essentially used to control and protect the metal flowing from the ladle to the water-cooled mold. The composition and texture of alumina-graphite refractories vary enormously depending on the graphite sources and the manufacturers' particular recipes.

Graphite powder with sizes ranging from minus 100 mesh to minus 325 mesh, equivalent to graphite B in this study, is used in the lubricants and foundry industries and in the making of conductive coatings and paints.

This chapter addresses the resources and economics of flake and high-crystalline graphite. Flake graphite and high-crystalline graphite products and prices vary according to their end uses. Based on the specifications for various graphite products and information on products from the properties in this study, it was decided that for this study a two-tier product-pricing arrangement would be most applicable to economic analysis. The two-tiered system comprises graphite A and B product groups. Graphite A products essentially represent all products with 75 to 100 pct of the flakes being retained on 80- or 100-mesh screens, and graphite B products essentially represent those products with 75 to 100 pct of the flakes less than 80 or 100 mesh in size. The differing mesh-size cutoffs are due to the different sizing practices at various worldwide graphite operations.

The fact that graphite A and B products are essentially coproducts with different price levels meant that this study also had to apply a price proportioning methodology for economic analysis. Under this price proportioning, costs and revenues are allocated between both products, thus providing a determined cost of production for each product. For modeling purposes and comparison between operations, it is assumed that a relationship exists between market prices and the average total cost of production (see the methodology section of this Bulletin for a complete discussion).

In calculating the price ratios of graphite A to graphite B, the average price differentials between the maximum and minimum prices for graphite products from Madagascar, Norway, Sri Lanka, and the Federal Republic of Germany were compiled from the February issues of Engineering and Mining Journal for 1979-84. No published prices were available for the countries of Brazil, the Republic of Korea, India, Mexico, Zimbabwe, and Canada. These latter countries were assigned price proportions similar to the published prices of Norway since the Madagascar, Sri Lanka, and the Federal Republic of Germany prices basically reflect unique product situations. For Madagascar, the graphite A and graphite B price ratio is similar to its flake and dust ratio of 2.4:1.0. The ratio for all other countries is 2.5:1.0.

This chapter is an updated summary of Bureau of Mines Information Circular 9122 "Flake and High-Crystalline Graphite Availability—Market Economy Countries. A Minerals Availability Appraisal" (*1*).[1] Only one major change has been made to the data for this update. The change involves the Alabama flake graphite deposits, which now reflect a 1.0:1.0 graphite A to graphite B content as well as the higher cost level of the worst case scenario addressed in reference 1.

Additional information on graphite is available from other Bureau of Mines sources (*2-3*).

[1]Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

GEOLOGY AND RESOURCES

GEOLOGY

Flake graphite is derived from carbonaceous material in sedimentary rocks that has been metamorphosed to gneisses and schists (*4*). The present graphite grade and quality is dependent upon (1) the carbon content of the original sedimentary rocks, (2) the original distribution of the carbonaceous material, (3) the type of original carbonaceous material, and (4) the forces involved in the metamorphism.

Worldwide, flake graphite deposits occur as lenses or layers of disseminated or massive graphite flakes. The lower grade deposits (10 pct C or less) are represented by the Madagascar, Republic of Korea, Mexico, and India producers, as well as the larger of the two Brazilian producers and the Canadian and U.S. producers analyzed in this study. They are all disseminated flake deposits and most of the producers mine either heavily or partially weathered ores. The higher grade deposits (10-20 pct C) are represented by the underground mines of Norway, Federal Republic of Germany, and Zimbabwe and one small producer in Brazil. The higher grade represents a more massive occurrence of the graphite particles.

U.S. flake graphite resources occur in gneiss and schist in the States of New York, Pennsylvania, Alabama, Texas, and California; however, only the Alabama resources were evaluated in this study. The immense Madagascar flake graphite resources occur in lenses or layers distributed over a belt several hundreds of kilometers long in the eastern and southern portions of the island. Only the weathered ore from eight producers and standby concession areas in the Manampotsy District of the eastern part of the island were considered for economic evaluation. In Brazil, major graphite resources are located in northern Minas Gerais and southern Bahia. Only two producers in Minas Gerais were included for evaluation purposes. In Canada, the gneissic and metasedimentary belts of the Precambrian Grenville Series in the Provinces of Ontario and Quebec have hosted past producers of flake graphite. One producer and one nonproducing deposit included in the evaluation are in this region. In addition, one nonproducer in the Province of Saskatchewan was evaluated for economics in this study.

High-crystalline graphite deposits are solely represented in this study by the Sri Lanka deposits. These deposits occur in the form of veins, lenses, or pockets of almost pure graphite in fissures or fractures in crystalline limestones, quartzites, and garnet-sillimanite gneiss. The veins range from a few centimeters up to 3 m in thickness and show a form of zoning with an outer zone consisting of graphite crystals orientated at right angles to the vein wall and an inner zone of platy graphite.

RESOURCES

As shown in figure 1, the world's flake and high-crystalline graphite reserve base is estimated to be 91.6 million mt of contained carbon. Flake and high-crystalline graphite reserve base values for Africa, Asia, and Europe are estimated at 36.3, 31.8, and 18.1 million mt of contained carbon, respectively. For comparison, also included in figure 1 are the evaluated flake and high-crystalline graphite reserves of this study.

As shown, the reserve base indicates much higher tonnages in Asia, Africa, and Europe than the evaluated reserves of this study. Owing to a lack of production cost information, two centrally planned economy countries (CPEC's)—China, which is one of the world's largest producers, and North Korea, which is another large producer, were not evaluated in this study. This was also the case in Europe where it was not possible to evaluate Soviet Union and Czechoslovakian operations. In Africa, only the reserves at Madagascar's six present producers and two standby concession areas containing weathered ores in the Manampotsy District were evaluated. Other districts contain immense resources.

However, it is important to note that, except for underrepresentation of India, complete coverage of 1984 market

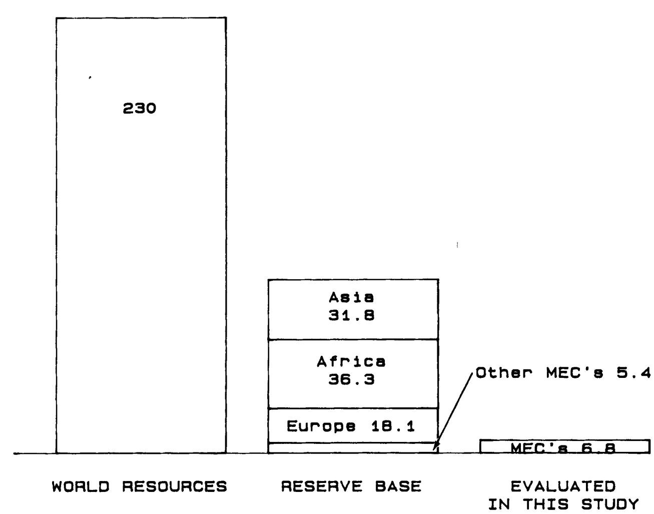

Figure 1.—Estimates of world flake and high-crystalline graphite resources (million metric tons of contained carbon).

economy country (MEC) production of flake and high-crystalline graphite is attained by the properties evaluated in this study. This reserve of 82 million mt of ore containing 6.8 million mt of carbon is detailed, by country, in table 1. The properties included in this study are listed in table 2.

The total contained carbon of table 1 is composed of 43 pct in Brazil, 17 pct in Madagascar, 11 pct each in the United States and Canada, and the remaining 18 pct in Zimbabwe, the Republic of Korea, Sri Lanka, Norway, the Federal Republic of Germany, India, and Mexico. The evaluated reserve of South America is primarily contained in one extremely large Brazilian producer, while the North American evaluated reserve is mostly nondeveloped with only one Canadian operation in production.

Table 1.—Summary of evaluated MEC flake and high-crystalline graphite resources evaluated for this study as of January 1985

Country	In situ ore, 10^6 mt	In situ grade, wt pct carbon	Contained carbon, 10^6 mt
Africa:			
Madagascar	16.39	7.0	1.15
Zimbabwe	1.83	26.0	.47
Total	18.22	NAp	1.62
Asia:			
India	.20	10.0	.02
Republic of Korea	1.67	5.2	.09
Sri Lanka	.13	94.2	.12
Total	2.03	NAp	.23
Europe:			
Germany, Federal Republic of.	W	W	W
Norway	W	W	W
Total	1.57	NAp	.39
North America:			
Canada	8.00	9.2	.74
Mexico	2.72	4.0	.11
Total	10.72	NAp	.85
South America: Brazil	28.56	10.3	2.94
Total, foreign	61.10	NAp	6.03
United States	20.90	3.7	.77
Grand total	82.00	NAp	6.80

NAp Not applicable. W Withheld to avoid disclosing proprietary data.

Table 2.—MEC graphite properties included in this study

Continent, country, and property name	Current status[1]	Mining method[2]	Ownership
Africa:			
Madagascar:			
Ambatomitamba	P	S	Soc. Miniere de la Grande Ile.
Andasifahatelo	P	S	Soc. Arsene Louys et Compagnie.
Antsirakambo	P	S	Etablissements Gallois.
Faliarano	P	S	Etablissements R. Izouard.
Marovintsy	P	S	Etablissements Gallois.
Sahamamy	P	S	Etablissements Rostaing.
Sahanavo	PP	S	Soc. Minere de la Grande Ile.
Tsaravoniany	PP	S	Etablissements R. Izouard.
Zimbabwe: Lynx Mine	P	U	I.D.C. Zimbabwe and Kropfmuhl A/G.
Asia:			
India: Temrimal	P	S	Agrawal Graphite Industries.
Korea, Republic of:			
Gun-Ja	P	S	Dae Han Graphite Mining Co.
Pyong-Taek	P	S	Pyong-Taek Graphite Mining.
Yong-Un	E	U	Republic of Korea.
Sri Lanka:			
Bogala	P	U	Sri Lanka Government (SMMDC).
Kahatagaha-Kolongaha	P	U	Do.
Ragedera	P	U	Do.
Rangala	P	U	Do.
Europe:			
Germany, Federal Rep. of:			
Kropfmuhl	P	U	Grafitwerk Kropfmuhl A/G.
Norway: Skaland	P	U	Atlantic Richfield Co.
North America:			
Canada:			
Bouthillier	E	S/U	Orrwell Energy Corp. Ltd.
Deep Bay	E	S	Superior Graphite Co.
Notre Dame Du Laus	P	S/U	Asbury Carbon, Inc.
Mexico: Telixtlahuaca Mine	P	S	Mexican Government.
United States:			
Alabama Mill1[3]	PP	S	Internat. Carbon and Minerals.
Alabama Mill 2[3]	PP	S	Various owners.
Alabama Mill 3[3]	PP	S	Do.
Alabama Mill 4[3]	PP	S	Do.
South America:			
Brazil:			
Itapecerica	P	S	CIA Nacional de Grafite, Ltd.
Pedra Azul	P	S	Do.

[1]P, producer; T, temporarily shut down; PP, past producer; E, explored.
[2]S, surface; U, underground, S/U, combined surface and underground.
[3]Designated proposed milling complexes to receive ore feed from 16 individual proposed mines.

U.S. AND WORLD HISTORICAL PRODUCTION

Most MEC flake and high-crystalline graphite production comes from Brazil, Madagascar, the Federal Republic of Germany, and Sri Lanka. MEC production in 1985 was estimated at 63,000 mt, with Brazil accounting for a total of 32,700 mt or 52 pct of the total. Figure 2 illustrates production trends in 5-yr intervals for major flake and high-crystalline graphite producing countries from 1950 through 1985. Madagascar has been a steady producer since 1950 with a peak production of 20,000 mt in 1970, and a low of 12,000 mt in 1980. Brazil produced 470 mt in 1950 and increased to 1,200 mt in 1965. Between 1965 and 1970, Brazil doubled production to 2,500 mt. An extremely rapid growth program resulted in 32,700 mt of production in 1985, an almost thirteenfold increase in the past 15 yr.

High-crystalline production from Sri Lanka accounted for 38 pct (13,000 mt) of the 1950 production of 34,000 mt, and has been steadily declining to a point where it represents only 8.6 pct (5,400 mt) of the 1985 production of 63,000 mt. A recent proposal between a group of Japanese investors and the Sri Lanka State Mining and Mineral Development Corporation, involving capital investments in equipment for restart of two abandoned mines, could reverse this declining trend.

Domestic production of flake graphite was insignificant compared with world production, and production ceased entirely in the late 1970's. The two largest world producers are China and the U.S.S.R. They account for approximately 50 pct of total world production of amorphous and flake graphite. Both countries are known to produce a great amount of amorphous graphite and the actual tonnage for flake graphite is not available. Consequently, production trends for China and the U.S.S.R. are not shown in figure 2. China is the major exporter of flake graphite to MEC's, and the U.S.S.R. is the major exporter of flake graphite to CPEC's.

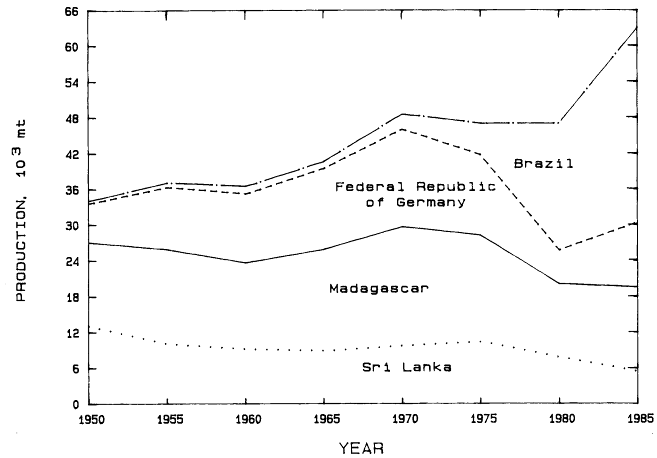

Figure 2.—MEC flake and high-crystalline graphite production, 1950–85.

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

Table 2 lists the 29 properties evaluated in this study by production status, mining method, and ownership. Madagascar, Brazil, and the United States employ or are proposed to employ surface mining methods exclusively. Sri Lanka, Zimbabwe, Norway, and the Federal Republic of Germany use underground mining methods. Approximately 83 pct of the total annual ore capacity for all 29 properties is or will be mined by surface methods, 10 pct will be mined by a combination of surface and underground methods, and the remaining 7 pct is mined by underground methods.

The Madagascar surface operations mine highly weathered soft ore usually at a site with a slope to facilitate the gravity transportation system. The feed box at the field washer is at least 5 m below the lowest elevation of the ore horizon being worked at the site. This results in the maximum resource exploitation through gravity flow. The Madagascar operations typically dispose waste by gravity flow to a nearby river.

The ore at the two Brazilian open pit operations is believed to require at least partial drilling and blasting. The Pedra Azul operation has the largest annual capacity of any of the MEC operations analyzed, accounting for 18 pct of total surface capacity. The 16 individual proposed mines that would feed the four proposed U.S. milling complexes in Alabama, would mine weathered, disseminated ore. The Canadian producer is currently mining only surface ore with 100 pct drilling and blasting but is expected to mine underground in the future.

The larger Sri Lanka underground operations mine veins by cut-and-fill methods in relatively deep workings. The other two Sri Lankan underground mines have the smallest annual capacity of all MEC producers with less than 1,000 mt of ore being mined per year; mining is done by picks and shovels, and ore transportation is by wheelbarrows. The underground mines in Europe and Zimbabwe mine much wider ore widths with cut-and-fill and sublevel stoping methods. The nonproducing operation in the Republic of Korea was analyzed to obtain an idea of the relative economics of small ore bodies of unweathered 10 pct C graphite material mined by underground methods with adit entry. This type of occurrence is common in the southern part of the country but radically different from the currently producing surface mining operations of the Republic of Korea.

PROCESSING

Beneficiation processes vary from complex four-stage flotation at the European mills to the simple hand sorting and screening of high-grade ore at the four Sri Lanka operations. The Madagascar ore is soft and no primary crushing and grinding are necessary, and the products contain the highest proportion of coarse flakes (graphite A) in the world. Ore is sluiced to the field washing plant where the ore undergoes desliming to remove clay fraction and is subjected to a rougher flotation to produce a rougher concentrate grading 60 to 70 pct C. This concentrate is transported to the refining mill for further grinding and flotation to 85-pct-C products that are screened to a variety of graphite A products marketed as large flakes, containing >75 pct minus 40 mesh and >97 pct minus 60 mesh, or fine flake containing >25 pct minus 30 mesh to >95 pct minus 80 mesh. Madagascar graphite B is marketed as extra-fine flake graphite of minus 80-mesh size.

South Korean and Indian operations produce graphite A and graphite B products for their own consumption. The actual number of flotation stages is unclear. It is believed that four-stage flotation is necessary to achieve the 90-pct-C grade in product.

The proposed mills for the Alabama deposits would use four-stage flotation to produce 85-pct-C products. The sophisticated mills in Europe have flexibility in product grades owing to the ability to produce a concentrate from each of the separate flotation stages. Another unique practice is that the final marketable concentrates are sized into various flake size fractions prior to the dewatering and drying stage and are ready for bagging after the drying stage.

Beneficiation for high-crystalline graphite is simple. Sri Lanka ore requires only a slight upgrading of the carbon content through hand sorting and sizing-screening operations.

PRODUCTION COSTS

The weighted-average total cost of production for each property is the cost of production for all products (combined graphite A and B). The weighted-average total production costs of 19 mines and deposits are shown, by country, in table 3 and figure 3. These costs are expressed in constant 1985 U.S. dollars per metric ton of graphite. The total property revenue determined for each operation is divided by the sum of graphite A and graphite B recoverable tonnages to provide a weighted-average total cost of graphite production by country. Weighted-average operating and capital costs are also compiled for country comparisons.

In Sri Lanka, high ore grade, low mill operating cost, and transportation are factors contributing to the relatively low total production cost at both a 0- and 15-pct discounted-cash-flow rate of return (DCFROR). At $195/mt product, the average mine operating cost estimate is relatively high despite having relatively low labor costs per worker and a high grade at about 94 pct C. The high cost level is due to thin veins, high water inflows, poor access, and a high degree of ventilation requirement for relatively deep workings. The mill operating cost estimate of $46/mt product is relatively low on a worldwide basis and is due to the need for only hand sorting, sizing, blending, and bagging in processing.

In Canada, one producer is surface mining at present but will move underground when the pit becomes too deep. The surface operating method requires drilling and blasting for nearly 100 pct of the ore, resulting in a mine operating cost more than double that of the Madagascar operations. Of the two nonproducers, one will be 100 pct surface mining and the other will be a combination of surface mining and underground mining over the entire life of its resources. Mine operating costs range from $14/mt to $21/mt of ore or an average $229/mt of product.

Mine operating costs for the surface mining methods of Madagascar and the United States average $5.41/mt and $6.69/mt of ore, or $100/mt and $180/mt of product, respectively. The low ore grade of the U.S. deposits contributed significantly to the higher mine operating costs based on dollars per metric ton of product.

Mill operating costs for two-stage flotation of Madagascar ore average $90/mt of product. For the sophisticated, five-stage flotation mills in Canada and the proposed four-stage flotation mills in the United States, mill operating costs average $309/mt and $297/mt of product, respectively. The Canadian mills are designed to fulfill customized blending, and the mill operating cost is 6.7 times higher than that of Sri Lanka.

Sri Lankan mine and mill methods are labor intensive with simple equipment, which is reflected in the very low capital recovery cost. Total production costs are lowest for the high-grade Sri Lanka mines averaging $369/mt at 0-pct DCFROR and $414/mt at 15-pct DCFROR. Sri Lanka has the lowest mill operating cost, transportation cost, and capital recovery cost; yet, the total production cost at 0-pct DCFROR is the same as Madagascar. The main reason is the high tax and royalty costs that the Sri Lankan Government has imposed on graphite operations.

The United States and Canada have relatively low tax and royalty costs that would be or are imposed on graphite production. The highest total production costs are for the low-grade graphite ore of Alabama. The Alabama operations would require large capital investments for mining and milling plant and equipment. The weighted-average total cost of production of $920/mt at a 15-pct DCFROR is nearly 1.8 times higher than that of the Madagascar mines and deposits.

Table 3.—Graphite production costs for producing and nonproducing operations in selected MEC's

(All costs are in January 1985 U.S. dollars per metric ton combined graphite A and B products on weighted-average basis)

Country	Operating cost		Transportation[1]	Total operating cost	Recovery of capital[2]	0-pct DCFROR		15-pct DCFROR		
	Mine	Mill				Taxes and royalties[3]	Total cost[4]	Taxes and royalties[5]	Return on investment[6]	Total cost[7]
Canada	229	309	75	613	88	31	732	78	95	874
Madagascar	100	90	41	231	91	47	369	124	77	523
Sri Lanka	195	46	34	275	8	86	369	116	15	414
United States[8]	180	297	65	542	100	18	660	88	190	920

[1]Includes cost of transportation.
[2]Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure as of January 1985, and investments required over the life of the operation.
[3]Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 0-pct DCFROR.
[4]Equal to the sum of net operating costs, taxation, and capital recovery determined at a 0-pct DCFROR.
[5]Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 15-pct DCFROR.
[6]The revenue increase per metric ton necessary to obtain a 15-pct DCFROR.
[7]Equal to the sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery *plus* the per metric ton increase necessary to provide a 15-pct DCFROR after taxation.
[8]Reflects worst case economic scenario from reference 1.

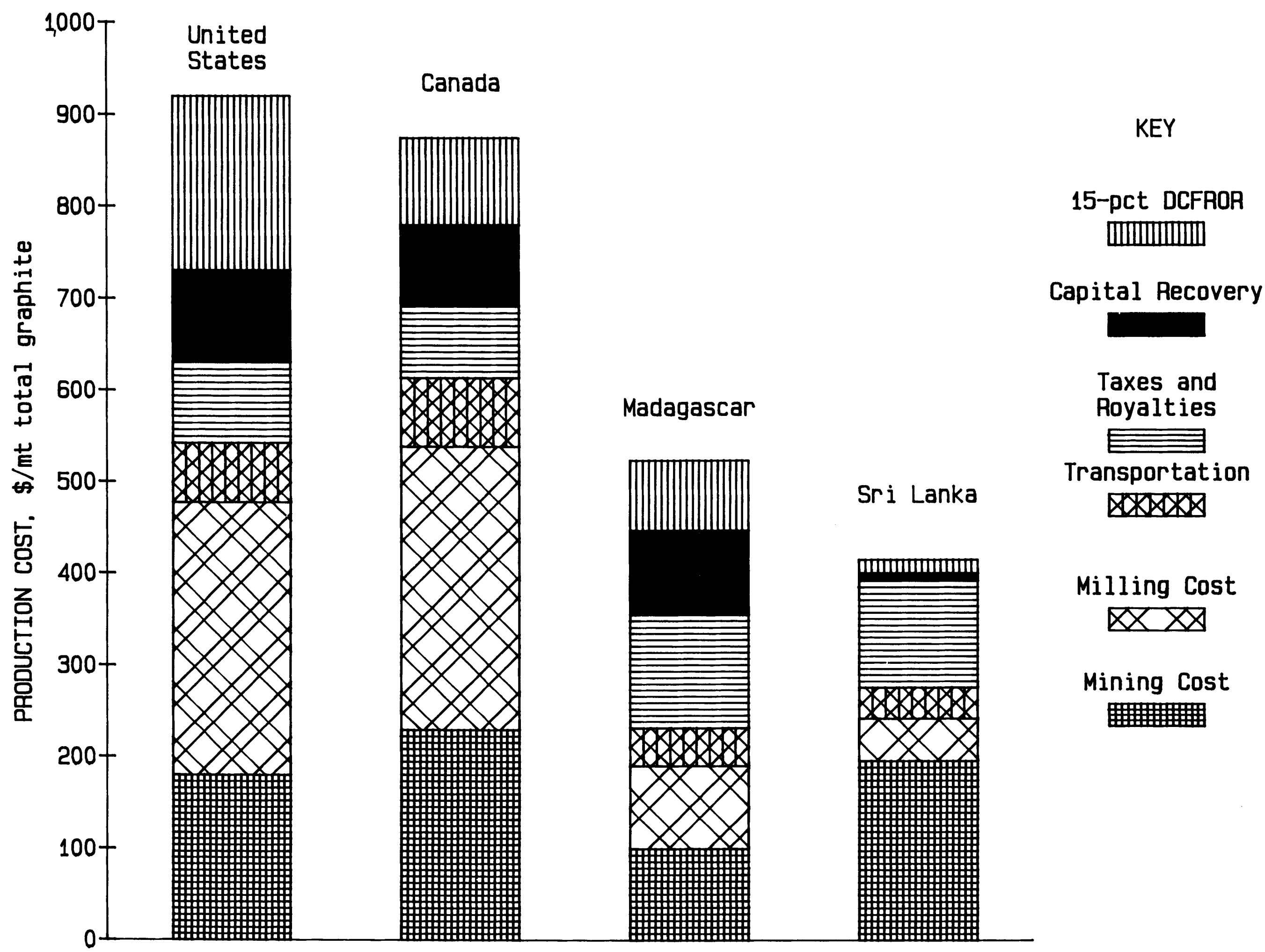

Figure 3.—Weighted-average production costs for operations in selected MEC's (January 1985 U.S. dollars).

AVAILABILITY

All 29 mines and deposits covered in this study recover graphite A. However, only 23 mines and deposits show recoverable graphite B. The four high-crystalline operations in Sri Lanka do not normally produce a product in the graphite B size range. In addition, one Brazilian producer and one nonproducer in Madagascar were indicated as producing or potentially producing only graphite A products, although the data were inconclusive. For deposits recovering both graphite A and graphite B products, the total cost of production for graphite A and graphite B are determined by the price proportion methodology (see the methodology section of this Bulletin for a detailed explanation). The price proportion ratios used in this availability study for the flake graphite properties are 2.4:1.0 for Madagascar and 2.5:1.0 for all other countries.

TOTAL RECOVERABLE

A total of 29 mines and deposits were analyzed, 25 foreign and 4 domestic. As of the beginning of 1985, 19 of the mines evaluated were in production and 1 was shut down after a fire. Nine nondeveloped deposits were evaluated, including two in Madagascar that will replace producing mines when their resources are depleted. Deposits in China and the U.S.S.R. were not evaluated owing to a lack of information. As shown in table 4, total MEC evaluated resources of 78 million mt are estimated to contain 3.06 million mt of recoverable graphite A products and 2.46 million mt of recoverable graphite B products.

GRAPHITE A AVAILABILITY

Figure 4 summarizes total graphite A availability from 20 producing mines at 0- and 15-pct DCFROR's and from 9 nonproducing deposits at a 15-pct DCFROR. As shown, of the total 3.06 million mt of available graphite A products, 2.23 million mt is present in the 20 crystalline-flake producing operations, including 0.12 million mt of high-crystalline graphite from four producers in Sri Lanka. The nine nonproducing crystalline flake deposits account for 0.83 million mt of graphite A.

Figure 4 illustrates that 1.71 million mt of recoverable graphite A is economic at a total production cost level (including a 15-pct DCFROR) of $600/mt or less. This tonnage represents 56 pct of the total graphite A available in all the evaluated deposits and 77 pct of graphite A available in the 20 producing operations.

At $800/mt or less, 70.5 pct (2.16 million mt) of the total graphite A available in all evaluated deposits is economic; the additional 0.45 million mt is contained in five producers and one nonproducer in Madagascar.

As shown, costs for nonproducing deposits are significantly higher than those of producing mines. None of the graphite A in the nonproducing deposits can be recovered at a total production cost level (including a 15-pct DCFROR) of less than $750/mt, only 10.8 pct can be produced at a total cost of less than $800/mt, and the vast majority—0.742 million mt—requires between $800/mt and $1,760/mt. Of this 0.742 million mt, 51.5 pct represents the deposits of Alabama, which would require prices over $1,000/mt to obtain a 15-pct DCFROR.

Table 4.—Summary of MEC resources and recoverable graphite product evaluated for this study as of January 1985, thousand metric tons

Country	Number of deposits	Evaluated resources	Graphite A	Graphite B	Total recoverable products[1]
Producers:					
Brazil	2	26,592	1,160	1,026	2,186
Europe	2	1,492	176	145	321
Korea, Republic of	2	1,644	22	50	72
Madagascar	6	12,603	605	80	685
Others[2]	4	5,440	149	506	655
Total	16	47,771	2,112	1,807	3,919
Nonproducers:					
Canada	2	6,225	309	227	536
Korea, Republic of	1	115	1	10	11
Madagascar	2	3,123	140	23	163
United States	4	20,904	[3]382	[3]395	777
Total	9	30,367	[3]832	[3]655	1,487
Sri Lanka (high crystalline)	4	142	118	NAp	118
Grand total	29	78,280	[3]3,062	[3]2,462	5,524

NAp Not applicable.
[1]Graphite A plus graphite B.
[2]Includes 1 deposit each in Canada, India, Mexico, and Zimbabwe.
[3]Reflects reestimate of relative proportions under worst case scenario for Alabama properties.

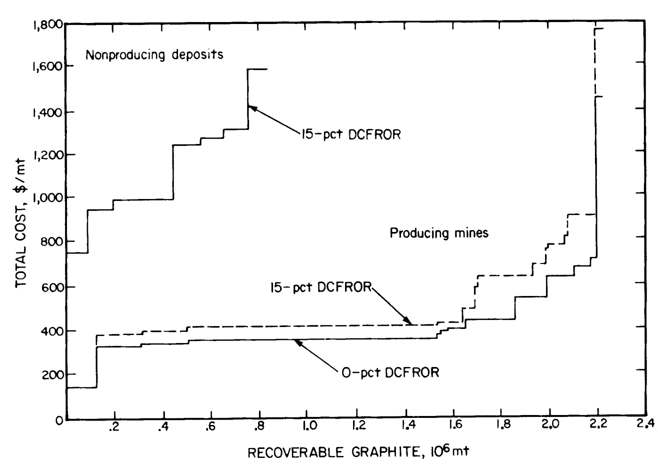

Figure 4.—Potential total graphite A available from MEC's (January 1985 U.S. dollars).

GRAPHITE B AVAILABILITY

Figure 5 summarizes total graphite B available from 15 producing mines at 0- and 15-pct DCFROR's and 8 nonproducing deposits at a 15-pct DCFROR. Total recoverable graphite B of 2.46 million mt is composed of 1.80 million mt from the 15 producers and 0.66 million mt from the 8 nonproducing deposits. One mine in Brazil accounts for 41 pct of the total recoverable graphite B.

Figure 5 illustrates that nearly 96 pct of the graphite B in the producing operations is economic at a total production cost level (including a 15-pct DCFROR) of $400/mt while fully 89 pct is economic at a total production cost level of $300/mt.

By contrast, none of the graphite B in the nonproducing deposits could be recovered with a 15-pct DCFROR at $300/mt and only about 26 pct could be recovered with a 15-pct DCFROR at the $400/mt level. The majority—0.490 million mt—of the graphite B in the nonproducers requires between $400/mt to over $600/mt to achieve a 15-pct DCFROR. The U.S. nonproducers represent approximately 60 pct of the graphite B available from nonproducing deposits and require a 15-pct DCFROR total production cost level of over $490/mt.

ANNUAL AVAILABILITY

Figure 6 presents annual availability curves for graphite A and B from producing mines for the years 1985 to 1995 at various total production cost levels (including a 15-pct DCFROR). The curves reflect the production potential at full capacity levels from currently producing operations.

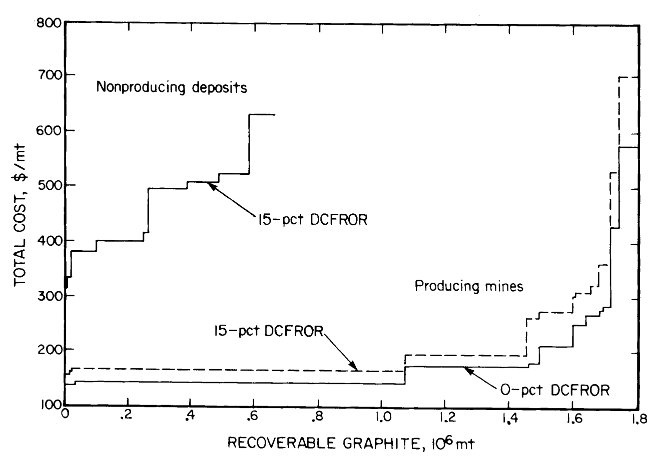

Figure 5.—Potential total graphite B available from MEC's (January 1985 U.S. dollars).

The curves for graphite A show no decline over the next decade in output from those producers with total production cost levels below $400/mt, a decline from 37,300 mt in 1985 to 31,600 mt in the 1990's at those operations with cost levels below $600/mt, and a decline from 53,400 mt/yr in 1985 to 48,100 mt/yr in the 1990's at all 20 of the producers. These very slight decreases reflect a static analysis of the demonstrated resources; it is likely that additional tonnages will be added to replace those depleted.

The annual availability curves for graphite B from 15 producers do not show any decline from the 1985 output level of 37,200 mt over the next decade due to reserve depletion. Those operations with total production cost levels of $400/mt or less account for 35,000 mt of the 1985 capacity.

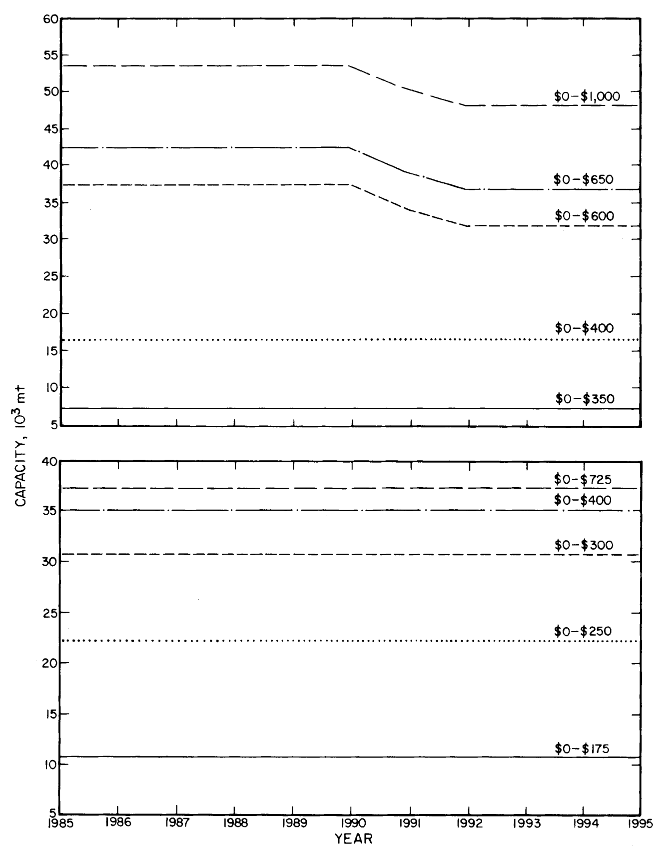

Figure 6.—Potential annual graphite A (top) and graphite B (bottom) production from producing mines in MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

CONCLUSIONS

The flake and high-crystalline natural graphite industry, although presently showing a wider range of suppliers than 20 yr ago, still appears to be a difficult industry to enter at the mining and milling stage. As this study shows, there is no shortage of reserves or resources at presently producing MEC operations and tonnages at the nonproducers are likewise ample. Further, the relative economics shown in this study indicate that the lower cost nonproducers are at or above the highest cost levels of the producers.

If these MEC producers are considered along with the U.S.S.R (a major flake graphite exporter to the Eastern Europe countries) and China (the largest exporter of flake graphite to MEC markets) it can be seen that effectively all flake graphite markets are well served by the present producers. Hence, any proposed new MEC producing operation will have to consider very carefully the particular markets that it will serve before a development decision is made.

Although the United States has large demonstrated resources in Alabama, the probability of developing these resources is very low and cost prohibitive under present market conditions. The United States will continue to rely on foreign supply in the near future. It is suggested that further new technical work on all aspects of all United States flake resources should be done if these resources are to be considered as important to the country's future economic goals.

REFERENCES

1. Fogg, C.T., and E. H. Boyle, Jr. Flake and High-Crystalline Graphite Availability—Market Economy Countries. A Minerals Availability Appraisal. BuMines IC 9122, 1987, 40 pp.

2. Taylor, H. A., Jr. Graphite. Ch. in Mineral Facts and Problems, 1985 edition. BuMines B 675, 1985, pp. 339–348.

3. _____. Graphite (Natural). Sec. in BuMines Mineral Commodity Summaries 1986, pp. 64–65.

4. Weis, P. L. Graphite. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 277–283.

IRON ORE

INTRODUCTION

Iron is the fourth most abundant element, composing almost 5 pct by weight of the Earth's crust and is also the most inexpensive and widely used metal. Iron ore is the source of primary iron for the iron and steel industries, which consume about 98 pct of all iron ore production. It is used almost exclusively for the production of pig iron, and is the main ingredient of the blast-furnace charge. Iron ore is marketed as a number of different products, and the product forms are based largely on physical characteristics.

A plethora of brand names and physical and chemical specifications exist for iron ore products. The products analyzed for availability in this study are lump ore, sinter fines, pellet feed, and pellets. Lump ore is composed of particles 1/4 in or larger, sinter fines are less than 1/4 in and greater than 100 mesh, and pellet feed is less than 100 mesh. Concentrates and fines are agglomerated by pelletization into pellets which are 10 to 20 mm in size.

The availability information in this study shows total long ton of iron ore products available at a specific total cost per long ton iron unit. For many of the mines and deposits in this study, more than one product was evaluated per site, which required the use of price proportions. The price proportions allowed the total cost of production to be allocated among all products rather than being applied to just one.

The iron unit refers to the metal content of the ore and is widely used in the industry as a basis for determining prices, shipping costs, etc. An iron ore unit may be defined as 0.01 lt of contained iron, i.e., 22.4 lb=1.0 pct of a long ton of iron (1 lt=2,240 lb). An iron ore of 65 pct Fe contains 65 iron ore units per long ton of ore.

The information in this chapter is an updated summary of Bureau of Mines Information Circular 9128 "Iron Ore Availability—Market Economy Countries. A Minerals Availability Appraisal" (*1*).[1] Additional information on the iron ore industry is available from other Bureau of Mines publications (*2-5*).

[1]Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

GEOLOGY AND RESOURCES

GEOLOGY

Iron ore can be classified into several categories on the basis of composition and mineralogy. Magnetite, hematite, goethite, and siderite are the most common ore minerals of iron. Other types of ore minerals are pyrite and pyrrhotite (iron sulfides) and chamosite (an iron silicate), which are of minor economic importance at the present time. There are many different types of iron ore deposits, but the vast majority of them can be classified as either bedded sedimentary deposits or massive deposits.

Bedded deposits in Precambrian rocks, called banded iron formations (BIF's), are by far the most important sources of iron ore. Occurrences of these deposits are predominantly in the Precambrian shield areas of the world. BIF's are thinly bedded chemical sediments containing at least 15 pct Fe, normally contain chert layers, usually are folded, and have low to steep dips. Thicknesses may range from less than 25 ft to more than 2,000 ft, and the beds are exposed in belts ranging from a few miles to several hundred miles in length.

The term "taconite" was originally used to refer to many of the unoxidized BIF ores that occur in the Mesabi range in the Lake Superior region of the United States. Taconites are bedded ferruginous cherts of extremely hard ore in which the iron is in either banded or well-disseminated form containing magnetite, iron silicates, and chert, sometimes with hematite and siderite. Taconite contains ores that are low grade containing 15 to 35 pct Fe and 40 to 55 pct SiO_2. Today, most deposits in BIF's with iron contents above 25 pct and which are amenable to beneficiation are considered taconite-type deposits.

Most North American iron formations contain 30 pct or more total iron, 60 to 80 pct of which is economically recoverable. South American itabirites are usually richer in iron content than those in North America, grading about 40 pct Fe. Itabirite is a laminated, metamorphosed iron formation in which the iron is present as thin layers of hematite, magnetite, or martite. The term was originally applied in Itabira, Brazil, to a high-grade, massive, specular-hematite ore (66 pct Fe). Metamorphism has sometimes caused a coarsening of the grain size, which has improved the beneficiation qualities of the deposits. Billions of tons of ore containing more than 64 pct Fe are in the Brazilian itabirite formations, with some deposits containing almost pure hematite.

Oölitic ironstones of Paleozoic to Cretaceous age compose another class of bedded iron deposits of regional importance in the southeastern United States, western Europe, and north Africa. They differ from the BIF's in that, although they are laterally extensive, they are usually less than 50 ft thick and usually average only 25 to 35 pct Fe. The ore consists of very fine grained hematite, quartz, chamosite, and siderite in varying proportions and is usually high in phosphorus. On a global basis, the relative significance of these ores is small.

Iron occurs in several types of massive deposits found mainly in tectonically deformed belts of the Earth and associated with igneous intrusions. The most important types appear to be magmatic segregations, injection, sedimentary, and extrusive deposits. Grades of iron ore range from about 30 to 65 pct Fe. Some of these deposits contain minerals of copper, titanium, phosphorus, vanadium, or other metals that may be produced as byproducts.

RESOURCES

As shown on figure 1, world resources of iron ore are estimated at 800 billion lt and the world reserve base at 206 billion lt of iron ore containing 98 billion st of iron (*5*). As shown on the figure, the centrally planned economy countries (CPEC's) contain an estimated 69 billion lt, of which 59 billion lt is in the U.S.S.R.

For this study, iron ore resources of 68.4 billion lt containing 25 billion lt of recoverable iron have been evaluated. These include 43 domestic deposits containing 19.3 billion lt of iron ore and 86 foreign deposits containing 37.8 billion lt of iron ore. Demonstrated resource information for these deposits, including status, mining and milling method, and products produced is presented in table 1. This table does not include the subeconomic resources of the magnetic taconites of the Mesabi range (21 deposits) that are estimated at 11.3 billion lt, even though these resources were included in the availability analysis (*6*, p. 12). The Carajas deposit in Brazil contains vast resources of over 18 billion mt of 66 pct iron, of which 1.3 billion lt has been evaluated in this study. It should be noted that this resource has had an immense impact on the iron ore industry and will continue to do so in the future.

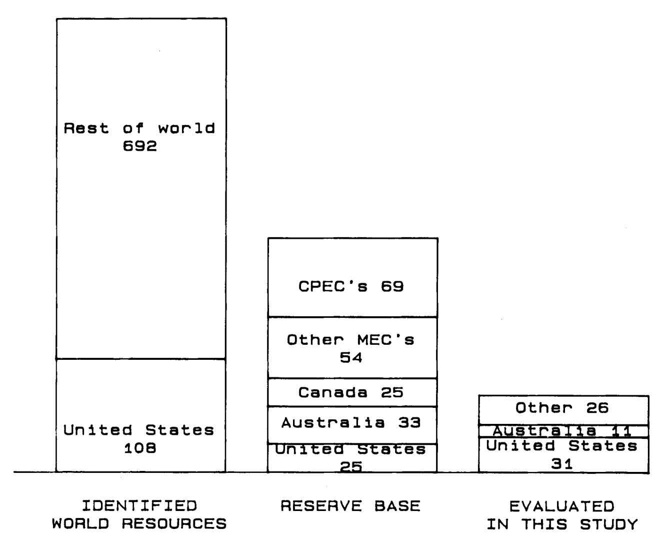

Figure 1.—Estimates of world iron ore resources (billion long tons).

Table 1.—MEC iron ore deposits included in this study

Country and property	Current status[1]	Mining method[2]	Milling method[3]	Products[4]	In situ iron grade, pct	In situ tonnage, 10^6 lt	Contained iron tonnage, 10^6 lt
Algeria:							
Gara Djebilet	E	S	M	S	51.8	969.4	502.1
Quenza	P	S	S	PF,S	53.0	35.4	18.8
Total or weighted average					51.9	1,004.8	520.9
Australia:							
Deepdale	E	S	S	S	57.2	984.2	563.0
Giles Mini	E	S	S	S	63.6	295.3	187.8
Koodaideri	E	S	S	S	61.7	711.6	439.1
Marandoo	E	S	S	L,S	62.1	339.2	210.6
Marillana	E	S	S	S	59.7	738.1	440.6
McCameys Monster	E	S	S	S,L	62.4	246.4	153.8
Middleback Range	P	S	S	L,P,S	63.7	72.8	46.4
Mining Area C	E	S	S	S	62.2	440.0	273.7
Mt. Brockman	E	S	S	S	62.2	393.7	244.9
Mt. Tom Price	P	S	M	L,P,S	61.9	607.5	376.0
Mt. Whaleback	P	S	S	S,L	61.4	1,605.9	986.0
Nammuldi	E	S	S	S	62.5	206.7	129.2
Paraburdoo	P	S	S	S,L	63.4	428.4	271.6
Rhodes Ridge	E	S	S	S	61.8	984.2	608.2
Robe River	P	S	S	P,S	57.0	118.9	67.8
Savage River	P	S	M	P	35.0	79.4	27.8
TR5585	E	S	S	S	62.2	206.7	128.6
West Angelas	E	S	S	S	62.2	314.2	195.4
Wittenoom	E	S	S	S	54.9	984.2	540.3
Yandicoogina	E	S	S	S	58.5	1,223.4	715.7
Total or weighted average					60.1	10,980.8	6,606.5
Brazil:							
Alegria	P	S	S	L,S	63.5	165.2	104.9
Aguas Claras	P	S	S	PF,L,S	62.2	239.7	149.1
Andrade	P	S	S	L,S	65.2	84.1	54.8
Capanema	P	S	S	L,S,P	61.2	154.0	94.2
Carajas	E	S	S	PF,L,S	66.1	1,319.8	872.4
Casa De Pedra	P	S	S	L,S	63.7	227.9	145.2
Caue	P	S	M	L,P,S	56.1	582.3	326.7
Conceicao, Dos Corregos	P	S	HMS	S,P,L	66.6	1,008.4	671.6
Corregos do Feijao	P	S	S	S,PF,L	64.7	101.0	65.3
Fabrika, Joao Pereira	P	S	M	S,L,P	62.5	250.4	156.5
Mutuca	P	S	S	S,L	63.7	55.6	35.4
Periquito	P	S	S	L,S	66.6	144.2	96.0
Tamandua	E	S	S	PF,L,S	63.8	274.6	175.2
Timbopeda	P	S	S	S,L	66.6	150.5	100.2
Samarco	P	S	F	PF,P	53.2	342.1	182.0
Total or weighted average					63.3	5,099.8	3,229.5
Cameroon: Les Mamelles	E	S	F	P	30.1	196.8	59.2
Canada:							
Carol Lake	P	S	S	S,P	37.6	1,707.8	642.1
Fire Lake	P	S	S	P	38.0	345.3	131.2
Mt. Wright	P	S	S	S	31.4	2,445.6	767.8
Wabush	P	S	M	P	36.0	1,674.0	602.6
Total or weighted average					34.7	6,172.7	2,143.7
Chile:							
El Algarrobo	P	S	M	S,L,P	54.0	57.7	31.2
El Romeral	P	S	M	S,L	55.0	74.6	41.0
Total or weighted average					54.5	132.3	72.2
Gabon: Belinga	E	S	S	S	63.9	506.9	323.9
Guinea:							
Mt. Nimba	E	S	S	S	66.7	344.5	229.8
Simandou	E	S	S	PF,L,S	63.1	590.5	372.6
Total or weighted average					64.4	935.0	602.4
India:							
Bailadila No. 5 (Zone B)	P	S	S	S,L	64.4	201.5	129.8
Bailadila No. 14 (Zone B)	P	S	S	S,L	66.9	72.2	48.3
Bolani	P	S	S	S,L	58.9	470.6	277.2
Kudremukh (Zone E)	P	S	F	PF,P	38.1	630.6	240.2
Zone D (GOA)	P	S	S	S,L	60.8	185.3	112.7
Total or weighted average					52.2	1,560.2	808.2
Ivory Coast: Mt. Klahoyo	E	S	M	P	35.7	659.4	235.4
Liberia:							
Bea Mountain	E	S	M	S	41.8	120.9	50.5
Bong	P	S	M	S,P	37.1	245.5	91.1
Mano River	PP	S	S	S	51.5	74.0	38.1
Nimba	P	S	F	PF,P,S	59.1	53.9	31.8
Western Area	P	S	M	S	52.2	402.5	210.1
Wologisi	E	S	M	S	32.7	812.0	265.5
Total or weighted average					40.3	1,708.8	687.1
Libya: Wadi Shatti	E	S	S	S	51.4	782.4	402.2
Mauritania:							
F'Derik	P	S	S	S	64.5	121.2	78.2
Guelbs	E	S	M	S,PF	36.8	380.0	139.8
Total or weighted average					43.9	501.2	218.0

See footnotes at end of table.

Table 1.—MEC iron ore deposits included in this study—Continued

Country and property	Current status[1]	Mining method[2]	Milling method[3]	Products[4]	In situ iron grade, pct	In situ tonnage, 10⁶ lt	Contained iron tonnage, 10⁶ lt
Mexico:							
El Encino-Aquila	P	S	M	L,P	36.4	85.1	31.0
Las Truchas-Ferrotepec	P	S	M	P	48.8	99.5	48.6
La Perla	P	S	M	P,L	50.7	36.2	18.4
Las Hercules	P	S	F	P	58.3	91.9	53.6
Pena Colorada	P	S	M	P	37.7	125.6	47.4
Total or weighted average					45.5	438.3	199.0
New Zealand: Waipipi	P	D	G	S	15.5	262.8	40.7
Norway: Sydvaranger	P	S	P	P	32.6	180.4	58.8
Peru: Hierro (Marcona)	P	S	M	PF,P,S	53.5	1,432.6	761.6
Portugal: Moncorvo	E	S	M	P	37.0	243.7	90.2
Senegal: Faleme Area	E	S	S	S,L	63.6	334.6	212.8
Sierra Leone: Marampa	P	S	G	S	32.3	54.4	17.6
South Africa, Republic of: Sishen	P	S	HMS	S,L	64.0	1,264.6	809.3
Spain: Marquesado	P	S	S	S,L	53.7	30.5	16.4
Sweden:							
Kirunavaara	P	U	M	P,S,L	49.2	1,570.7	772.8
Malmberget	P	U	M	P,S	40.4	204.7	82.7
Svappavaara	P	S	S	P,L	45.0	233.2	104.9
Total or weighted average					47.8	2,008.6	960.4
United States:							
Alabama:							
Big Sandy Area	PC	U	H	P	35.2	24.4	8.6
Birmingham District	PC	U	H	P	34.9	1,061.0	370.3
Southeast Alabama	PC	S	H	P	26.4	258.9	68.4
Total or weighted average					33.3	1,344.3	447.3
Alaska:							
Klukwan	E	S	M	P	10.8	883.5	95.4
Port Snettisham	E	S	M	S	18.9	521.6	98.6
Total or weighted average					13.8	1,405.1	194.0
California: Eagle Mountain	PC	S	H	P	33.4	339.5	113.4
Michigan:							
Cascade Deposits	E	S	H	P	34.2	1,377.9	471.2
Empire Mine	PT	S	M	P	31.5	1,205.5	379.7
Groveland Mine	PC	S	M	P	35.1	101.8	35.7
Republic Mine	TC	S	F	P	34.2	39.8	13.6
Tilden Mine	PT	S	F	P	33.3	842.6	280.6
Total or weighted average					33.1	3,567.6	1,180.8
Minnesota:[5]							
Butler Taconite	PC	S	M	P	32.0	99.5	31.8
Erie Mine	PT	S	M	P	31.7	1,434.4	454.7
Hibbing Taconite	PT	S	M	P	30.7	1,011.1	310.4
Minntac Mine	PT	S	H	P	[6]22.0	1,639.0	361.6
Minorca Mine	PT	S	M	P	[6]21.0	254.6	53.6
National Steel	PT	S	M	P	31.0	944.9	292.9
Peter Mitchell (Reserve)	PT	S	M	P	[6]23.5	1,078.8	254.2
Thunderbird Deposits (Eveleth)	PT	S	M/P	P	32.3	1,187.0	383.4
Total or weighted average					[6]23.6	7,649.3	2,142.6
Missouri:							
Bourbon Deposit	PP	U	M	P	30.4	178.6	54.3
Camels Hump	PP	U	M	P	36.6	22.3	8.2
Pea Ridge	PT	U	M	PF	57.0	103.9	59.2
Total or weighted average					41.3	304.8	121.7
Montana:							
Black Butte	E	S	M	P	22.2	137.9	30.6
Carter Creek Iron Mine	E	S	M	P	30.0	74.6	22.4
Copper Mountain	E	S	M	P	27.8	27.3	7.6
Total or weighted average					25.3	239.8	60.6
Nevada:							
Buena Vista	E	S	M	P	19.0	140.7	26.7
Dayton Iron Deposit	E	S	P	P	42.0	40.4	17.0
Modarelli Mine	E	S	H	P	51.8	26.1	13.5
Pumpkin Hollow	E	S	M	P	26.7	178.1	47.6
Total or weighted average					27.2	385.3	104.8
New Jersey: Mt. Hope Iron Mine	PC	U	M	S	38.4	4.5	1.7
New York:							
Benson Mines	PC	S	M	P	23.5	181.0	42.5
Mineville Mines	PC	U	M	P	42.0	90.0	37.8
Total or weighted average					29.6	271.0	80.3
Texas: Lone Star Deposits	PC	S	H	S	27.0	76.9	20.8
Utah:							
McCahill Ore Body	PC	S	M	P	52.5	49.2	25.8
Rex Ore Body	E	S	M	P	52.5	150.0	78.8
Total or weighted average					52.5	199.2	104.6
Wisconsin:							
Agenda Deposit	E	S	M	P	[6]25.5	157.5	40.2
Black River Falls	PC	S	M	P	30.0	14.5	4.4
Gogebic Deposit	E	S	M	P	31.0	778.5	241.3
Penokee Deposits	E	S	M	P	33.4	2,257.7	754.1
Pine Lake Taconite	E	S	M	P	[6]23.0	202.7	46.7
South Butternut	E	S	M	P	[6]25.9	52.2	13.6
Total or weighted average					31.5	3,463.1	1,100. 3
Wyoming: Atlantic City	PC	S	M	P	26.1	69.6	18.2
U.S. total or weighted average					25.6	19,320.0	5,691.1

See footnotes at end of table.

Table 1.—MEC iron ore deposits included in this study—Continued

Country and property	Current status[1]	Mining method[2]	Milling method[3]	Products[4]	In situ iron grade, pct	In situ tonnage, 10^6 lt	Contained iron tonnage, 10^6 lt
Venezuela:							
Altamira	P	S	S	P,L,S	63.1	128.4	81.0
Cerro Arimagua	E	S	S	PF,L,S	62.2	133.9	83.3
Cerro Bolivar	P	S	S	PF,L,S	63.1	175.4	110.7
Cerro Redondo	E	S	S	P,L,S	61.1	162.4	99.2
El Trueno	E	S	S	PF,L,S	61.1	108.3	66.2
El Pao	P	S	W	S,L	63.1	29.8	18.8
Los Barrancos	E	S	S	PF,L,S	63.1	228.3	144.1
San Isidro	P	S	S	S,PF,L	64.1	385.8	247.3
Total or weighted average					62.9	1,352.3	850.6
Grand total					NAp	57,163.9	25,617.4

NAp Not applicable.
[1]As of January 1986. P, producer; E, explored deposit; PC, permanently closed; PP, past producer; PT, producing but with temporary closures; TC, temporarily closed.
[2]S, surface; U, underground; D, dredge.
[3]G, gravity separation; HMS, heavy-media separation; M, magnetic separation; P, pyrometallurgical processing; S, sizing; W, washing.
[4]P, pellets; S, sinter fines; L, lump ore; PF, pellet feed.
[5]The magnetic taconites in Minnesota contain subeconomic resources of approximately 11.3 million lt of iron at 26 pct Fe and are not included in this table but the 21 deposits were evaluated in the availability analysis.
[6]These deposits are magnetic iron and not total iron, in which the iron content of the silicates and carbonates is not recovered.

U.S. AND WORLD HISTORICAL PRODUCTION

Actual world iron ore production has exceeded 800 million lt in recent years. Of the 1985 total world production of iron ore, approximately 57 pct was produced in market economy countries (MEC's). Five-year trends for the major iron ore producing countries from 1950 through 1985 are shown in figure 2. There are five major MEC's that account for nearly 40 pct of the world production in 1985. Australia and Brazil led production in MEC's in 1985 at 90 and 95 million lt, respectively. The United States, India, and Canada followed with 48 million lt, 42 million lt, and 38 million lt, respectively. The U.S.S.R., the largest iron ore producer in the world, accounted for 30 pct of world production in 1985. In total, CPEC's, primarily U.S.S.R. and China, accounted for 43 pct of 1985 world production.

Production of iron ore peaked between 1975 and 1980 but has declined since 1980 to 799 million lt in 1985. From figure 2 it can be seen that all countries, with the exception of the United States, have had an increase in production over the 35-yr period. The United States shows a steady level of production with a decline by 1985. The increase in production of "other" countries from less than 100 million lt to over 300 million lt can be attributed to the development of iron ore mines in developing countries.

The world iron ore industry has encountered many problems since the mid-1970's such as high energy costs, declining demand, increased competition, and overcapacity. The immediate outlook continues to portray an excess of capacity compared with the demand for iron ore, which will cause the levels of production to remain at similar levels or decline in the near future.

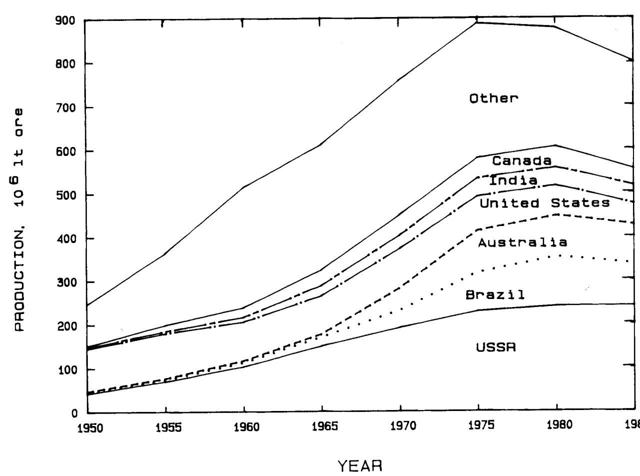

Figure 2.—World iron ore production, 1950-85.

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

Iron ore mining systems for mines evaluated in this study are generally all open pit with the most notable exception being the underground mines of northern and central Sweden. Mining methods are essentially the same for foreign and domestic iron ore. Computer technology has been incorporated into many of the mining and beneficiation processes to increase efficiency and reduce labor.

Drilling of overburden, waste rock, and ore is done by percussion, rotary, or jet-piercing drills with the rotary drill used most widely in large-scale iron mining. Ammonium nitrate, mixed with fuel oil (ANFO), is the most widely used blasting agent. Drilling and blasting is necessary in the U.S. taconite mines while natural ores generally can be excavated without blasting. Large-capacity equipment and automated operations are common in iron ore mining because of the large volumes of material moved and because they reduce labor requirements. Scrapers, shovels, and trucks are all common in ore handling.

Open pit taconite mines in the United States differ from conventional iron ore surface mines in the character of the ore and the relatively larger size of the operation. The ore is of a lower grade and harder than conventional deposits because of the siliceous nature of taconite. Ore is often transported in off-highway trucks, with capacities of 100 lt or more, or in trains. An average pit wall slope of 57° is assumed to provide good stability for hanging wall rock and taconite ore.

Deep, flat-lying bedded deposits, where the ore is not more than 6 m thick, are usually recovered by room-and-pillar operations essentially identical to the methods used in coal mining. Ore recovery is about 80 pct in such operations. Massive- and vein-type iron ore deposits are mined principally by caving methods, supplemented by shrinkage and sublevel stoping.

Common haulage methods used to transport ore to a beneficiation plant include rail, trucks, and conveyors. Rail and conveyors are most often the least expensive haulage method; however, the geometry of the ore body, depth of the pit, and other factors dictate the methods used at any particular site. Combination haulage methods, utilizing conveyors, rail, and trucks, are common in many surface operations.

BENEFICIATION

Almost all iron ore mined is beneficiated in order to obtain uniformly sized products, to improve the iron content, and to eliminate impurities. Beneficiation is accomplished by various methods which may include roasting, leaching, crushing, screening, blending, agglomeration, flotation, and heavy-media separation.

Iron ores generally fall into two broad categories, magnetic taconites and natural ores, and therefore require different beneficiation processes. Normally, owing to the large size of a magnetic taconite open pit, an on-site mill and pellet plant would be constructed. The beneficiation (mill) process would consist of crushing, grinding, magnetic separation, thickening, and filtering. The pelletizing process would consist of balling, sizing, drying, firehardening, and cooling.

Natural ore may be of direct shipping quality that only requires a crushing and screening process followed by direct shipment to the blast furnace. Normally, however, natural ore must be concentrated before shipping. The concentration methods that may be utilized include crushing, screening, heavy-media separation, jigging, and dewatering. Fines are further processed by sintering to produce an acceptable product.

Primary crushing is carried out in jaw crushers, gyratory crushers, or rolls. Secondary crushing is normally accomplished in a cone crusher, by rolls, or in a hammer mill. Grinding is mainly carried out in ball or rod mills.

Commonly used methods for iron ore concentrating are heavy-media separation and the use of the Humphrey's spiral. Flotation is another method utilized to separate iron ore minerals from gangue. Magnetic separation is an especially important process in the beneficiation of iron ores in which the magnetic mineral is separated from other non-magnetic minerals, roasted pyrite from sphalerite, etc., using permanent magnets or electromagnets. Practically all magnetic separation of iron ores is now accomplished by wet separators.

Dewatering, or solid-liquid separation, produces a relatively dry concentrate for shipment. Partial dewatering is also performed at various stages in the treatment, so as to prepare the feed for subsequent processes. The drying of concentrates prior to shipping is the last operation that may be performed in the mineral-processing plant. It reduces the cost of transport and is usually aimed at reducing the moisture content to about 5 pct by weight.

Physical properties of iron ore are important in beneficiation and affect milling costs. Magnetism is important, not only for the concentration of magnetite but also for hematite (hence the use of high-intensity magnetic separators). Specific gravity differences permit concentration of ores by washing, heavy-media separation, and the use of Humphrey's spirals, Reichert cones, cyclones, etc. Some ore can be concentrated merely by screening. Physical-chemical differences permit concentration by flotation. Chemically combined water in hydrous minerals such as goethite (limonite) is hard to drive off; hence, such ore contains less iron and results in a lower price.

AGGLOMERATION

One of the most important physical characteristics of iron ore is the size of the particles. Iron ore feed that contains fine particles causes operational problems in the blast furnace. Hence, most iron ore less than 1/4 in. in size must be agglomerated before it can be used in the blast furnace. Agglomeration is a process in which small particles are combined to produce larger, permanent masses. The two principal methods of agglomeration used for iron ore are sintering and pelletizing.

Sinter is made by igniting a mixture of fine ore (between 1/4 in and 100 mesh in size), lime or limestone, and coke on a moving horizontal grate. Sinter plants are almost all located adjacent to steel mills because sinter is brittle and deteriorates easily when handled. Another benefit of locating sinter plants near steel mills is that it enables the recovery and the use of steel plant dust and coke breeze, both generated during steelmaking.

Pellets, on the other hand, have excellent handling characteristics and are easily transported. Hence, most pellet plants are located near mines because the fines that comprise pellets are difficult to transport. Pellets are made by combining ore particles less than 100 mesh with a binder, usually bentonite, and then hardening them in furnaces. The pellets produced, generally, have a very high iron content—rarely less than 60 pct and usually 65 pct or more. Pellets are made by rolling ore with controlled moisture content around in a drum or on a rotating inclined disk. Some small pressure is necessary to consolidate the pellets as they form but this comes mainly from their own weight applied to each small particle as it is picked up. They are hardened or indurated by firing at such a temperature that a good bond is produced either by recrystallization of the minerals present or by the formation of glasses.

Initially, magnetite concentrates were pelletized because the heat of reaction constituted a large portion of the necessary process fuel. Now, however, hematite, mixtures of hematite and magnetite, and hematite and limonite can also be pelletized. The exothermic reaction from pelletizing magnetite ore reduces the amount of fuel required and can have a very favorable effect on the economics of an operation. Pellets made from hematite and hematite-limonite ores may require as much as 30 L of fuel per ton of ore, while fuel requirements for magnetite ores are considerably lower.

PRODUCTION COSTS

Table 2 shows operating cost ranges and averages for selected MEC iron ore mines and deposits. Costs are presented by country or region and by production status on a dollars per long ton of ore basis for mining, milling, and transportation.

Mine operating costs typically range from \$1.00/lt to \$5.00/lt in all countries with the exception of Canada, Europe, Mexico, Australia, and the domestic nonproducers. The mining costs in Europe have an average cost of nearly \$7.00/lt and are higher primarily because of the underground mining costs in Sweden. The European countries all have higher ranges and averages because of high labor and energy costs. The South American properties, including Brazil, have the lowest costs with averages from \$1.10/lt to \$1.70/lt. The mining cost range and average for India is low because of the large Kudremukh project, another example of low labor costs.

Specific processing is required for different types of ore, which impacts beneficiation costs. Some ores, specifically the taconites produced in the Lake Superior region and the hematites in Canada, require intense grinding that results in higher energy expenses. As seen in table 2, these cost ranges are higher and have an average beneficiation cost of approximately \$5.00/lt. Beneficiation averages for operating costs in other regions range from \$0.10/lt to \$5.70/lt. The ranges and averages for the Brazilian producer region tend to be low relative to the other regions in both the mining and beneficiation categories. This can be attributed not only to inexpensive labor costs but to the Carajas project, which has vast resources of ore that requires minimum beneficiation.

A variety of transportation methods can be utilized in the transport of iron ore, such as truck, barge, slurry, and rail; however, because of high volume, most of the ore is transported by rail. There are mines that use slurry pipelines in Mexico, India, and Brazil. While not shown in the table, the cost range for mines using slurry as a method of transportation is \$0.10/lt to \$2.00/lt. The costs in this table include all costs for transport of the product from the mine to the port for further resale and do not include ocean transportation. The U.S. data, however, do include shipping on the Great Lakes resulting in higher cost ranges and

Table 2.--Iron ore production costs for producing and nonproducing operations in selected MEC's

(All costs are in January 1985 U.S. dollars per long ton of ore on a weighted-average basis)

Region or country	Mine		Beneficiation		Transportation to port	
	Range	Wtd av	Range	Wtd av	Range	Wtd av
Africa:						
Producers	1.00- 3.20	2.70	0.80- 2.60	1.60	0.10-2.00	1.70
Nonproducers	1.60- 5.10	2.20	1.10- 3.00	1.40	.20-1.90	.90
Australia:						
Producers	1.50- 7.10	2.10	.40- 1.50	1.20	.20-1.20	.90
Nonproducers	1.70- 3.20	2.60	.20- .40	.40	.60-2.00	
Brazil: Producers	.70- 3.70	1.10	.10- 4.00	1.20	.20-3.00	1.00
Canada: Producers	2.10- 7.40	4.30	3.50- 9.20	4.70	.70-1.40	1.10
Europe: Producers	2.50- 7.70	6.80	1.50- 5.20	3.20	.20-2.00	1.00
India: Producers	.80- 4.70	2.00	.50- 1.60	1.20	.10-5.80	1.30
Mexico: Producers	4.50- 7.20	6.00	2.10- 3.70	2.70	.10-3.20	.50
Other South America:						
Producers	1.50- 3.60	1.70	.70- 2.70	.80	.10-2.90	.80
Nonproducers	1.90- 2.10	1.50	.70- .80	.80	.50- .90	.70
United States:						
Lake Superior producers[1]	1.80- 4.70	3.00	3.10- 6.30	4.50	2.70-5.90	4.60
Lake Superior nonproducers[2]	2.60- 6.00	3.10	3.40- 8.00	5.70	1.50-4.60	2.70
Other nonproducers[3]	3.20-12.90	5.00	2.10-10.10	5.00	.10-5.60	4.20

[1]Includes mines in the Mesabi and Marquette Ranges.
[2]Includes mines and deposits in the Mesabi, Marquette, and Gogebic Ranges.
[3]Includes mines and deposits in California, Missouri, Montana, Nevada, New Jersey, New York, Texas, Utah, and Wyoming.

averages for the United States. The average transportation cost for the other regions varies between $0.50/lt and $1.70/lt.

Pelletizing operating cost ranges are shown in table 3 for those countries that have pellet production. Pelletizing is an energy-intensive operation that is directly related to the type of ore being processed. Magnetite ores are the least expensive to pelletize because an exothermic reaction is created during the process, which minimizes the amount of fuel required. Hematite ores are more expensive to process, consuming approximately 85 pct more fuel on a per ton basis than processing of magnetite ores. The pelletizing costs for Canada are about twice those of the other regions because the ores mined are hematite.

Table 3.—Pelletizing operating costs for selected MEC's

(All costs are in January 1985 U.S. dollars per long ton of product on a weighted-average basis)

Region or country	Operating costs, $/lt product	
	Range	Wtd av
Brazil: Producers	12.20–12.80	12.30
Canada: Producers[1]	15.00–22.70	19.30
Europe: Producers	6.30– 8.90	7.50
Mexico: Producers	8.70–12.00	9.40
United States:		
Lake Superior producers	6.20–10.80	8.00
Lake Superior nonproducers	8.20–13.20	8.90
Other nonproducers	6.60–14.90	8.50

[1]Includes mines processing only hematite ores.

AVAILABILITY

The iron ore products of sinter fines, lump ore, pellet feed, and pellets are sold on different price bases that vary by country, company, and contract. The availability curves in this study are presented on a free on board (f.o.b.) basis, in which the total cost includes all costs required to take the iron ore product to the port.

Iron ore is not a homogenous commodity with respect to chemical composition or physical structure, which causes pricing to be complex and reflect the regional variation in ore quality and grade. Most iron ore is bought on long-term contracts with renegotiations of prices annually. On the international level, prices are negotiated f.o.b. with some exceptions in Venezuela, Brazil, and Australia, where it is sold cost and freight (c&f), or cost, insurance, and freight (c.i.f.). A major cost in the iron ore industry is ocean freight. The United States sells iron ore on a delivered basis, which includes the transport cost of shipping on the Great Lakes, resulting in a higher market price than foreign pellet prices. A 1985 market price for domestic pellets delivered at lower lake port was approximately $0.80/lt. Approximate 1985 international market prices, f.o.b., were lump ore, $0.32/iron unit; sinter fines, $0.27/iron unit; pellets, $0.36/iron unit; and pellet feed, $0.28/iron unit.

TOTAL RECOVERABLE

There is approximately 19.4 billion lt of sinter fines potentially available from 73 of the 129 mines and deposits that were evaluated in this study (fig. 3). Of the total, 7.4 billion lt (43 pct) is from the 40 producers while 12.0 billion lt (69 pct) is from the 33 nonproducers. The curves for producing mines show production costs at upper and lower levels. The upper level reflects the total cost including a 15-pct discounted-cash-flow rate of return (DCFROR) while the lower level reflects the costs at 0-pct DCFROR. The curve for the nonproducing deposits shows costs including a 15-pct DCFROR. The 1985 price for sinter fines on international markets was about $0.27/iron unit. Therefore, at a total cost of $0.27/iron unit and a 15-pct DCFROR, there is approximately 3.4 billion lt potentially available from producers (3.9 billion lt at a 0-pct DCFROR) and 2.8 billion lt from nonproducers.

There is approximately 4.8 billion lt of lump ore potentially available from 40 of the 129 mines and deposits evaluated in this study. Figure 4 shows that 3.6 billion lt (75 pct) is potentially available from 29 producers and 1.2 billion lt (25 pct) from 11 nonproducers. The market price for lump ore was approximately $0.32/iron unit in 1985. At a total cost of $0.32/iron unit, including a 15-pct DCFROR, there is approximately 2.9 billion lt of lump ore potentially available from producing mines (3.0 billion at a 0-pct DCFROR) and 0.6 billion lt available from nonproducing deposits.

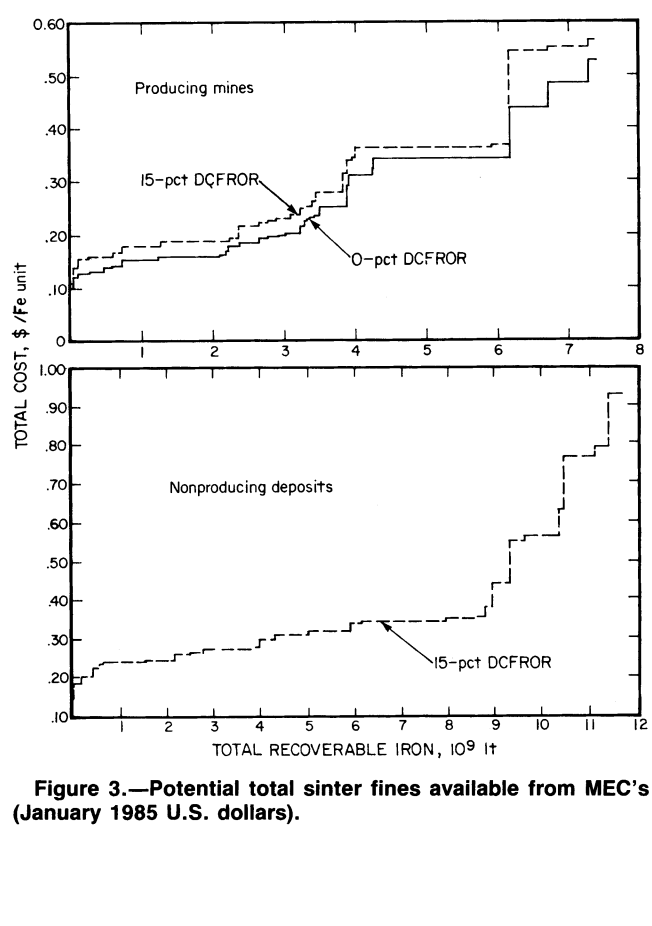

Figure 3.—Potential total sinter fines available from MEC's (January 1985 U.S. dollars).

There is approximately 805 million lt of pellet feed available from 17 mines and deposits in MEC's. Figure 4 shows that approximately 490 million lt (61 pct) is potentially available from nine producers while 315 million lt (39 pct) is potentially available from eight nonproducers. The 1985 market price for pellet feed was approximately $0.28/iron unit, f.o.b. port. At a total production cost (including a 15-pct DCFROR) equal to this price, 68 million lt is potentially available from producers (140 million lt at 0-pct DCFROR) and nearly 170 million lt from nonproducers. Analysis indicates that, over the long run, prices exceeding the 1985 level will be required for producing mines to operate profitably.

Figure 5 shows the potential availability of pellets from foreign and domestic deposits. A total of 14.5 billion lt is potentially available, nearly 80 pct of which is from domestic operations. From foreign deposits, 2.2 billion lt is potentially available from 25 producers and 0.59 billion lt from 6 nonproducers. Market prices for pellets on the foreign market in 1985 were approximately $0.36/iron unit, f.o.b. port. At this price, approximately 2.6 billion lt is potentially available from foreign producers at 15-pct DCFROR, and 590 million lt from nonproducers.

From domestic deposits, 11.3 billion lt of pellets is potentially available, 3.1 billion lt from the 11 producing mines and 8.2 billion lt from the 30 nonproducing deposits. The 1985 domestic price for pellets was approximately $0.80/iron unit. At a total production cost (including a 15-pct DCFROR) equal to the 1985 price, there is approximately 1.8 billion lt available from producers and only 0.26 billion lt from nonproducers. As shown, the price would have to increase above the $1.00/iron unit level before tonnage from most nonproducers would be available.

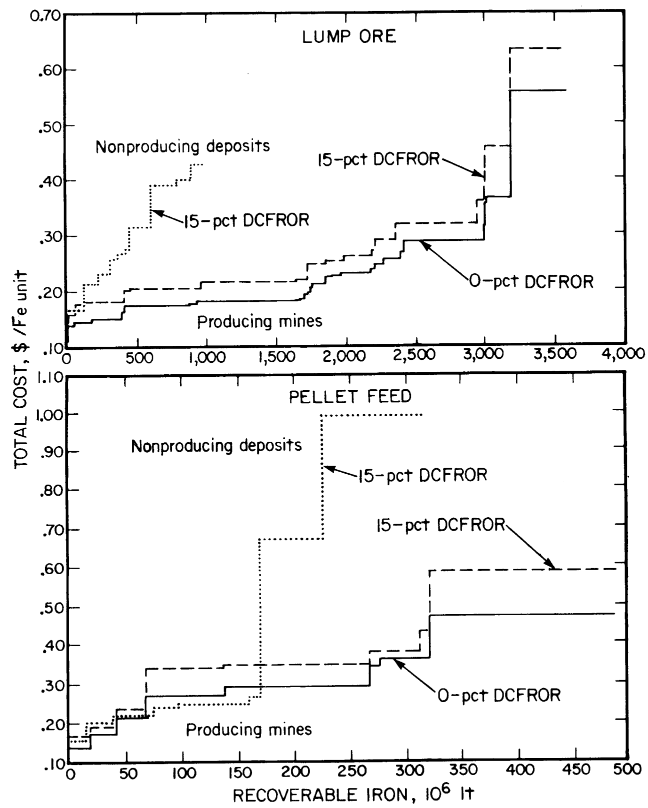

Figure 4.—Potential total lump ore and pellet feed available from MEC's (January 1985 U.S. dollars).

ANNUAL CAPACITY

Potential 1985 annual production of iron ore products from major regions representing specific markets are shown in figures 6 and 7. Availability is shown on a regional basis to illustrate what is available at given market prices because prices are established by product and region. Illustrated for each of the selected regions is a production potential cost curve (at full capacity levels) at a 0-pct DCFROR level along with the approximate 1985 market price for the given region.

At production cost levels (0-pct DCFROR) that are less than or equal to the 1985 price of $0.80/iron unit for domestic pellets, there is approximately 44 million lt of pellets potentially available from producing U.S. mines. Figure 6 illustrates that nearly 36 million lt of lump ore is potentially available in Brazil from producers at cost levels less than the 1985 market price for lump ore.

A comparison of the 1985 production potential of sinter fines from Brazil, Australia, and Africa in figure 7 shows there is approximately 66 million lt, 65 million lt, and 22 million lt, respectively, at total costs less than a 1985 market price of $0.27/iron unit. While both Brazil and Australia show that all of the available sinter fines are at less than the market price, Africa has an additional 15 million lt available at total costs that are greater than the market price.

Figures 8 and 9 show the potential annual production of iron ore products through the year 1995 from the demonstrated resources of producing mines. The curves show on an annual basis the potential production at various

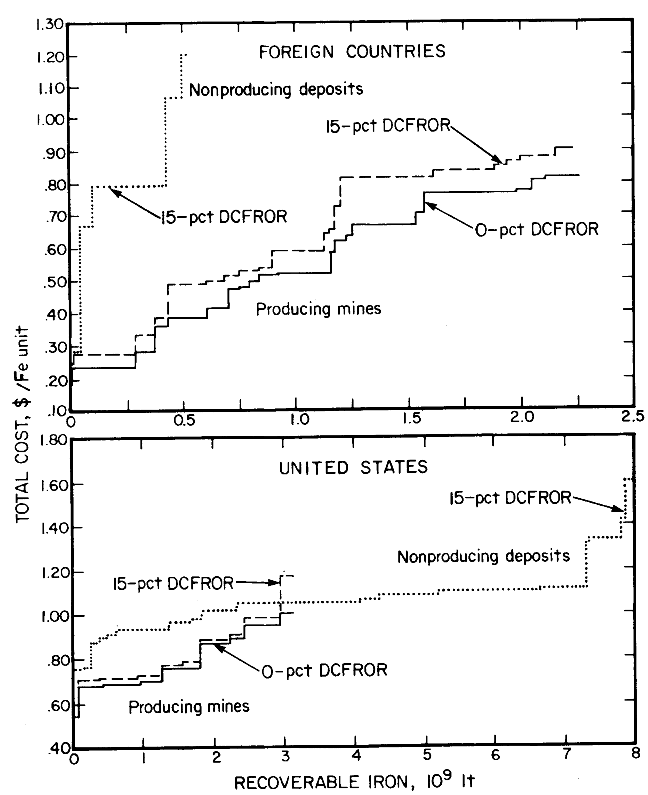

Figure 5.—Potential total pellets available from MEC's (January 1985 U.S. dollars).

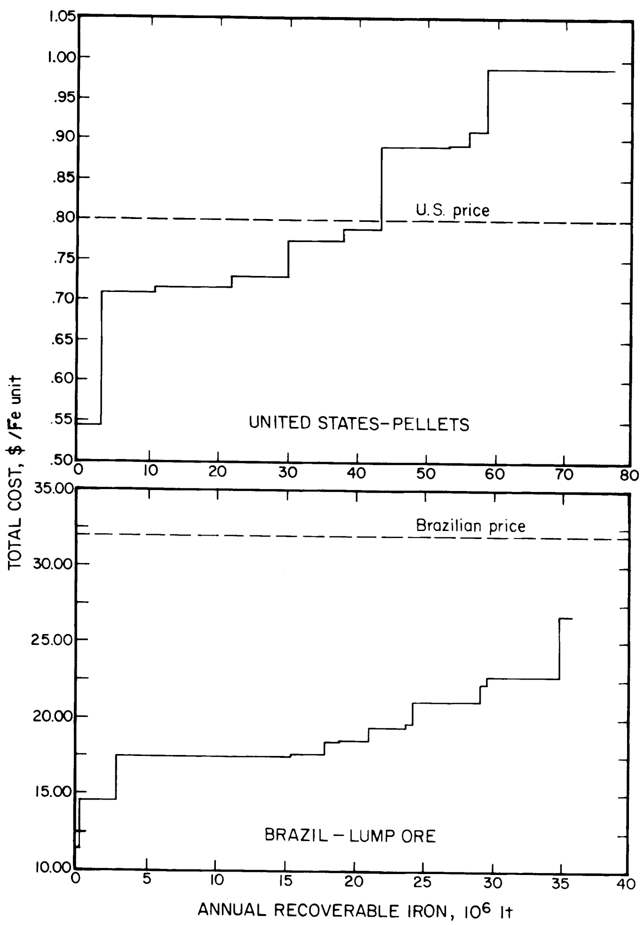

Figure 6.—1985 pellet and lump ore capacity from producing mines in the United States and Brazil (January 1985 U.S. dollars).

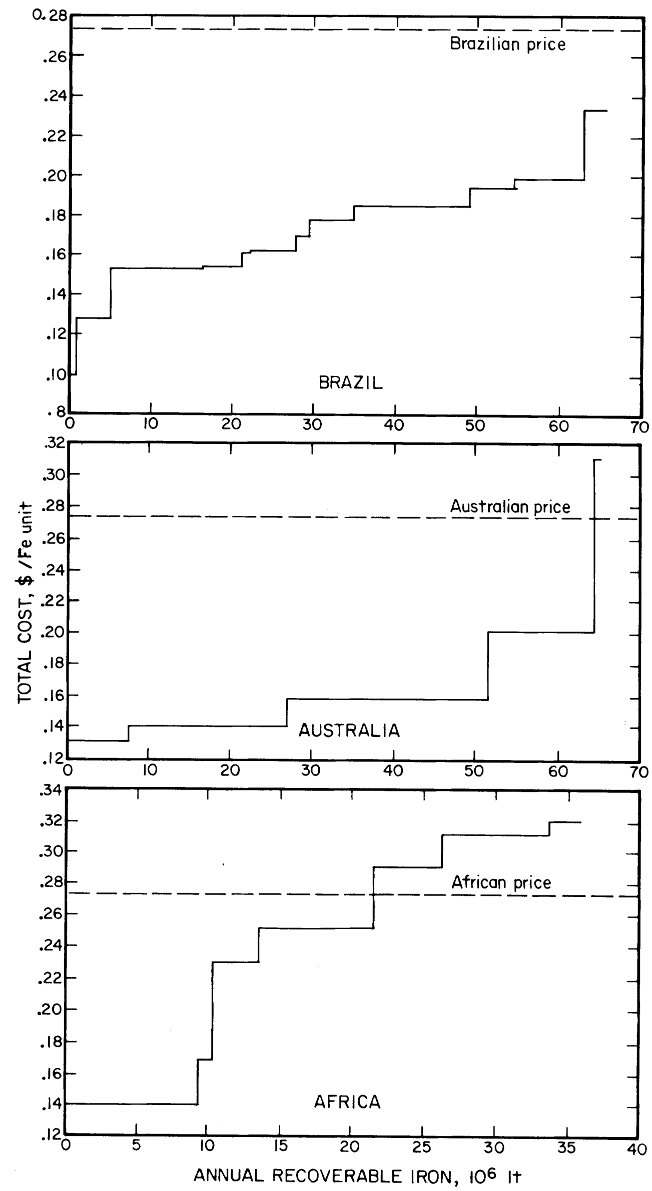

Figure 7.—1985 sinter fine capacity from producing mines in Brazil, Australia, and Africa (January 1985 U.S. dollars).

total cost levels (including a 15-pct DCFROR) for each product assuming that all operations produce at full capacity (100 pct) over the life of the mine.

The curves for domestic pellets in figure 8 show that through the year 1995 the availability of this product does not decline, while foreign production declines slightly at cost levels of $0.40/iron unit. These curves indicate that there is an adequate supply of this product in both regions. At a 1985 market price of $0.80/iron unit for domestic pellets there is approximately 44 million lt of pellets available in 1985. Comparing potential 1985 availability of pellets for foreign countries to the market price of $0.36/iron unit, there is over 9 million lt available at a total cost of $0.30/iron unit, while there is over 16 million lt available at less than a total cost of $0.40/iron unit.

The annual curves for sinter fines, lump ore, and pellet feed in figure 9 are somewhat different in that they show a decline in the available tonnage through the year 1995. The decrease in the available tonnage on some of the curves is caused by depletion of available demonstrated resources. There are several factors that may affect some of these declines. Operations may actually produce at less than the rated capacity (as they currently are), which would cause depletion to occur less rapidly. Inferred resources may become better defined resulting in additional resources at the demonstrated level. In addition, new discoveries could add resources to the already existing vast iron ore resources.

The curves indicate that there was over 140 million lt of sinter fines potentially available in 1985 at less than a total cost of $0.25/iron unit. There was nearly 70 million lt of lump ore potentially available at less than a total cost of $0.25/iron unit in 1985. Finally, there was almost 4 million lt of pellet feed potentially available in 1985 at less than a total cost of $0.30/iron unit.

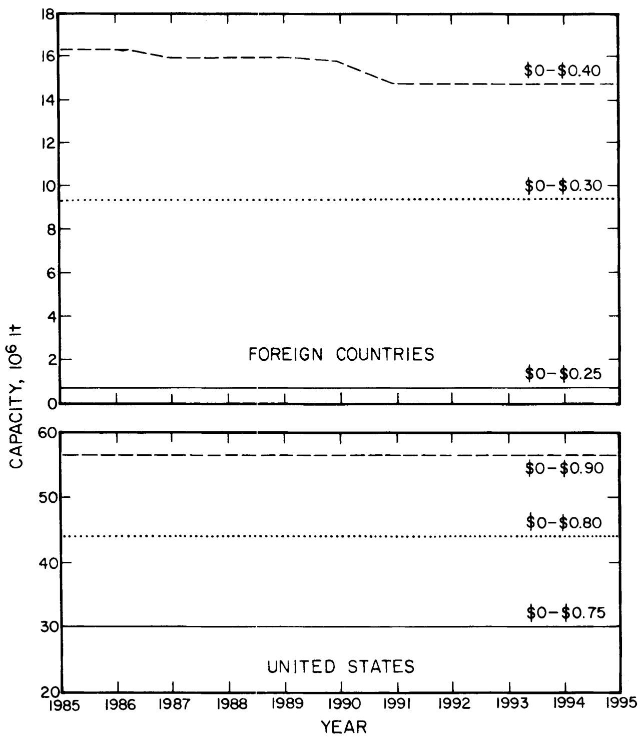

Figure 8.—Potential annual pellet production from producing mines in the United States and foreign countries at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

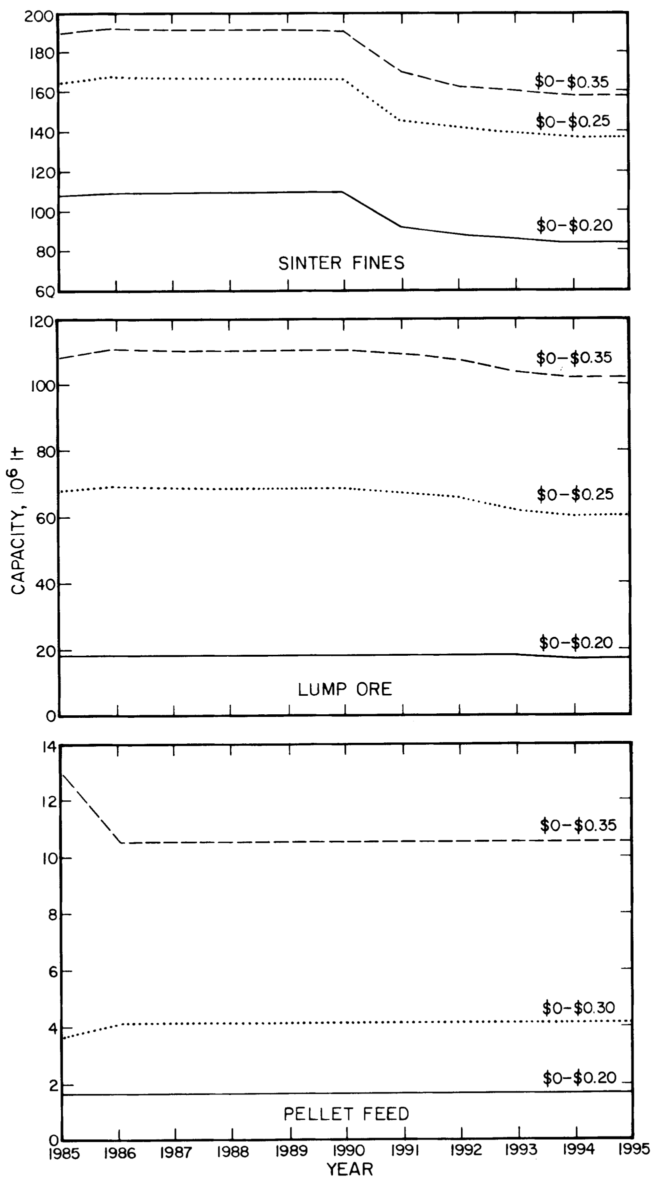

Figure 9.—Potential annual sinter fines, lump ore, and pellet feed production from producing mines in MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

CONCLUSIONS

The world's iron and steel industry is primarily dependent upon a supply of iron derived from a variety of iron ore deposits. This analysis shows that the supply of demonstrated resources of iron ore from producing deposits in MEC's is more than adequate to satisfy the demand for most iron products through the year 1995. This study shows that not only do the producing mines have an adequate supply but also that operations that have not been developed contain adequate reserves for future demands.

Of the marketable iron ore products potentially available, sinter fines compose the greatest share, totaling 19.4 billion lt. Iron ore pellets are second with approximately 3.2 billion lt available from foreign properties and 11.3 billion lt from domestic operations. The potential availability of lump ore is 4.8 billion lt and approximately 805 million lt of pellet feed is potentially available.

On an annual basis, there is less than 50 million lt of pellets available domestically at less than the 1985 market price. While the U.S. market for pellets is very localized, foreign countries compete in world markets for the other products. An example is over 35 million lt of lump ore in Brazil being potentially available at less than market prices. All of the sinter fines in Brazil (66 million lt) and Australia (65 million lt), and most in Africa (23 million lt) are available at less than the 1985 market prices, which allows Brazil and Australia to remain very competitive on foreign markets.

In the past several years many factors have affected the iron ore industry, changes in the world economy, high inflation, high labor rates in developed countries, and high energy costs, affecting beneficiation and transportation. Compounding these problems have been the added effect of the decline in the steel demand domestically and continuing imports of foreign steel. With the stabilization of inflation and the decline in energy costs, the general outlook for the iron ore industry appears somewhat more encouraging than in the past few years with the exception of the domestic industry. Because the domestic iron ore industry is directly linked to the U.S. steel industry, there is a general consensus that the continuing contraction of the U.S. steel industry, due to increasing imports of foreign steel, will result in a synergistic adverse effect on the domestic iron ore industry.

In summary, the iron ore industries of the world have substantial resources of iron ore to satisfy the demand for many years into the future. However, the revitalization of the industry on the domestic scene is directly linked to what happens to the U.S. steel industry.

REFERENCES

1. Bolis, J. L., and J. A. Bekkala. Iron Ore Availability—Market Economy Countries. A Minerals Availability Appraisal. BuMines IC 9128, 1987, 56 pp.
2. U.S. Bureau of Mines. Iron Ore. Mineral Industry Surveys, monthly and annual.
3. Klinger, F. L. Iron Ore. Ch. in Mineral Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 385-403.
4. Kuck, P. H. Iron Ore. Ch. in BuMines Minerals Yearbook, 1985, v. 1, pp. 517-540.
5. Klinger, F. L. Iron Ore. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 78-79.
6. Marsden, R. W. Estimation and Evaluation of the Iron Ore Reserves of the United States—A Partial Survey 1974-75. Iron Ore Reserves of the Mesabi Range, Minnesota. A Minerals Availability System Report (contract J0188074, Univ. MN). BuMines OFR 109-80, 1980, 81 pp.; NTIS PB 80-228984.

LEAD AND ZINC

INTRODUCTION

The major uses of lead are in storage batteries and in materials for the metalworking and construction industries. The use in storage batteries makes it a necessary material for both civilian and military purposes.

Lead resources are abundant throughout the world, with the United States, Australia, Canada, and the U.S.S.R. producing roughly half of world production. Based on the properties included in this study, the United States presently has one-fifth of the demonstrated resources of lead in market economy countries (MEC's). Because of its status as the world's largest recycler of lead scrap, the United States has been able to provide nearly 50 pct of its demand requirement from scrap in some years.

At the present time, about 90 pct of the lead ore in the United States is mined in Missouri. The proximity and relative ease of accessibility of lead mines in Canada and Mexico, and the modern and efficient status of mines, mills, smelters, refineries, and major manufacturing plants, renders a relatively low risk of supply disruption for the United States.

The major uses of zinc are as an alloying metal with copper to make brass, in chemical compounds for rubber and paint, as protective coatings on steel, and as diecastings. Because there is no effective substitute for zinc for corrosion protection for iron and steel, it is a necessary material for both civilian and military use.

Zinc resources are abundant throughout the world, with Canada, the U.S.S.R., Australia, Peru, and the United States being major producing countries. Based on the properties included in this study, the United States has about one-third of the demonstrated resources of zinc in MEC's. In the last 16 yr, the United States has gone from a position of near self-sufficiency in zinc smelter production to one requiring large metal imports, owing to the development of foreign deposits and smelters. Because of the proximity and relative accessibility of zinc metal in Canada and Mexico (which provide 60 pct of U.S. imports), the risk of supply disruption is fairly low to the United States.

The major lead and zinc minerals are the sulfides galena (PbS) and sphalerite (ZnS). Lead and zinc are typically found either individually or in combination with each other or with copper and/or silver. The final product from the mill is either a lead and/or zinc concentrate. The concentrate is then processed at a smelter and refinery to produce a refined metal product. This study analyzes lead and zinc through the refinery to the production of metal. Transportation costs to the smelter and refinery are included.

Most of the information presented in this chapter is an updated summary of Bureau of Mines Information Circular 9026 "Primary Lead and Zinc Availability—Market Economy Countries. A Minerals Availability Program Appraisal" (*1*).[1] Additional information on the domestic and foreign lead and zinc industries is available from other Bureau of Mines publications (*2-9*).

GEOLOGY AND RESOURCES

GEOLOGY

The minerals in lead and zinc ores are relatively simple, with the sulfides galena (PbS) and sphalerite (ZnS) occurring as the major minerals, respectively. Most lead deposits contain galena associated with sphalerite, pyrite (FeS_2), chalcopyrite ($CuFeS_2$), and other base metal sulfides or sulfosalts, some of which are recovered to yield byproducts or coproducts. Galena usually contains variable amounts of silver (argentiferous galena) (*2*). Sometimes galena is altered and minable in ore bodies consisting of cerussite ($PbCO_3$), anglesite ($PbSO_4$), or other oxidized lead minerals but, in general, galena is resistant to chemical alteration. Lead is a major constituent in several important deposits types, including volcanogenic stratabound, replacement, vein, and contact metamorphic deposits.

Most sphalerite has associated cadmium in quantities from traces to 2 pct, and small quantities of germanium, gallium, indium, and thallium. A few important zinc deposits contain oxide, carbonate, or silicate zinc minerals such as zincite (ZnO), smithsonite ($ZnCO_3$), willemite (Zn_2SiO_4), or hemimorphite ($Zn_4Si_2O_7(OH)_2 \cdot H_2O$); commonly derived from the altered sulfide minerals. Most zinc ores mined occur in three types of deposits: massive sulfides, breccia-filled replacement, and volcanogenic stratabound.

Massive sulfide deposits occur in two major types of host rock: limestone or dolomites and volcanic or other sedimentary (e.g., sandstone) type rocks. Included in the first type of deposit are the Mississippi Valley type (MVT) ore deposits, which represent the majority of U.S. lead-zinc deposits. These deposits generally range more extensively in the lateral than in the vertical direction and are predominated by open-space fillings.

[1]Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

Sedimentary-structural features, such as reefs, facies changes, zones of minor jointing, or collapse breccias associated with ancient karst drainage, serve as loci for ore bodies within the favorable formations. Examples of such stratabound deposits are in the Southeast Missouri lead district; the Missouri-Oklahoma-Kansas district; the Upper Mississippi Valley district; the Pine Point deposit, Northwest Territory, Canada; the Laisvall deposit, Sweden; and the central and eastern Tennessee zinc deposits.

Volcanogenic-stratabound type deposits contain massive sulfide bodies commonly interlayered with volcanic or sedimentary rocks. Most such deposits are found in folded and disturbed belts that have been metamorphosed. Their size can range from small lenses to enormous masses. The ore is commonly a fine-grained mixture of pyrite or pyrrhotite, sphalerite, galena, and chalcopyrite, with minor amounts of nonmetallic and carbonate minerals. Examples of massive sulfide deposits are those near Bathurst, Kidd Creek, and Sullivan in Canada; Broken Hill and Mount Isa, Australia; and Kuroko, Japan.

Replacement deposits of lead and zinc are commonly irregular hydrothermal type deposits in carbonate rocks, but some also occur in quartzites or metamorphic rocks. The form and extent of the ore bodies are determined by the structural and stratigraphic elements that localized the replacement activity of the ore-bearing solutions. They include tabular or cylindrical flat-lying bodies called mantos, pipelike structures that cross the bedding, and irregular branching bedded deposits associated with veins. Some of the well-known replacement deposits include Cerro de Pasco, Peru; the silver lead district of Central Mexico; Tsumeb, Namibia; Tintic, Utah; and Leadville and Gilman districts, Colorado.

Veins are the best known type of ore deposits. They are the most obvious and consequently were the first deposits to be exploited by ancient miners. Lead and zinc vein deposits are commonly situated in faults, joints, or at formational contacts. They contain ore and gangue minerals in varying amounts. Veins can be 1 to 10 m long horizontally, and extend downwards hundreds of meters. Some of the better known lead-zinc-silver-copper vein systems occur in the Coeur d'Alene district in Idaho; the Silverton area of Colorado; the Santa Barbara, Fresnillo, and Taxco Mines in Mexico; and the German Harz Mountains, Clausthal, and Freiberg deposits.

Contact metamorphic deposits are associated with igneous intrusions, which have either provided the solutions or emanations creating the deposit, or have altered and recrystallized (or replaced) a mineral deposit present prior to the intrusion. Deposits range in size from small vein systems to massive pods, hundreds of meters long. The Kamioka, Obori, Chichibu, Kuroko, and Nakatatsu deposits in Japan are examples of this type of deposit. Although many deposits of this type are mined for other metals in the United States, only a few have produced significant amounts of lead, usually as a byproduct (*1*).

RESOURCES

Lead

Demonstrated resources of contained lead are estimated to be approximately 25 million mt (table 1) for properties evaluated for primary lead. An additional 60.2 million mt of contained lead is available as a byproduct or coproduct from zinc properties (table 1) and 4.2 million mt as a byproduct or coproduct of copper and silver production.

As shown on figure 1, for the properties evaluated in this study, the United States accounts for approximately 25 pct (22.4 million mt) of the total contained lead for both primary and byproduct properties, while Australia accounts for 24 pct (21.6 million mt) and Canada 18 pct (16.2 million mt). The reserve base was reported to be approximately 115 million mt for MEC's.

Table 1.—Summary of MEC demonstrated lead resources evaluated for this study, as of January 1985

Country	Mines-deposits	In situ resources, 10^3 mt	Grade, pct	Lead, 10^3 mt Contained	Lead, 10^3 mt Recoverable
PRIMARY LEAD PROPERTIES					
Australia	1	5,882	12.60	741	689
Mexico	2	6,605	6.25	413	288
Morocco	5	14,712	7.77	1,143	953
Namibia	1	5,000	6.98	349	268
South Africa, Republic of.	2	119,150	3.57	4,255	3,402
Spain	1	1,250	5.56	69	57
Sweden	2	32,201	4.40	1,416	1,049
United States	12	289,417	5.74	16,603	15,390
Total or av	26	474,218	5.15	24,991	22,096
PRIMARY ZINC PROPERTIES					
Algeria	1	1,462	1.33	20	14
Argentina	1	4,326	6.89	298	241
Australia	11	397,570	5.09	20,234	11,917
Austria	1	6,595	1.38	91	63
Bolivia	1	281	1.13	3.2	2.3
Brazil	1	17,613	1.48	261	181
Burma	1	1,860	6.25	116	80
Canada	24	477,083	3.31	15,819	11,100
Finland	1	6,485	.30	20	14
France	4	8,515	1.61	137	69
Germany, Fed. Rep.	3	18,360	2.52	436	307
Greece	2	15,800	3.5	553	382
Greenland	1	726	4.40	32	23
Honduras	1	5,131	4.22	216	159
India	5	166,040	1.92	3,180	1,952
Ireland	4	48,975	1.9	931	614
Italy	5	29,098	1.49	433	280
Japan	8	50,345	.64	324	207
Mexico	18	204,533	1.51	3,092	2,334
Morocco	1	1,500	1.50	23	14
Namibia	1	3,398	2.0	68	53
Peru	18	84,166	2.92	2,460	1,697
Portugal	1	137,109	1.16	1,590	997
South Africa, Republic of.	2	336,540	.57	1,929	519
Spain	5	159,400	1.17	1,867	759
Sweden	2	16,974	2.15	364	276
Turkey	1	338	1.20	4.1	2
United States	19	308,596	1.79	5,550	4,665
Zambia	1	933	11.3	105	85
Total or av	144	2,510,362	2.40	60,185	39,008

NOTE.—Data may not add to totals shown because of independent rounding.

Zinc

Demonstrated resources of contained zinc are estimated to be 191.4 million mt (table 2) from primary zinc deposits. An additional 22.7 million mt is available as a byproduct or coproduct from lead, copper, and silver properties (which were evaluated in this study). As shown on figure 2, for the properties evaluated in this study, Canada accounts for nearly 21 pct (44.1 million mt), Australia 21 pct (45.6 million mt), and the United States 22 pct of the total (47.4 million mt). World identified resources of zinc are estimated to be 1.8 billion mt and the world reserve base is estimated at 300 million mt. Nearly 90 pct of the reserve base occurs in MEC's (*5*).

Tables 3 through 6 list the primary lead, zinc, copper, and silver properties included in this investigation.

Table 2.—Summary of MEC demonstrated zinc resources from primary zinc properties evaluated for this study, as of January 1985

Country	Mines-deposits	In situ resources, 10^3 mt	Grade, pct	Zinc, 10^3 mt	
				Contained	Recoverable
Algeria	1	1,462	5.56	81	59
Argentina	1	4,326	8.44	365	281
Australia	11	397,570	9.55	39,983	21,066
Austria	1	6,595	4.75	313	238
Bolivia	2	891	9.27	83	57
Brazil	2	21,557	7.63	1,646	1,102
Burma	1	1,860	5.00	93	57
Canada	30	538,955	6.84	36,864	26,133
Finland	2	19,172	2.83	542	466
France	4	8,515	7.69	655	425
Germany, Fed. Rep.	3	18,360	8.9	1,636	1,372
Greece	2	15,800	4.5	711	487
Greenland	1	726	13.40	97	74
Honduras	1	5,131	8.0	411	288
India	5	166,040	7.62	12,644	6,403
Ireland	4	48,975	8.69	4,254	2,923
Italy	5	29,098	4.35	1,267	786
Japan	8	50,345	4.92	2,478	1,711
Mexico	19	204,840	3.55	7,263	4,846
Morocco	1	1,500	6.5	98	76
Namibia	1	3,398	7.0	238	183
Peru	18	84,166	6.72	5,659	4,235
Portugal	1	137,109	3.06	4,196	2,267
South Africa, Republic of.	2	336,540	5.04	16,967	8,303
Spain	5	159,400	2.81	4,485	3,220
Sweden	2	16,974	10.69	1,815	1,460
Turkey	1	338	24.50	83	75
United States	50	952,523	5.56	43,190	37,063
Zaire	1	40,114	12.62	5,063	2,504
Zambia	1	933	22.5	208	176
Total or av	186	3,273,215	5.85	191,384	128,336

NOTE.—Data may not add to totals shown because of independent rounding.

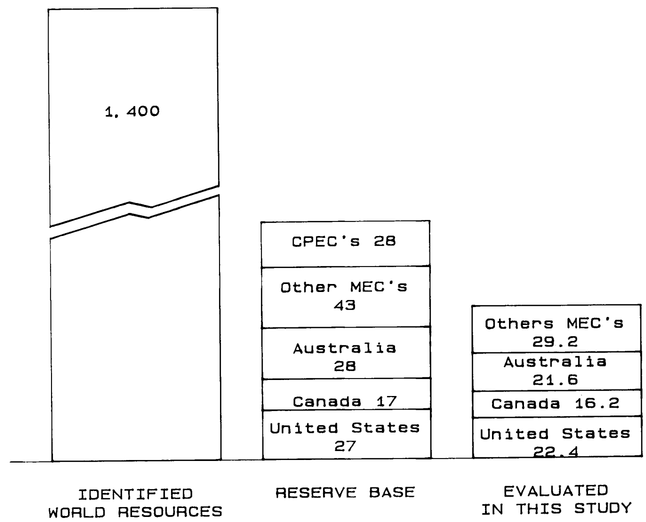

Figure 1.—Estimates of world lead resources (million metric tons of contained lead).

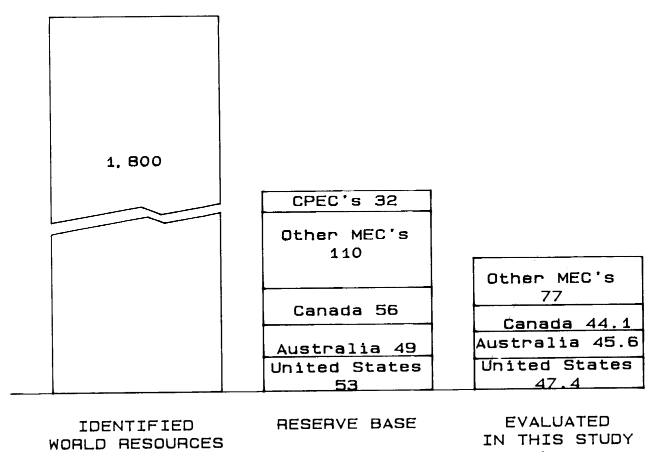

Figure 2.—Estimates of world zinc resources (million metric tons of contained zinc).

Table 3.—MEC primary lead properties included in this study

Country and deposit name	Owner	Current status[1]	Type[2]	Est ore capacity, 10^3 mt/yr	Mining methods
Australia: North Broken Hill	North Broken Hill Ltd	P	U	500	Combined methods.
Mexico:					
La Encantada	La Encantada S A	P	U	355	Do.
Rosario	Industria Minera Mexico S A	D	U	235	Cut and fill.
Morocco:					
Aouli-Mibladen	Penarroya	P	C	150	Combined methods.
Djebel Aouam	BRPM-Royal Asturienne-Vielle	P	U	200	Do.
Sidi Lachen	BRPM-Armico	D	U	84	Overhand.
Touissit	Royal Asturienne des Mines	P	U	400	Room and pillar.
Zeida	SODIM-ZELLIDJA-BRPM	P	S	1,200	Open pit.
Namibia: Tsumeb	Tsumeb Corp. Ltd	P	U	450	Cut and fill.
South Africa, Republic of:					
Black Mountain	Phelps Dodge-GFSA	E	U	3,440	Do.
Broken Hill	..do	P	U	1,800	Do.
Spain: Linares (El Cobre)	Cia Minera la Cruz	P	U	145	Do.
Sweden:					
Laisvall	Boliden Metall AB	P	U	1,500	Room and pillar.
Vassbo-Cuttusjo	..do	P	U	300	Do.
United States:					
Colorado: Bulldog	Homestake Mining	P	U	102	Cut and fill.
Idaho: Lucky Friday	Hecla Mining Co	P	U	168	Do.
Missouri:					
Boss-Bixby	Getty Oil-AZCON-Hanna Mining	E	U	136	Room and pillar.
Brushy Creek Division	St. Joe Minerals Corp	PP	U	1,134	Do.
Buick	Homestake Lead	P	U	1,932	Do.
Fletcher Division	St. Joe Minerals Corp	P	U	1,134	Do.
Frank R. Milliken	Kennecott (Ozark Lead Co.)	P	U	2,058	Do.
Magmont	Cominco American-Dresser	P	U	952	Do.
Viburnum No. 28 and No. 29	St. Joe Minerals Corp	P	U	1,769	Do.
Viburnum No. 35	..do	D	U	867	Do.
West Fork	ASARCO	D	U	454	
Utah: Ontario	United Park City-Noranda	T	U	212	Combined methods.

[1]D, developing deposit; E, explored deposit; P, producing mine; PP, past producer; T, temporarily shut down.
[2]C, combined surface and underground; S, surface; U, underground.

Table 4.—MEC primary zinc properties included in this study

Country and deposit name	Owner	Current status[1]	Type[2]	Est ore capacity, 10^3 mt/yr	Mining methods
Algeria: El Abed	SONAREM	P	U	500	Room and pillar.
Argentina: El Aguilar	St. Joe Minerals Corp	P	U	600	Overhead square set.
Australia:					
Dugald River	CRA Ltd	E	U	1,000	Open stope.
Elura	EZ Industries Ltd	D	U	1,100	Do.
Hilton	Mount Isa Mines (MIM) Ltd	D	U	710	Combined methods.
Lady Loretta	Triako Mines NL-MIM Holdings	E	U	710	Open stope.
McArthur River	MIM Holdings Ltd	E	C	7,000	Combined methods.
Mount Isa	MIM Ltd	P	U	3,600	Do.
New Broken Hill	CRA Ltd	P	U	1,100	Do.
Que River	Aberfoyle Ltd.-Paringa Mining	P	U	200	Open stope.
Roseberry-Hercules	EZ Industries Ltd	P	U	675	Combined methods.
Woodlawn	St. Joe-Phelps Dodge-CRA Ltd	P	S	1,050	Do.
Zinc Corporation	Zinc Corp. Ltd	P	U	900	Cut and fill.
Austria: Bleiberg-Kreuth	Bleiberger Bergwerks Union	P	U	500	Combined methods.
Bolivia:					
Matilde	COMIBOL	P	U	245	Cut and fill.
Quechisla	..do	P	U	161	Shrinkage.
Brazil:					
Paracatu	Mineracao Morro Agudo	D	U	825	Room and pillar.
Vazante	Companhia Mineria de Metais	P	S	420	Bench (berm).
Burma: Bawdwin	No. 1 Mining Corp	P	S	310	Do.
Canada:					
Abcourt-Barvue	Abcourt Silver Mines-Noranda	E	U	350	Cut and fill.
Anvil Range	Cyprus Anvil Mining Corp	P	C	3,400	Open pit.
Brunswck No. 12	Brunswick Mining and Milling	P	U	3,500	Cut and fill.
Buttle Lake	Westmin Res. Ltd.-Brascan Ltd.	P	U	300	Do.
Caribou Mine	Anaconda Canada Ltd	T	U	2,800	Do.
Chisel Lake	Hudson Bay Mining and Smelting	P	U	181	Sublevel open stope.
Cirque	Cyprus Anvil-Hudson Bay	E	U	1,750	Cut and fill.
Daniels Harbor	Teck Corp	P	U	529	Room and pillar.
Gallen	Noranda-MacDonald	P	S	350	Do.
Gays River	Canada Wide Mines Ltd	T	U	286	Room and pillar.
Goldstream	Noranda Mines Ltd	D	C	496	Combined methods.
Goz Creek	Barrier Reef Resources Ltd	E	S	544	Open pit.
Great Slave Reef	Westmin-Dupont-Philipp Bros	E	U	350	Room and pillar.
Hackett River	Bathurst Norsemines-Cominco	E	U	1,500	Cut and fill.
Half Mile Lake	Texasgulf Inc	E	U	1,000	Do.

See footnotes at end of table.

Table 4.—MEC primary zinc properties included in this study—Continued

Country and deposit name	Owner	Current status[1]	Type[2]	Est ore capacity, 10^3 mt/yr	Mining methods
Canada—Con.					
Heath Steel (Little River Joint Venture)	Heath Steel-Noranda-ASARCO	P	U	1,570	Open stope.
Howard's Pass	Placer Dev.-U.S. Steel Corp	E	U	3,500	Combined methods.
King Fissure	Internat. Standard Resources	E	U	350	Cut and fill.
Lyon Lake	Noranda Mines Ltd	P	U	350	Do.
Mattabi	Noranda-Abitibi Mines Ltd	P	U	508	Sublevel open stope.
Mattagami Lake	Noranda Mines Ltd	P	U	1,476	Cut and fill.
Mel	St. Joseph-Sovereign Metals	E	U	495	Do.
Nanisivik	Mineral Resource Internat	P	U	562	Room and pillar.
Pine Point	Pine Point Mines Ltd.	P	S	3,290	Open pit.
Polaris	Cominco Ltd.	D	U	749	Cut and fill.
Prairie Creek	Cadillac Procan Explorations	D	U	315	Do.
Restigouche	Placer Development	E	S	350	Open pit.
Robb Lake	Texasgulf-Arrow Inter Am-Bar	E	U	450	Room and pillar.
Sullivan	Cominco Ltd	P	U	2,130	Sublevel caving.
Tom	Hudson Bay Mining and Smelting	E	U	700	Cut and fill.
Finland:					
Pyhasalmi	Outokumpu Oy	P	U	1,100	Sublevel open stope.
Vihanti	..do	P	U	1,000	Do.
France:					
Bodennec	BRGM	E	S	260	Open pit.
Malines	Penarroya	P	U	260	Cut and fill.
Porte-Aux-Moines	BRGM	E	U	182	Sublevel open stope.
Saint Salvy	Penarroya	P	U	240	Cut and fill.
Germany, Fed. Rep. of					
Grund	Preussag AG Metail	P	U	407	Do.
Meggen	Sachtleben Bergbau Gmbh	P	U	850	Sublevel open stope.
Rammelsburg	Preussag AG Metall	P	U	277	Cut and fill.
Greece:					
Mavres Petres-Madem Lakos	Hellenic Chemical Prod. Co.	P	U	550	Sublevel caving.
Olympias	..do	P	U	800	Do.
Greenland: Black Angel	Cominco Ltd	P	U	640	Room and pillar.
Honduras: El Mochito	Rosario Resources-Amax Inc.	P	U	700	Combined methods.
India:					
Ambaji	Gujarat Mineral Dev.	D	C	350	Open pit
Mochia-Balaria	Hindustan Zinc, Ltd.	P	U	1,040	Top slicing.
Rajpura-Dariba	..do	D	U	900	Open slope.
Rampura Agucha	..do	E	S	980	Open pit.
Zawarmala-Baroi	..do	E	U	900	Shrinkage.
Ireland:					
Ballinalack	Noranda-Barymin	E	U	350	Sublevel.
Bula	Bula Ltd-Govt. of Ireland	E	U	700	Cut and fill.
Sabina-Tatestown	Sabina-Messina-Irish Base Met.	E	U	350	Room and pillar.
Tara (Navan)	Tara Exp.-Govt. of Ireland	P	U	2,250	Combined methods.
Italy:					
Funtana Raminosa	SAMIM	P	U	350	Sublevel open stope.
Masua	..do	P	U	950	Sublevel caving.
Monteponi	..do	D	U	300	Combined methods.
Montevecchio	..do	E	U	300	Cut and fill.
Ralbl	..do	P	U	468	Combined methods.
Japan:					
Ezuri	Dowa Mining Co. Ltd.	P	U	120	Top slicing.
Fukazawa	..do	P	U	240	Do.
Hosokura	Mitsubishi Metal Corp.	P	U	444	Cut and fill.
Kamioka:					
Mozumi	Mitsui Mining & Smelting	P	U	329	Sublevel open stope.
Tochibora	..do	P	U	954	Do.
Kosaka	Dowa Mining Co. Ltd.	P	U	546	Cut and fill.
Nakatatsu	Nippon Zinc Mining Co. Ltd.	P	U	384	Do.
Toyoha	..do	P	U	396	Do.
Mexico:					
Charcas	Industrial Minera Mexico	P	U	390	Combined methods.
Cuale	Cia Fresnillo S.A.	P	S	300	Open pit.
El Monte-El Carrizal	..do	P	U	140	Open stope.
El Orito	Cia Minera Real de Asientos	P	U	150	Cut and fill.
El Tecolote	Industrial Minera Mexico	P	U	124	Sublevel caving.
Fresnillo	Cia Fresnillo S.A.	P	U	241	Combined methods.
Gochico	Industrias Penoles S.A.	P	U	150	Do.
La Negra	..do	P	U	240	Sublevel open stope.
Naica	Cia Fresnillo S.A.	P	U	756	Cut and fill.
Parral	Industrial Minera Mexico	P	U	218	Combined methods.
Real De Angeles	Minera Real de Angeles	D	S	3,500	Open pit.
Rey de la Plata	Industrias Penoles S.A.	P	U	325	Combined methods.
San Francisco Del Oro	Frisco S A de C V	P	U	852	Shrinkage.
San Martin	Industrial Minera Mexico	P	U	770	Cut and fill.
Santa Barbara	..do	P	U	1,622	Combined methods.
Santa Eulalia	..do	P	U	312	Cut and fill.
Santa Maria de la Paz	Min. Santa Maria de la Paz	P	U	1,026	Shrinkage.
Taxco	Industrial Minera Mexico	P	U	1,034	Combined methods.
Velardena	..do	P	U	281	Shrinkage.
Morocco: Bou Madine	Government of Morocco	E	U	60	Overhand.
Namibia: Rosh Pinah	Imcor Zinc (Pty.) Ltd	P	U	420	Sublevel open stope.

See footnotes at end of table.

Table 4.—MEC primary zinc properties included in this study—Continued

Country and deposit name	Owner	Current status[1]	Type[2]	Est ore capacity, 10^3 mt/yr	Mining methods
Peru:					
Atacocha	Cia Minera Atacocha S A	P	U	460	Combined methods.
Carahuacra	Volcan Mines Co	P	C	320	Do.
Carhuacayan:	Sindicato Minero Rio Pallanga	P	U	315	Shrinkage.
Casapalca	CENTROMIN	P	U	892	Do.
Cerro de Pasco	..do	P	C	2,113	Do.
Colquijirca	Sociedad Minera el Brocal S.A.	P	S	270	Open pit.
Hercules	Cia. Minera Alianza S A.	P	U	432	Room and pillar.
Huachocolpa	Compania de Minas Buenaventura	P	U	100	Shrinkage.
Huanzala	Cia. Minera Santa Luisa S.A.	P	U	273	Cut and fill.
Huaron	Cia. Minera Huaron S.A.	P	U	600	Combined methods.
Madrigal	Homestake Mining	P	U	547	Do.
Milpo	Cia. Minera Milpo S.A.	P	U	540	Cut and fill.
Morococha	CENTROMIN	P	U	513	Combined methods.
Raura	Cia. Minera Raura S A	P	U	350	Do.
San Cristobal	CENTROMIN	P	C	636	Do.
San Vicente	San Ignacio de Morococha	P	U	406	Cut and fill.
Santander	St. Joe Minerals	P	U	300	Sublevel open stope.
Yauricocha	CENTROMIN	P	U	497	Combined methods.
Portugal: Aljustrel	Empresa Minera D'Aljustrel	P	U	1,200	Cut and fill.
South Africa, Republic of:					
Rep. of Gamsberg	Gamsberg Zinc Corp	E	U	3,000	Sublevel open stope.
Pering	Shell Oil Co. of South Africa	E	S	1,200	Open pit.
Spain:					
Aznalcollar	Soc. Andalusa de Piritas S A.	P	S	4,000	Open pit.
Cartagena	Penarroya	P	S	1,907	Do.
Reocin	Asturiana De Zinc S A	P	U	400	Room and pillar.
Rubiales	Exminesa	P	U	885	Cut and fill.
Sotiel	Minas de Almagresa S A	D	U	600	Room and pillar.
Sweden:					
Garpenberg	Boliden	P	U	475	Cut and fill.
Zinkgruven	Soc. des Mines et Fonderies	P	U	599	Do.
Turkey: Aladag	Cinko-Kursan Metal Sanayii	P	S	90	Open pit.
United States:					
Alaska:					
Arctic Camp	Kennecott Copper Corp	E	S	2,640	Do.
Greens Creek	Noranda-Others	E	U	240	Cut and fill.
Lik	Houston Oil & Minerals-GCO	E	S	900	Open pit.
Red Dog	Cominco	E	S	900	Do.
Colorado:					
Black Cloud	ASARCO-Resurrection	P	U	189	Room and pillar.
Idarado	Newmont Mining	PP	U	381	Sublevel open stope.
Sunnyside	Standard Metals	P	U	272	Shrinkage.
Idaho: Bunker Hill	Bunker Hill-Gulf Resources	PP	U	525	Combined methods.
Illinois: Minerva No. 1-Spivey	Inverness Mining	P	U	181	Room and pillar.
Kentucky:					
Burkesville Project	Cominco-ASARCO-Others	E	U	625	Do.
Fountain Run	St. Joe Minerals Corp	E	U	625	Combined methods.
Maine:					
Bald Mountain	Superior Oil Co	E	S	1,656	Open pit.
Kerr American-Blue Hill	Kerr American-Black Hawk	P	U	192	Room and pillar.
Nevada:					
Ruby Hill Mine	Ruby Hill-Hecla-Others	E	U	324	Combined methods.
Ward Mountain	Gulf Oil-Silver King Mines	E	U	725	Room and pillar.
New Jersey: Sterling	New Jersey Zinc Co.	P	U	182	Cut and fill.
New Mexico: Pinos Altos	Boliden-Exxon Minerals	E	U	643	Open stope.
New York:					
Balmat	St. Joe Zinc	P	U	960	Room and pillar.
Pierrepoint	..do	P	U	113	Do.
Tennessee:					
Beaver Creek	..do	P	U	544	Do.
Big War Creek	..do	E	U	272	Do.
Carthage Property	St. Joe Minerals-Others	E	U	400	Combined methods.
Copperhill: Boyd North-South	Cities Services Corp	P	U	311	Sublevel caving.
Eureka-Calloway	..do	P	U	709	Do.
Coy	ASARCO	PP	U	228	Room and pillar.
Cub Creek	New Jersey Zinc-Others	E	U	625	Combined methods.
Cumberland	Jersey Miniere Zinc Co	E	U	625	Do.
Cumberland Deposit	Exxon Minerals	E	U	625	Do.
Cumberland Property	St. Joe Minerals-Others	E	U	625	Room and pillar.
East Gainesboro	Getty Oil-Tennessee Zinc Dev	E	U	625	Combined methods.
Gainesboro	New Jersey Zinc-Others	E	U	625	Room and pillar.
Gordonsville-Elmwood	Jersey Miniere Zinc Co	P	U	2,041	Do.
Hartsville	Marathon Oil-J. F. Landers	E	U	625	Combined methods.
Hartsville Area	Cominco American-NL Ind	E	U	275	Do.
Idol	New Jersey Zinc Co	P	U	272	Room and pillar.
Immel	ASARCO	PP	U	525	Do.
Lost Creek	..do	P	U	136	Shrinkage.
New Market	..do	P	U	568	Combined methods.
Pall Mall	ASARCO-Others	E	U	400	Do.
Right Fork	ASARCO	E	U	625	Room and pillar.
Roaring River	New Jersey Zinc Co.-AMAX Inc.	E	U	625	Do.
Stonewall	Jersey Miniere Zinc Co	E	U	400	Do.
Young	ASARCO	PP	U	794	Do.
Zinc	U.S. Steel Corp	P	U	544	Do.

See footnotes at end of table.

Table 4.—MEC primary zinc properties included in this study—Continued

Country and deposit name	Owner	Current status[1]	Type[2]	Est ore capacity, 10^3 mt/yr	Mining methods
United States—Con.					
Washington:					
Boundary Dam-Metaline Falls	Metaline-Washington Res	PP	U	544	Do.
Washington Zinc Unit	Callahan Mining-Others	PP	U	635	Sublevel open stope.
Wisconsin:					
Crandon	Exxon Minerals Corp	E	U	2,820	Do.
Crawhall-Elmo No. 3	Inspiration Mines	PP	U	114	Room and pillar.
Pelican River	Noranda Corp	E	U	355	Cut and fill.
Shullsburg-Bearhole	Inspiration Mines	PP	U	340	Room and pillar.
Zaire: Kipushi	Gecamines	P	U	1,450	Combined methods.
Zambia: Broken Hill	Nchanga Consolidated	P	U	240	Sublevel open stope.

[1]D, developing deposit; E, explored deposit; P, producing mine; PP, past producer; T, temporarily shut down.
[2]C, combined surface and underground; S, surface; U, underground.

Table 5.—MEC primary copper properties (with lead and zinc as byproducts) included in this study

Country and deposit name	Owner	Current status[1]	Type[2]	Est ore capacity, 10^3 mt/yr	Mining methods
Australia:					
Benambra	Western Mining-British Petrol	E	S	520	Open pit.
C.S.A.	Conzinc Riotinto of Australia	P	U	875	Open stope.
Golden Grove	North Broken Hill Holdings	E	S	1,000	Open pit.
Canada:					
Geco	Noranda Mines Ltd.	P	U	1,588	Do.
High Lake	Kennarctic Explorations	E	U	312	Shrinkage.
Kidd Creek	Texasgulf, Inc.	P	U	4,393	Open stope.
Ruttan	Sherritt Gordon Mines Ltd.	P	U	3,175	Do.
Izok Lake	Texasgulf, Inc.	E	S	733	Open pit.
Finland: Keretti	Outokumpo Oy	P	U	500	Cut and fill.
Japan: Hanaoka	Dowa Mining Co. Ltd.	P	U	876	Do.
Namibia: Kombat-Asis West	Tsumeb Corp. Ltd.	P	U	350	Combined methods.
Norway: Tverrejellet	Folldal Verk A/S	P	U	650	Sublevel open stope.
Peru: Antamina	Minero Peru	E	S	10,500	Open pit.
Sweden: Stekenjokk	Boliden Metall AB	P	C	600	Combined methods.
Turkey:					
Cayeli	Etibank	D	U	600	Cut and fill.
Siirt	..do	E	U	705	Shrinkage.

[1]D, developing deposit; E, explored deposit; P, producing mine.
[2]C, combined surface and underground; S, surface; U, underground.

Table 6.—MEC primary silver properties (with lead and zinc as byproducts) included in this study

Country and deposit	Ownership	Current status[1]	Type[2]
Canada:			
Beaverdell	Teck Corp	P	U
Detour Project	Selco Mining Ltd. Corp.	P	S
United Keno Hill	United Veno Hill-Falconbridge.	N	C
France: L'Argentiere	Penarroya	N	U
Mexico:			
Angangueo	Impulsora Minera de Angangueo.	P	U
Avino	Cia Minera Mexicana de Avino.	P	C
Huatla	Rosario Mexico S.A. de C.U.	P	U
La Colorada	Minera Victoria Eugenia.	P	U
Lampazos	Minera Campazos S.A. de C.U.	P	U
Real del Monte y Pal	Cia Real del Monte y Pachuca.	P	U
Tocayos	Minera Victoria Eugenia.	P	U
Peru:			
Alpamarca	Sindicato Minera Rio Pallanca.	P	S
Arcata Deposit	Minas de Arcata S.A.	P	U
Julcani Mine	Cia de Minas Buejaventura S.A.	P	C
Quiruvilca	Corporacion Minera Nor-Peru.	P	U
San Genaro	Cia Minera Castrovirneuva S.A.	P	U
Sayapullo	Cia Minera Sayapullo S.A.	P	U
Uchucchacua	Cia DeMinas Buejaventura S.A.	P	U
United States:			
Colorado: Revenue Virginius	Revenue Virginius Inc. Mines.	N	U
Idaho: Clayton Silver mine	Clayton Silver Mines	P	U
Maine: Big Hill	Scintilore Explorations	N	S
Montana: Butte District Zinc	Anaconda Copper Co.	N	U

[1]P, producing; N, nonproducing.
[2]C, combined surface and underground; S, surface; U, underground.

U.S. AND WORLD HISTORICAL PRODUCTION

LEAD

Lead production in 1985 was estimated to be approximately 3.4 million mt from 48 countries. Almost 75 pct was from MEC's. Australia, the United States, Canada, Mexico, and Peru accounted for nearly 47 pct of world production, while Morocco, the Republic of South Africa, and Yugoslavia produced another 10 pct. The U.S.S.R., China, and Bulgaria, provided about 21 pct of world production, and 80 pct of centrally planned economy country (CPEC) production.

World lead production in 5-yr intervals for the 1950-85 period is shown figure 3. Since 1950, world lead production has increased twofold, with the United States, Canada, Peru, Mexico, and Australia slightly increasing their production levels while other MEC's and the CPEC's have increased their production levels substantially.

ZINC

Zinc production in 1985 was estimated to be 6.6 million mt from 51 countries. Slightly over 77 pct was from MEC's. Canada, Australia, Peru, Mexico, and the United States accounted for about 46 pct of world production, while Japan, Spain, Ireland, Sweden, and the Republic of South Africa produced another 15 pct. The U.S.S.R., China, Poland, and North Korea produced about 20 pct of world production, and 85 pct of CPEC production.

World zinc production for the 1950-85 period is shown in figure 4. Since 1950, world zinc production has increased threefold, with the United States production declining in the last 20 yr and other countries, especially those in the CPEC's and "other" MEC groups, increasing production levels substantially.

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

Nearly 80 pct of the lead and zinc properties producing in MEC's extract ore by underground mining methods. The remaining 20 pct is recovered by surface mining methods. Tables 3 and 4 show specific deposit data for primary lead and zinc deposits, respectively, such as status, capacities, and mining methods, while tables 5 and 6 show the same information for primary copper and silver properties that contain lead and zinc.

The major underground methods used in the lead-zinc industry are room-and-pillar, shrinkage, and cut-and-fill, as well as combinations of these methods. The actual method chosen depends on the depth and attitude of the ore body as well as the characteristics of the mineralized zone and the surrounding rock. Except for some open pit mines in Australia, Brazil, and Canada, open pit and block caving methods are not extensively employed. Four stages of underground mining are generally utilized: (1) Drilling with compressed air percussion drills to set up the charge pattern, (2) blasting, (3) loading with diesel- or electric-powered shovels or loaders, and (4) hauling, which utilizes trucks or load-haul-dump units (*3*).

PROCESSING

Beneficiation

Over 90 pct of all producing properties in this study primarily utilize conventional crushing, grinding, and flotation. This combines gyratory and jaw crushers with grizzly

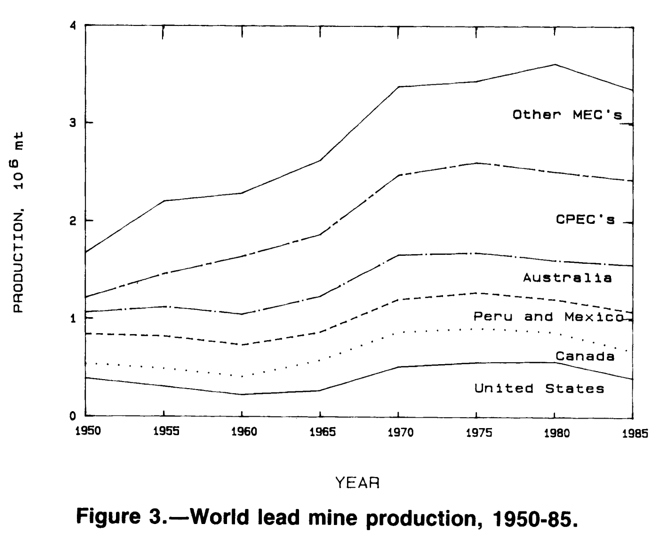

Figure 3.—World lead mine production, 1950-85.

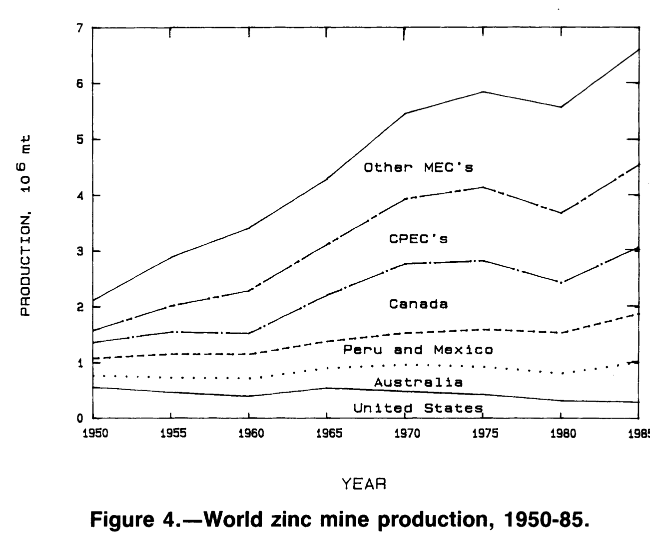

Figure 4.—World zinc mine production, 1950-85.

bars, screens, and conveyors, to prepare lead and/or zinc ore to a proper size for the rod and ball mills that further reduce the size of the ore particles. This pulverized ore is then fed into flotation cells where generally four stages of ore preparation are utilized (*3*): (1) The flotation of lead-copper materials and the depression of zinc and iron minerals, (2) the flotation separation of the lead-copper concentrate into lead and copper concentrate, (3) the activation and flotation of sphalerite from iron and gangue minerals, and (4) the flotation of the pyrite if recovery is desired. As many as three concentrates (lead, zinc, and copper) are produced in the flotation circuit.

Smelting and Refining

In about 90 pct of the deposits evaluated, conventional blast furnaces are used to smelt the lead concentrate. To prepare the concentrate for the final product four steps are generally used (*2*): (1) Sintering, which agglomerates the mixture of flotation concentrates, dust, flux, and return sinter flues and also removes sulfur, which is usually recovered by the acid plant; (2) smelting, in which the sintered product is sent to the blast furnace to be smelted, and where impure lead bullion, slag, and fume are produced; (3) drossing, in which the impure lead bullion is reduced in drossing kettles and most copper is removed (the blast furnace slag is crushed and treated in a slag fuming plant to recover zinc and lead oxide fume); and (4) pyrometallurgical refining in which any remaining copper and silver present are removed by adding sulfur and zinc dust (respectively). Also bismuth, arsenic, antimony, calcium, and magnesium are removed to produce a 99.95- to 99.99-pct-pure lead. The Imperial smelting furnace is utilized in some applications to treat complex lead-zinc ores and electrothermic refining is used in about one-fifth of present-day lead refineries.

Reduction of zinc concentrates to metal is accomplished either by distillation (retorting in a furnace) or by electrowinning the zinc from a zinc sulfate solution. Electrolytic refining accounts for approximately 80 pct of total zinc refining. Four processing stages are generally used in electrolytic zinc refining: (1) Calcining to eliminate sulfur and form leachable zinc oxide, (2) leaching with H_2SO_4 to form a $ZnSO_4$ solution, (3) purification of the solution to remove deleterious elements, e.g., copper and cadmium, and (4) electrowinning the zinc to produce a zinc product as high as 99.995 pct pure.

NEW TECHNOLOGY

Lead

Two important pyrometallurgical smelting technologies could prove beneficial for the lead industry in the future (*2*). The Queneau-Schumann-Lurgi (QSL) process combines direct reduction of concentrate metal in a single furnace with autogenous smelting. This process reportedly reduces fossil fuel costs by more than one-third and also reduces costs for removal of SO_2 by sulfur fixation as sulfuric acid. About 30,000 mt/yr of lead bullion has been produced by Metallgesellschaft AG's Metallhuttenwerk Berzelius lead smelter in Duisberg, Federal Republic of Germany. Lurgi signed a contract in 1985 with China to build a commercial smelter using QSL technology, and there is also a probability that a QSL commercial unit will be built at the Trail, B.C., smelter complex. A 50,000-mt/yr smelter would be located at Beiying.

The KIVCET-CS furnace, which utilizes a flash technique that combines the traditional processes of sintering, blast furnacing, and slag fuming, has been developed in the Soviet Union. A plant in Bolivia utilizing this technology was scheduled to commence production late in 1984; but due to financial problems and shortages of feedstock, the plant has not come up to full operating capacity.

Zinc

Several processes or systems are being developed that will further enhance zinc mine production in the future (*3*). Computer technology is playing a larger role in coordination of mining, milling, storage, and transportation for both lead and zinc. Also special techniques dealing with permafrost are being developed and utilized, e.g., zinc mines in polar regions, including the Polaris Mine in northwest Canada and the Red Dog deposit in Alaska.

Increased efficiency in material handling, electrowinning, smelting, roasting, automatic cathode handling and stripping, and improvement of sulfuric acid generation and treatment of zinc plant residues have resulted in increased capacity.

In Canada, Kidd Creek and Cominco put in their first commercial-scale pressure leaching units at their zinc smelters. This process replaces standard roasting-leaching and eliminates the sulfuric acid plant. The KIVCET-CS process, described previously, offers the possibility of recovering either zinc or zinc oxide, along with lead, from bulk lead-zinc concentrates.

PRODUCTION COSTS

LEAD

The costs presented in this section are in constant January 1985 U.S. dollars, with average costs determined over the producing life of each operation, based on current technology. Technological or other productivity improvements would affect the future costs of these operations, particularly those with long lives.

Table 7 and figure 5 compare weighted-average production costs per pound of lead for producing and nonproducing operations in the United States and other MEC's. Costs for producing mines in the United States are comparable to foreign costs except that mining costs and byproduct credit are lower. Mine operating costs average $0.09/lb lead in the United States compared with $0.12 for foreign countries because of the low-cost, highly productive room-and-pillar mining methods used in the Missouri lead mines. However, foreign mines receive the benefit of byproduct credits that average nearly twice those of domestic mines.

Table 7.—Lead production costs for producing and nonproducing operations in the United States and other MEC's

(All costs are in January 1985 U.S. dollars per pound of recoverable lead on a weighted-average basis)

	Operating cost		Smelting-refining	Transpor-tation	Less byproduct credit[1]	Net operating cost	Recovery of capital[2]	0-pct DCFROR		15-pct DCFROR		
	Mine	Mill						Taxes and royalties[3]	Total cost[4]	Taxes and royalties[5]	Return on investment[6]	Total cost[7]
PRODUCERS												
Other MEC's...	0.12	0.05	0.14	0.04	0.19	0.16	0.02	0.02	0.20	0.03	0.02	0.23
United States ..	.09	.05	.15	.03	.10	.22	.09	.01	.32	.02	.02	.35
Total or av...	.10	.05	.15	.03	.12	.21	.06	.02	.29	.03	.02	.32
NONPRODUCERS												
Other MEC's...	0.30	0.11	0.16	0.09	0.28	0.38	0.12	0.01	0.51	0.26	0.35	1.11
United States ..	.09	.06	.15	.04	.07	.27	.13	.03	.43	.05	.04	.48
Total or av...	.17	.08	.16	.06	.15	.32	.12	.02	.46	.13	.16	.72

[1]Includes all byproduct revenue credits for the operation.
[2]Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure, and reinvestments required over the life of the operation.
[3]Includes property, State, Federal, and severance taxes, and royalties, where applicable, calculated at a 0-pct DCFROR.
[4]Equal to the sum of net operating costs, taxation, and capital recovery determined at a 0-pct DCFROR.
[5]Includes property, State, Federal, and severance taxes, and royalties, where applicable, calculated at a 15-pct DCFROR.
[6]The revenue increase per pound necessary to obtain a 15-pct DCFROR.
[7]Equal to the sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery, *plus* the return on investment.

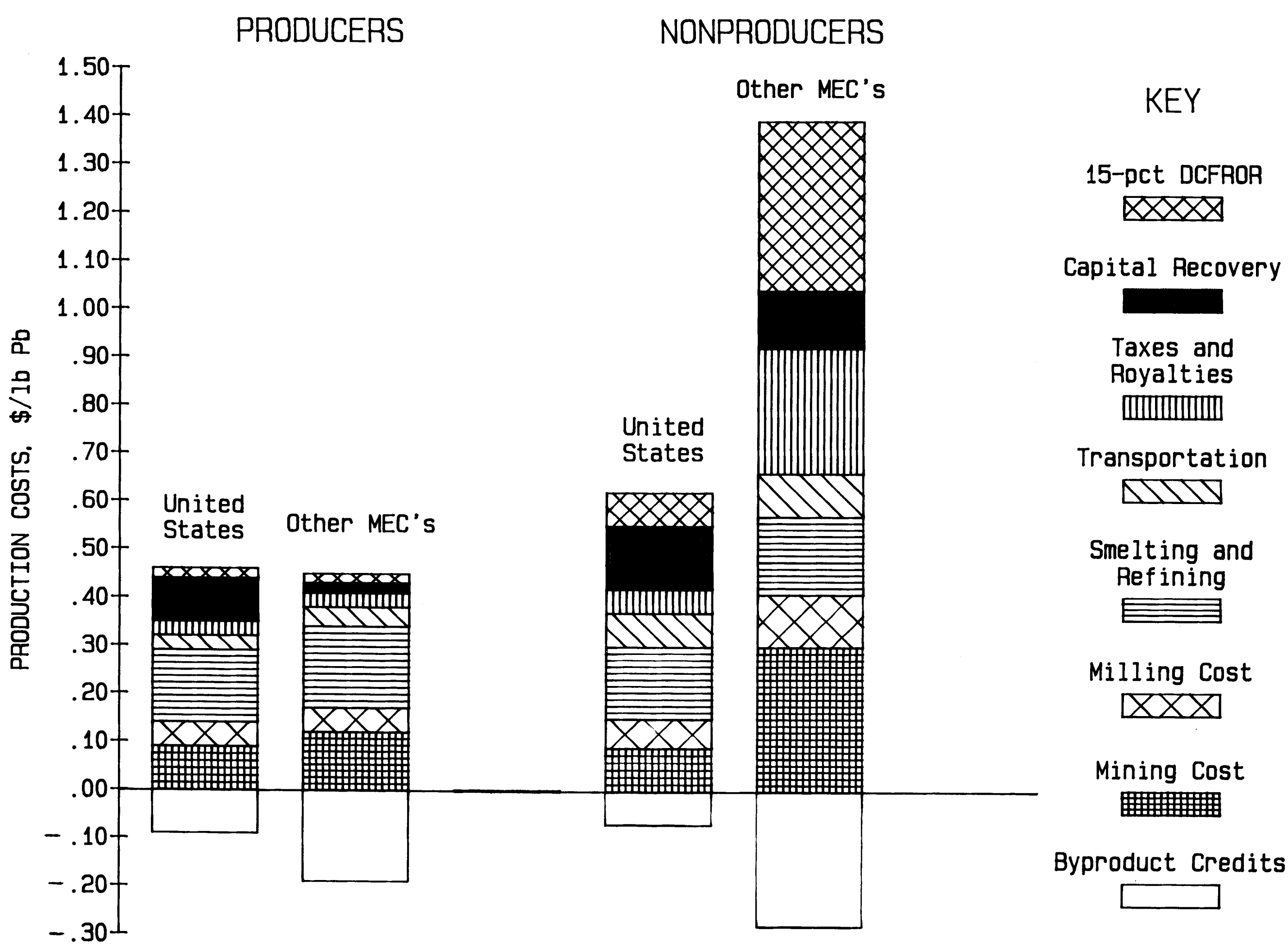

Figure 5.—Lead production costs for selected United States and other MEC operations (January 1985 U.S. dollars).

As a result, the weighted-average net operating cost of producing foreign mines is $0.06/lb less ($0.16 compared with $0.22) than that for U.S. mines. On a total cost basis, the U.S. mines have a higher charge for recovery of capital than foreign mines ($0.09 versus $0.02) because they tend to be newer and more capital intensive than the foreign mines. As a result, total production costs at a 15-pct discounted-cash-flow rate of return (DCFROR) are estimated to be $0.35/lb for U.S. producers, while foreign producers average $0.12 less, at $0.23/lb.

ZINC

Table 8 and figure 6 illustrate estimated total production costs per pound of zinc for producing and nonproducing zinc operations. Weighted-average mine operating costs for producing mines range from a low of $0.12/lb zinc in Peru and Spain to a high of $0.28/lb in Japan. The relatively high cost of underground mining in Japan and Mexico is due to the complex nature of the deposits and small capacity of many of the mines. Although the mining cost per metric ton of ore is lower in the United States than in foreign countries, the lower zinc grades of domestic deposits result in a relatively high cost on a per-pound-zinc basis.

Mill operating costs for producing mines range from a high of $0.19/lb zinc in Spain to a low of $0.04/lb in Australia. Mill operating costs vary depending upon the zinc grade and the complexity of the ore. A complex ore requiring the flotation of several different concentrates will incur a greater mill operating cost than a simple ore.

For producing mines, the average cost to refine zinc (including the cost of smelting and refining byproducts) ranges from $0.18/lb to $0.34/lb of zinc. Actual refining costs of zinc do not vary appreciably among countries, but other costs, including smelting and refining of lead and other byproduct commodities, can vary considerably. This wide range is due mainly to differences in the number of byproducts and byproduct grades, which result in additional smelting and refining charges. The operations with high costs in this category usually recover much or all of the added cost in the form of byproduct credits.

Longrun weighted-average total costs for all producing operations including taxes and recovery of capital (0-pct DCFROR) range from $0.32/lb to $0.62/lb of zinc. The United States and Canada are the two highest total cost producers among the MEC's, while Australia and Mexico are the lowest. Generally, countries that produce complex ore with high zinc grades tend to be the low-cost producers and countries such as the United States and Spain, which produce simple ores with few byproducts and relatively low zinc grades, tend to be the highest.

Table 8.—Zinc production costs for producing and nonproducing operations in selected MEC's

(All costs are in January 1985 U.S. dollars per pound of recoverable zinc on a weighted-average basis)

	Operating cost		Smelting-refining	Transportation	Less byproduct credit[1]	Net operating cost	Recovery of capital[2]	0-pct DCFROR		15-pct DCFROR		
	Mine	Mill						Taxes and royalties[3]	Total cost[4]	Taxes and royalties[5]	Return on investment[6]	Total cost[7]
PRODUCERS												
Australia	0.15	0.04	0.32	0.03	0.30	0.24	0.05	0.03	0.32	0.05	0.02	0.36
Canada	.20	.12	.34	.07	.20	.53	.09	.00	.62	.03	.04	.69
Italy	.18	.07	.21	.01	.12	.35	.09	.00	.45	.02	.02	.48
Japan	.28	.10	.19	.01	.10	.48	.07	.00	.55	.02	.02	.59
Mexico	.26	.08	.25	.03	.39	.23	.06	.05	.34	.07	.03	.39
Peru	.12	.09	.20	.07	.20	.38	.10	.04	.52	.06	.04	.59
Spain	.12	.19	.21	.02	.18	.36	.08	.00	.45	.01	.04	.49
United States	.22	.11	.28	.02	.13	.50	.10	.02	.62	.03	.03	.67
Other	.17	.08	.18	.02	.18	.27	.04	.02	.33	.04	.02	.37
Total or av.	.19	.09	.26	.04	.22	.36	.08	.02	.46	.04	.03	.51
NONPRODUCERS												
Australia	0.12	0.04	0.27	0.06	0.15	0.34	0.09	0.03	0.46	0.16	0.16	0.75
Canada	.26	.09	.27	.10	.14	.58	.16	.04	.78	.17	.18	1.10
India	.18	.06	.25	.01	.10	.40	.08	.00	.49	.08	.08	.64
Ireland	.19	.08	.18	.02	.04	.43	.12	.02	.57	.16	.18	.89
United States	.13	.08	.21	.10	.07	.45	.35	.03	.83	.15	.38	1.33
Other	.12	.07	.14	.04	.10	.27	.07	.01	.35	.07	.01	.52
Total or av.	.15	.07	.23	.07	.10	.42	.12	.03	.65	.14	.18	.99

[1]Includes all byproduct revenue credits for the operation.
[2]Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure, and reinvestments required over the life of the operation.
[3]Includes property, State, Federal, and severance taxes, and royalties, where applicable, calculated at a 0-pct DCFROR.
[4]Equal to the sum of net operating costs, taxation, and capital recovery determined at a 0-pct DCFROR.
[5]Includes property, State, Federal, and severance taxes, and royalties, where applicable, calculated at a 15-pct DCFROR.
[6]The revenue increase per pound necessary to obtain a 15-pct DCFROR.
[7]Equal to the sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery, *plus* the return on investment.

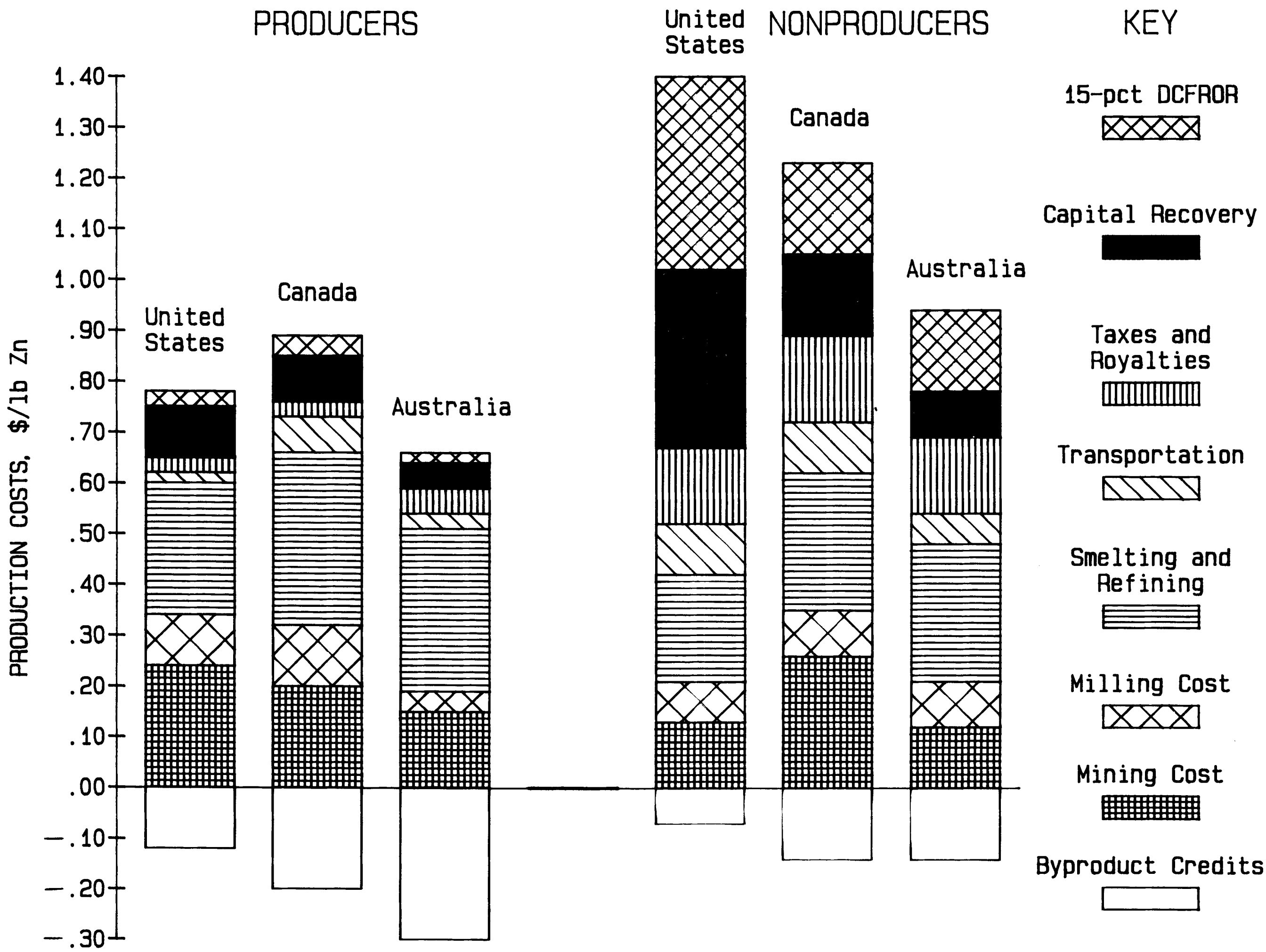

Figure 6.—Zinc production costs for selected MEC's (January 1985 U.S. dollars).

AVAILABILITY

TOTAL RECOVERABLE

The study analyzed 212 lead and zinc mines and deposits. Of this total 26 were evaluated for primary lead and 186 as primary zinc. Also included were 16 primary copper properties and 22 primary silver properties that have byproduct or coproduct lead and/or zinc. Production and resource costs in CPEC's were not estimated owing to a lack of data.

Lead

Of the 26 primary MEC lead mines and deposits evaluated, 18 were domestic and 8 were foreign. In 1985, 18 of the mines were in production, 7 in the United States and 11 in foreign countries. A total of eight nonproducing deposits were also evaluated, four in the United States and four in foreign countries, including deposits that are developing or have developmental plans, as well as explored prospects.

The tonnage of recoverable lead potentially available from the producing mines and nonproducing deposits analyzed in MEC's for primary and byproduct lead is shown in figure 7. Costs for operating mines are bracketed by lower and upper cost levels. The lower level reflects costs at a 0-pct DCFROR and the upper level reflects total cost including a 15-pct DCFROR.

A total of 22.1 million mt of recoverable lead is potentially available from the demonstrated resources of the MEC deposits evaluated as primary lead properties. Of the total, 74 pct (16.4 million mt) occurs in mines that were producing in 1985. At a cost of $0.19/lb (the U.S. lead price as of January 1985), including a 0-pct DCFROR, total availability from producing mines would be only 3.0 million mt. At a price of $0.45/lb (including a 15-pct DCFROR) 19.0 million mt of lead is potentially available, 62 pct of which would be from domestic properties.

Approximately 39 million mt of byproduct lead is potentially recoverable from 144 of the 186 mines and deposits evaluated as primary zinc deposits. Below the 1985 U.S.

zinc price of $0.41/lb (U.S. high grade), approximately 9.24 million mt of byproduct lead is potentially available. An additional 382,000 mt is potentially available from 8 of the 16 MEC properties evaluated as primary copper operations and 535,000 mt from 20 of the 22 MEC properties evaluated as primary silver operations.

Zinc

Of the 186 primary MEC zinc mines and deposits evaluated, 50 were domestic and 136 were foreign. As of 1985, 111 of the mines were in production, 15 in the United States and 98 in foreign countries. A total of 73 nonproducing deposits were also evaluated (35 in the United States and 38 in foreign countries); included were deposits that are developing or have developmental plans, as well as explored prospects.

The tonnage of potentially recoverable zinc from evaluated primary zinc mines is shown in figure 8. Production costs for producing mines are bracketed by lower and upper cost levels. The lower level reflects costs at a 0-pct DCFROR and the upper level reflects total cost including a 15-pct DCFROR. A total of 128 million mt of zinc is potentially recoverable from the demonstrated resources of the primary zinc deposits. Of the total, 38 pct is from mines that were producing in 1985.

At the 0-pct DCFROR return level, a total of 24 million mt is potentially recoverable from producing mines at under $0.41/lb (the January 1985 U.S. high-grade metal price). At $0.41/lb, Mexico would account for 15 pct, Australia 14 pct, Peru 12 pct, Namibia 10 pct, Europe 28 pct, Canada 2 pct, and the United States 0.2 pct of the tonnage from producers.

From 19 of the 26 MEC properties that were evaluated as primary lead deposits, approximately 3.7 million mt (0.3 million mt from producing mines) of byproduct zinc is potentially available. An additional 11.2 million mt of byproduct zinc (4.5 million mt from producing mines) is potentially recoverable from 15 of the 16 MEC properties evaluated as primary copper properties. Another 1.2 million mt (0.1 million mt for producers) is potentially recoverable from 13 of the 22 MEC properties evaluated as primary silver properties.

ANNUAL CAPACITY

Lead

Potential 1985 lead production at full capacity from all producing MEC mines, and the U.S share of that production, is shown in figure 9. Curves are presented for primary and byproduct or coproduct operations. Each graph is bracketed by a lower operating cost level and an upper cost level that includes a 0-pct DCFROR. In 1985, about 2.2 million mt of lead was potentially available from the evaluated mines, 0.711 million mt from primary lead and about 1.4 million mt as a byproduct of primary zinc mines. This compares with an estimated 1985 actual lead production of 2.5 million mt from MEC's (*8*). U.S. mines have the

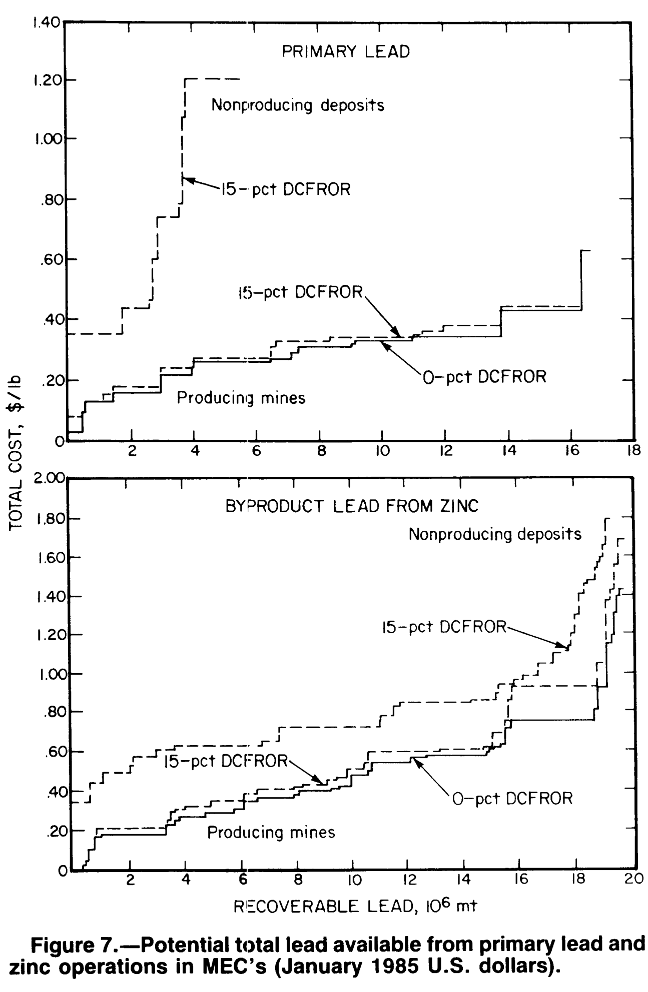

Figure 7.—Potential total lead available from primary lead and zinc operations in MEC's (January 1985 U.S. dollars).

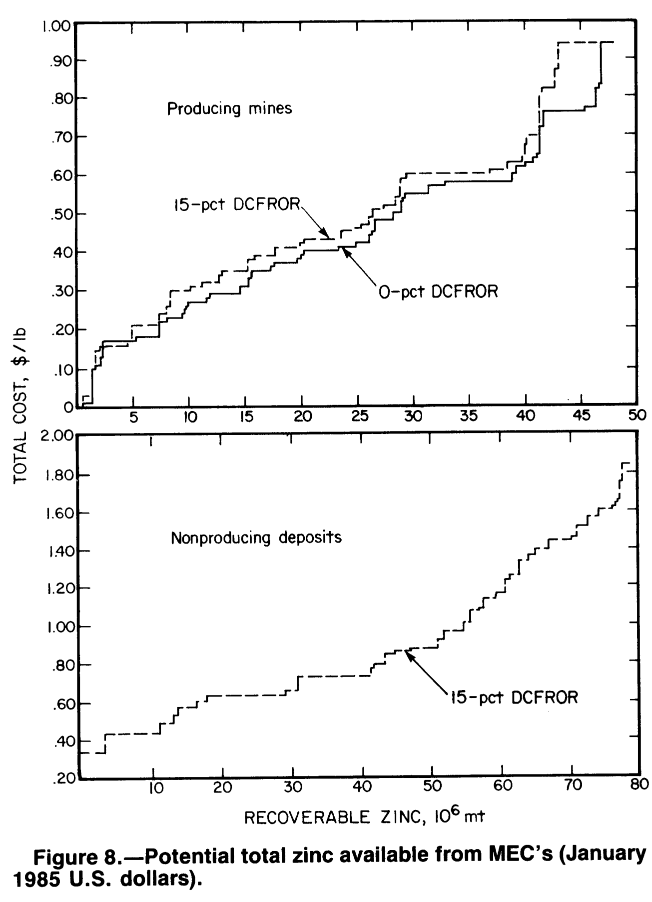

Figure 8.—Potential total zinc available from MEC's (January 1985 U.S. dollars).

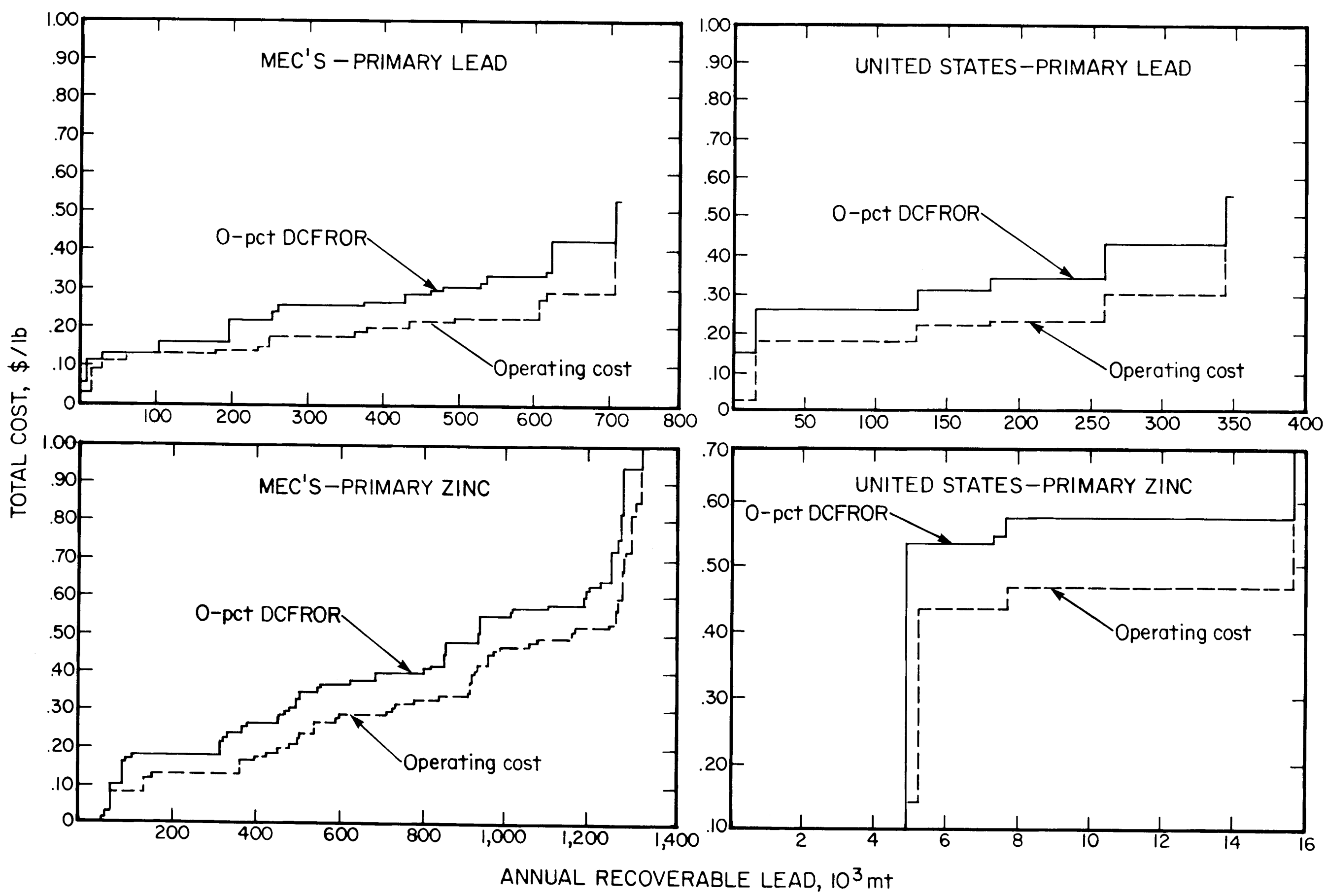

Figure 9.—1985 lead capacities from producing primary lead and zinc operations in MEC's and the United States (January 1985 U.S. dollars).

capacity to produce an estimated 347,000 mt from primary lead properties and 15,700 mt from primary zinc properties, compared with a 1985 actual production estimate of 400,000 mt. Capacities for Australia were estimated to be 54,000 mt from primary lead mines and 403,000 from primary zinc mines, Canada—227,000 mt from primary zinc mines, Peru—170,000 mt from primary zinc mines, Mexico—15,000 from primary lead mines and 156,000 from primary zinc mines, and the Republic of South Africa—92,000 mt from primary lead mines. Differences in capacities and estimated 1985 actual production figures are most likely due to additional tonnages available from lead produced as a byproduct or coproduct of copper and silver properties as well as other smaller mines not included in this study.

At an average total cost (including a 0-pct DCFROR) equal to the January 1985 price for lead in the United States of approximately $0.19/lb, an estimated 16,000 mt could potentially be mined from primary U.S. lead mines. At a cost equal to the January 1985 U.S. high-grade zinc price of $0.41/lb, an additional 5,500 mt is potentially available from mines evaluated as primary U.S. zinc mines. As shown by the curves, many mines have continued to operate even though they are not able to cover their longrun production costs at current market prices.

Potential production of lead from 1985 to 1995 from the demonstrated resources of producing primary lead and zinc mines in all MEC's and in the United States is shown in figure 10. The curves reflect the production capacity of existing mines, including planned expansions. It is assumed that all operations produce at full (100 pct) capacity over the life of the mine. If actual production is at less than capacity, the curves will not actually decline as rapidly as shown. Because it was assumed that over the long run all costs of production, including a 15-pct DCFROR must be recovered, potential annual capacities are shown for selected cost ranges that include a 15-pct DCFROR. The MEC curve for primary lead properties shows a prominent increase from 1987 to 1988, which is due to planned expansions coming on stream. A gradual decline begins in 1989, due to the potential depletion of several foreign mines. As shown, domestic primary lead mines have sufficient resources to maintain production at capacity levels through at least 1995.

The curve for lead from primary zinc mines shows a much more rapid decrease as demonstrated resources from many mines are depleted. However, it is doubtful that the demonstrated resources of these producing mines will decline at the rate indicated by the curve. Many of the zinc and lead mine operators report their demonstrated resources for only a few years ahead of their current mining position and increase or maintain their reserves as mining continues. Also, in many cases, demonstrated resources are increased each year as a result of ongoing exploration programs. Therefore, there is a high probability that additional resources exist that will allow most of the currently producing mines to continue beyond the time frame of reported reserves.

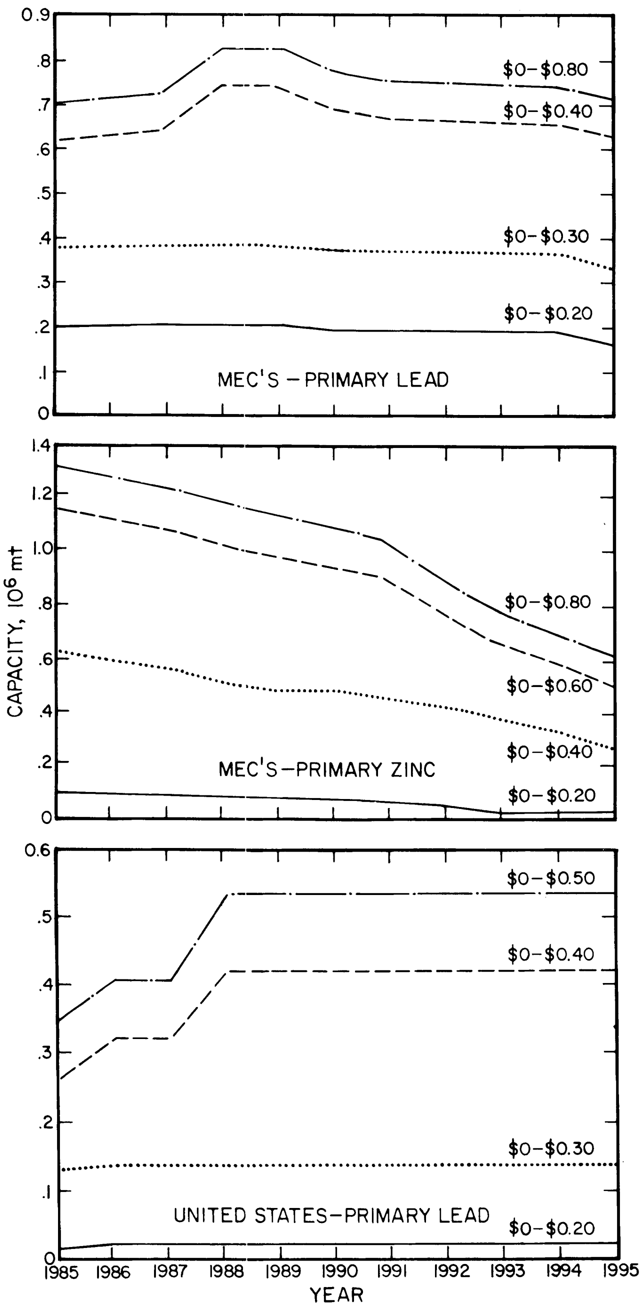

Figure 10.—Potential annual lead production from producing primary lead and zinc mines in MEC's and the United States at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

Zinc

Potential 1985 production, at full capacity, from producing primary MEC mines and from the United States, Canada, and Australia, is shown in figure 11. A total of 3.7 million mt of zinc was potentially available from all producing MEC primary zinc mines. The U.S. mines studied had the capacity to produce an estimated 278,000 mt, compared with an estimated 1985 production of 245,000 mt; Canada had 781,000 mt of capacity compared with estimated production of 1.2 million mt; Australia had an estimated capacity of 500,000 mt compared with estimated production 720,000 mt; and Mexico had 312,000 mt of capacity compared with 310,000 mt of production. In addition Peru (not shown on figure 11) had 430,000 mt of capacity compared with estimated production of 560,000 mt (*5*). Differences in capacity and production are due to byproduct and coproduct production from lead, silver, and copper properties, and production from smaller mines that were not included in the study.

Each graph is bracketed by a lower operating cost level and an upper cost level that includes a 0-pct rate of return. At an average total cost (including a 0-pct DCFROR) roughly equal to the January 1985 price for zinc in the United States of approximately $0.41/lb, only about half of the operations were able to cover production costs. At this price, 2.1 million mt was potentially available from all MEC mines, while 1985 production was estimated to be 5.1 million mt. This indicates that many mines continued to produce even though they were not able to cover their full cost of production.

Potential annual production of zinc from the demonstrated resources of producing mines in MEC's from 1985 to 1995 is shown in figure 12. The curves reflect the production capacity of existing mines, including planned expansions when known. It was assumed that all operations produce at full (100 pct) capacity over the life of the mine. If actual production is at less than capacity levels, the curves will not actually decline as rapidly as shown. Since it was assumed that over the long run all costs of production, including a 15-pct DCFROR must be recovered, potential annual capacities are shown for selected cost ranges that include a 15-pct DCFROR. Costs for producing mines average $0.51/lb for all MEC's, $0.67/lb for the United States, $0.69/lb for Canada, and $0.36/lb for Australia (see table 9).

The curves indicate a more rapid depletion rate for zinc than for lead. However, as stated earlier, because many companies report reserves for only a few years, the actual decline will likely be much less rapid than indicated by the curves.

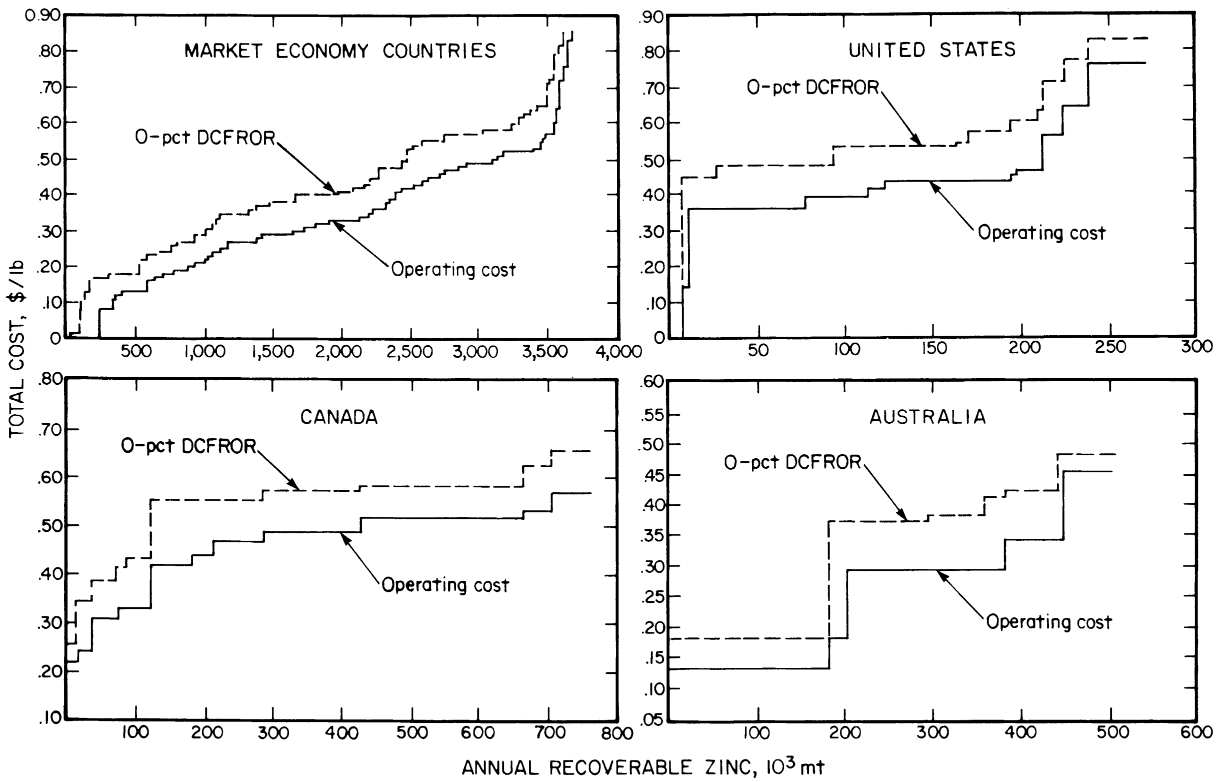

Figure 11.—1985 zinc capacities from producing mines in selected MEC's (January 1985 U.S. dollars).

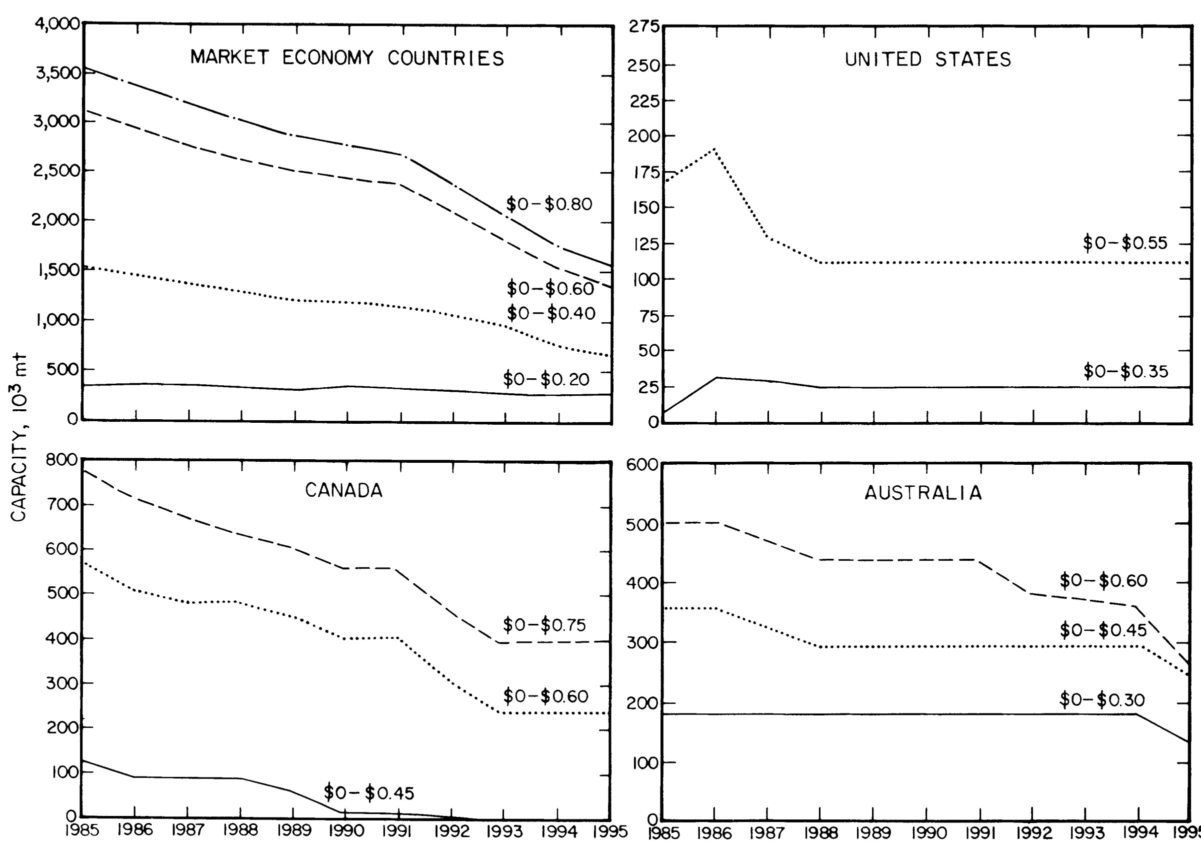

Figure 12.—Potential annual zinc production from producing mines in various MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

CONCLUSIONS

Approximately 61 million mt of lead is estimated to be recoverable from the 212 MEC primary lead and zinc mines and deposits included in this study. The 26 producing mines evaluated as primary lead mines have a weighted-average total production cost (including a 15-pct DCFROR) of $0.32/lb. Total potential lead production from these mines amounts to 22.1 million mt with 3.0 million mt available at or below a longrun total cost of $0.19/lb. The seven U.S. producing lead mines have a weighted-average estimated total cost of $0.35/lb while foreign mines average $0.23/lb. Nearly 5.7 million mt of lead is potentially recoverable from nonproducing lead properties; at a 15-pct DCFROR these deposits have a weighted-average total cost of $0.72/lb.

A total of 132 million mt of zinc is potentially recoverable from the properties evaluated in this study. The 111 producing mines evaluated as primary zinc mines have a weighted-average total production cost (including a 15-pct DCFROR) of $0.51/lb. Total potential zinc production from these mines amounts to 48.6 million mt with 24 million mt potentially available at estimated cost levels below the January 1985 high-grade-metal market price of $0.41/lb. The weighted-average estimated total production cost (including a 15-pct DCFROR) for U.S. producing mines is estimated to be $0.67 for the 3.2 million mt of recoverable zinc. Nearly 80 million mt is potentially recoverable from nonproducing zinc deposits which have a weighted-average total cost of $0.99/lb (including a 15-pct DCFROR).

Low prices and high total production costs for both lead and zinc have painted a bleak picture for the industry in the United States. Also the financial position of producers has become more precarious over the past few years. Several marginal producers have shut down permanently and other high-cost producers may shut down if low prices continue.

REFERENCES

1. Peterson, G. R., K. E. Porter, and A. A. Soja. Primary Lead and Zinc Availability—Market Economy Countries. A Minerals Availability Program Appraisal. BuMines IC 9026, 1985, 44 pp.
2. Woodbury, W. D. Lead. Ch. in Mineral Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 433-452.
3. Jolly, J. H. Zinc. Ch. in Mineral Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 923-940.
4. Kilgore, C. K., S. J. Arbelbide, and A. A. Soja. Lead and Zinc Availability—Domestic. A Minerals Availability Program Appraisal. BuMines IC 8962, 1983, 30 pp.
5. Jolly, J. H. Zinc. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 178-179.
6. ______. Zinc. Ch. in BuMines Minerals Yearbook 1985, v. 1, pp. 1033-1060.
7. U.S. Bureau of Mines. Lead and Zinc. Quarterly and Annual Mineral Industry Surveys.
8. Woodbury, W. D. Lead Sec. in BuMines Mineral Commodity Summaries 1986, pp. 88-89.

LITHIUM

INTRODUCTION

Lithium is consumed in a variety of forms including mineral concentrates (such as spodumene and petalite), lithium carbonate (Li_2CO_3), lithium monohydrates ($LiOH{\cdot}H_2O$), lithium chloride (LiCl), and lithium metal. Most consumption is as Li_2CO_3 by the aluminum industry or as feedstock for other lithium chemicals (*1*).[1]

The largest use of Li_2CO_3 is in electrolytic aluminum reduction cells, which accounts for an estimated 30 pct of U.S. consumption (on a contained lithium basis) (*2*). The addition of Li_2CO_3 to the cryolite bath lowers the cell's melting point and increases its electrical conductivity, both of which reduce the energy demands of aluminum potlines (*3*). $LiCO_3$ is also used in the steel enameling (glazing) process to enhance the enamel's resistance to thermal shock.

The ceramic and glass industry and lubricants, together account for an estimated 40 pct of the lithium consumed in the United States (*2*). Low-iron spodumene is used in ceramics and in rigid foam insulation to impart low thermal expansion properties. Petalite concentrates and $LiCO_3$ are also used for making thermal shock-resistant glass cookware. Other ceramic and glass uses include lithium in sealed beam headlights, photochromatic glass lenses, and large telescopic lenses.

Lithium monohydroxide is a component of over 50 pct of the greases used in the United States (*1*). These greases contain about 2 pct Li and are effective as lubricants over a wide range of temperatures.

Monohydroxide can also be used to make LiCl. The most important use of LiCl is as a feedstock for the production of lithium metal. The metal in turn is used for the production of butyllithium, which is used as a catalyst in the production of synthetic rubber. Research has recently developed a lightweight, high-strength aluminum-lithium alloy that could increase the lithium metal consumption in the aerospace industry.

Small quantities of lithium are also used by the medical industry where $LiCO_3$ is a primary component of drugs prescribed for the treatment of manic depression.

Lithium in pure form is a soft, silvery white metal that is the lightest of all solid elements. It is highly reactive as a pure element and does not occur in nature as a metal but instead is always combined in more stable compounds. Lithium is most commonly recovered from spodumene minerals in pegmatites and is increasingly being recovered from salt brines. Other pegmatitic minerals mined for their lithium content are lepidolite, petalite, and amblygonite (*4*). In addition to economic concentrations of lithium-bearing minerals, pegmatites may also contain economic concentrations of mica, pollucite, sandspar, tantalum, and tin.

This study includes three types of lithium commodities: $LiCO_3$ carbonate, spodumene concentrates, and petalite concentrates. $LiCO_3$ is derived by further processing of spodumene concentrates and is the principal product of brine operations.

Most of the information presented in this chapter is an updated summary of Bureau of Mines Information Circular 9102 "Lithium Availability—Market Economy Countries. A Minerals Availability Appraisal" (*5*).

GEOLOGY AND RESOURCES

GEOLOGY

Lithium is obtained from pegmatites and salt brines. Lithium-bearing pegmatites occur in Precambrian metamorphosed shield-type rocks. Brine deposits occur in closed drainage basins in areas of low precipitation and high evaporation.

Pegmatites

The occurrence of lithium-bearing pegmatites are widespread throughout the world in shield-type rocks. Generally, the geological environment for the formation of spodumene pegmatites produces swarms of pegmatites that may contain hundreds of small pegmatites. Principal pegmatitic lithium minerals include spodumene ($LiAlSi_2O_6$), petalite ($LiAlSi_4O_{10}$), lepidolite [$K_2Li_3Al_3(AlSi_3O_{10})_2(O,OF,F)_4$], amblygonite ($LiAlPO_4[F,OH]$), and eucryptite ($LiAlSiO_4$) (*5*).

Lithium-bearing pegmatites have been classified into two categories: (1) unzoned pegmatites that contain a consistent spodumene content throughout the pegmatite and (2) zoned pegmatites that contain spodumene, petalite, and lepidolite. Unzoned pegmatites are far more important quantitatively and where mined, the spodumene may compose up to 25 pct of the rock. Zoned pegmatites generally contain other economically important minerals. The largest known zoned pegmatite is the Bikita pegmatite in Zimbabwe, which contains petalite, spodumene, lepidolite, amblygonite, and pollucite.

Brines

Brines are remnant subsurface waters, rich in dissolved salts, that occur in playa lakes (salares in Latin America).

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

Playa lakes are found in closed or restricted drainage basins where the rate of evaporation is greater than the rate of precipitation. Complete evaporation leaves a flat lake bed encrusted with precipitated mineral salts. The lithium-rich brines occur within the interstices of this porous crust.

RESOURCES

Figure 1 shows identified lithium resources, the world reserve base, and the resources of the properties evaluated in this study. The reserve base includes additional brine resources for the Salar de Uyani, Bolivia, deposit (*2*).

Total demonstrated resources evaluated amount to 3.4 million mt of contained lithium, 53 pct contained in brines and 47 pct in pegmatites. Of the total demonstrated resource, 2.2 million mt is recoverable (table 1); 59 pct in Chile, 13 pct in the United States, 11 pct in Australia, and 11 pct in Canada. The remaining 6 pct is located in Bolivia, Zaire, and Zimbabwe. Inferrred resources for the deposits evaluated for this study amount to 8.4 million mt of contained lithium, of which 95 pct is located in South American brine deposits.

Table 2 lists the properties included in this study.

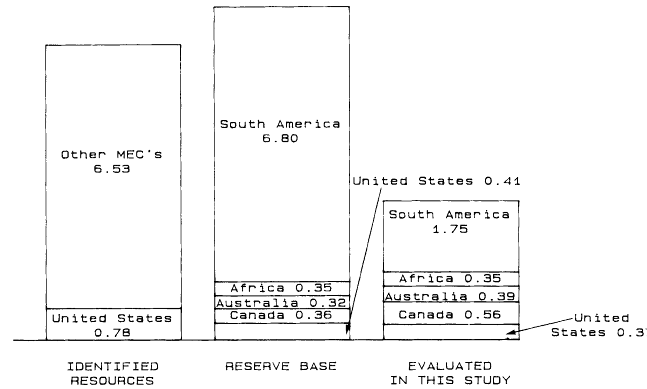

Figure 1.—Estimates of MEC lithium resources (million metric tons of lithium).

Table 1.—Summary of MEC lithium resources evaluated for this study, as of January 1985

Country and property	In situ material: Demonstrated, 10^6 mt	In situ material: Grade, wt pct Li	Resources, 10^3 mt Li: Demonstrated: Contained	Resources, 10^3 mt Li: Demonstrated: Recoverable	Resources, 10^3 mt Li: Inferred, contained
PEGMATITES					
Australia: Greenbushes	33.37	1.16	387	247	NR
Canada:					
Bernic Lake	6.65	1.28	85	50	NR
Buck, Coe, Pegli	.80	.99	8	4	NR
Georgia Lake	3.20	.59	19	11	NR
Jean Lake	1.50	.60	9	6	NR
Lac la Croix	1.40	.59	8	5	NR
Nama Creek	5.55	.48	27	16	NR
Quebec Lithium	14.50	.60	88	59	32
Yellowknife	49.11	.65	319	91	NR
Total or wtd av	82.71	.68	563	242	32
United States:					
Bessemer City	21.56	.68	147	101	NR
Kings Mountain	21.74	.68	148	115	20
Total or wtd av	43.30	.68	295	216	20
Africa:					
Zaire: Kitotolo	31.50	.98	307	12	188
Zimbabwe: Bikita	3.30	1.35	45	15	100
Total or wtd av	34.80	1.02	352	27	288
Total or wtd av	194.18	.82	1,597	732	340
BRINES					
Bolivia: Salar de Uyuni	505.00	.025	126	101	5,374
Chile: Salar de Atacama	1,300.00	.125	1,625	1,300	2,675
United States: Silver Peak	240.00	.033	72	65	52
Total or wtd av	2,045	.089	1,823	1,466	8,101
Grand total or wtd av	2,239.18	.154	3,420	2,198	8,441

NR Not reported

[1] Includes measured plus indicated quantities.

[2] Recoverable brine resources are reported as lithium contained in recoverable product.

Note.—Data may not add to totals shown because of independent rounding.

Table 2.—MEC lithium properties included in this study

Country and property	Ownership	Current status[1]	Mine method[2]	Mill method[3]	Initial year of production	Production capacity, mt/yr	Product[4]	Product grade
PEGMATITES (product grade in percent LiO_2)								
Australia: Greenbushes	Greenbushes Tin Ltd.	P	OP	F	1982	49,200	S	7.0
Canada:								
Bernic Lake	Tanco Mining Group	P	UG	F	1985	72,900	S	6.2
Buck, Coe, Pegli	Lithium Corporation of Canada	E	UG	F	NAp	30,000	S	6.0
Georgia Lake	Various owners	E	UG	F	NAp	22,200	S	5.8
Jean Lake	Unclaimed land	E	UG	F	NAp	21,700	S	5.8
Lac la Croix	..do	E	UG	F	NAp	21,100	S	5.8
Nama Creek	Cominco Ltd.	E	UG	F	NAp	34,800	S	5.8
Quebec Lithium	Sullivan Mining Group Ltd.	PP	UG	F	Unk	49,000	S	5.6
Yellowknife	Canadian Superior Exploration Ltd.	E	OP	F	NAp	45,300	S	6.0
United States:								
Bessemer City	Lithium Corp. of America (Lithco)	P	OP	F	1954	124,000	S[5]	5.1
Kings Mountain	Foote Minerals Co.	P	OP	F	1943	66,600	S[5]	5.5
Zaire: Kitotolo	Geomines and Zaire Government	E	OP	F	NAp	39,500	S	6.0
Zimbabwe: Bikita	Bikita Minerals Ltd.	P	OP	F	Unk	38,500	P	4.1
							S	7.1
							L	4.1
BRINES (product grade in percent Li_2CO_3)								
Bolivia: Salar de Uyuni	Government	E	B	E	NAp	[6]6,350	LC	99.5
Chile: Salar de Atacama	Foote Minerals Co. and CORFO	P	B	E	1984	[6]6,350	LC	99.5
United States: Silver Peak	Foote Minerals Co.	P	B	E	1966	[6]6,350	LC	99.5

NAp Not applicable. Unk Unknown
[1] P, producing; E, explored; PP, past producer.
[2] OP, open pit; UG, underground; B, brine.
[3] F, flotation; E, evaporation.
[4] S, spodumene; LC, $LiCO_3$; P, petalite; L, lepidolite.
[5] Production reported as metric tons of spodumene concentrate; however, most U.S. spodumene goes to the production of Li_2CO_3.
[6] Metric tons of $LiCO_3$

U.S. AND WORLD HISTORICAL PRODUCTION

In recent years, nearly 70 pct of world lithium production has been from U.S. deposits, which are owned and operated by the Lithium Corporation of America (Lithco), a subsidiary of FMC Corp., and by the Foote Mineral Co., a subsidiary of Newmont Mining Corp. Lithium production for the 1980-85 period is listed in table 3 (*2-3, 6-8*).

The market structure of lithium has been relatively stable in view of the longstanding supply situation of the two U.S. companies. Lithco produces numerous products at its Bessemer City, NC, plant and its subsidiary, Lithco Europe Ltd., in the United Kingdom. Foote Minerals produces mainly $LiCO_3$ at its Kings Mountain, NC, plant (a few kilometers from the Lithco operation) and its brine operations in Nevada and Chile. Foote Mineral's spodumene concentrate and $LiCO_3$ are supplied to company-owned plants in Pennsylvania, Tennessee, and Virginia to produce other downstream (value-added) products. Both companies supply raw materials to European plants for the production of lithium products for the European market. The principal producer in Europe (other than Lithco Europe) is Chemettall, a subsidiary of Metallgesellschaft, Federal Republic of Germany.

In 1984, Chile entered into the market by developing the Salar de Atacama operation. The Chilean Government possesses a 45-pct share of the operation through the Government's development company, Corporacion de Fomento de la Produccion (CORFO). The remaining 55 pct

Table 3.—World lithium production, 1980-85,[1] metric tons contained lithium

Country	1980	1981	1982	1983	1984	1985[e]
Argentina	10	NA	NA	5	NA	NA
Australia	NA	NA	5	70	210	270
Brazil	55	60	60	55	10	10
Canada	NA	NA	NA	NA	NA	135
Chile	NA	NA	NA	NA	480	900
China	270	270	280	320	450	450
Namibia	NA	35	25	20	15	15
Portugal	15	15	15	10	5	5
United States[2]	4,920	4,920	3,470	4,455	4,445	W
U.S.S.R.	1,180	1,090	1,090	1,270	1,630	1,630
Zimbabwe	400	420	290	140	160	180
Total	6,850	6,810	5,235	6,345	7,405	W
United States share of total pct	72	72	66	69	60	NA

[e]Estimated. NA Not available.
W Withheld to avoid disclosing company proprietary data.
[1] Contained lithium estimated from data on mineral concentrate production.
[2] Based on company 10-K information.

is owned by Foote Mineral Co. The estimated annual $LiCO_3$ capacity for the Chilean brine operation is expected to be 6,350 mt. Amax has also recently entered into an agreement with the Chilean Government to investigate producing lithium and other industrial minerals from the same Atacama basin. Exact quantities of lithium to be mined have yet to be determined by Chilean authorities (*9-11*). With increased production from the vast South American brine resources, it is likely that U.S. deposits will eventually control a smaller share of the world lithium market.

Greenbushes in Australia currently produces high-grade spodumene concentrate for the ceramic and glass industry. The production from this mine started in 1982, and the spodumene concentrate is exported to Europe and Japan for applications in ceramics. The Bikita Mine in Zimbabwe produces petalite and spodumene or use in specialty glasses and ceramics.

Production statistics from centrally planned economy countries (CPEC's) are not well reported. Although the U.S.S.R. is probably the second largest producer of lithium products, most production is consumed internally. China is also a leading producer of lithium chemical products and is one of the leading exporters to the United States (*1*).

Market prices for the lithium products used in this evaluation are:

Lithium carbonate, 99.5 pct Li_2CO_3	$1.54/lb.
Spodumene concentrate, 4 to 7 pct Li_2O	$200/mt to $350 mt.
Petalite concentrate, 3.5 to 4.5 pct Li_2O	$185/mt to $250/mt.

Current market prices are adapted from published sources and personal communications (*12-14*). Prices for spodumene and petalite concentrates depend largely on the grade and purity of the product and are often purchased under contract.

Most U.S. hard-rock production and all brine production is marketed in the form of $LiCO_3$ and other lithium chemical products. Currently, all hard-rock production from Greenbushes, Bernic Lake, and Bikita is sold as mineral concentrates. Cost data were not available to estimate the production costs of Li_2CO_3 from spodumene concentrates. For this reason, this analysis only reports the costs of producing spodumene concentrates.

EXTRACTION AND PROCESSING TECHNOLOGY

Lithium is recovered either by surface or underground mining of lithium-bearing pegmatites or by pumping of lithium-rich brines to evaporative ponds. Table 2 shows specific deposit data for mines and deposits included in this study.

PEGMATITES

Mining of pegmatites is by conventional open pit or underground mining methods. Selective mining is important at the Bikita Mine in Zimbabwe to separate the spodumene from amblygonite, as the latter is deleterious to the downstream processing.

Beneficiation often consists of recovery of the tin, columbite-tantalite, or other minerals by gravity methods followed by flotation to recover spodumene. The spodumene concentrate is further upgraded by desliming along with magnetic separation to remove iron. Low iron content makes the concentrate advantageous for ceramic use. Concentrates can be used in either ceramics or glass or as feed to a Li_2CO_3 plant. Some plants produce a feldspar-quartz (glasspar) concentrate that is used in the glass industry.

Spodumene ore concentrated by the flotation process can produce a concentrate as high as 7 pct Li_2O (3.35 pct Li) with an 80-pct mill recovery. One feature of some concentrators is desliming after grinding; the gangue minerals are softer than the spodumene and therefore some of them grind finer and are discarded as slimes (minus 200 mesh).

$LiCO_3$ is produced using a H_2SO_4 process. The H_2SO_4 process involves treating a spodumene concentrate of about 6.0-pct Li_2O (2.79 pct Li). The concentrate is first heated to 1,075° to 1,100° C in a kiln to produce a more reactive and soft beta spodumene. This calcine is cooled, then mixed with concentrated H_2SO_4 and heated to 250° C in an acid roaster to dissolve the lithium. The acidified concentrate is neutralized with ground limestone and filtered, resulting in an impure solution of lithium sulfate (Li_2SO_4). The solution undergoes filtering, pH adjustment, and evaporation steps and is then reacted with Na_2CO_3 to produce Li_2CO_3. Other lithium coproducts (i.e., LiOH, LiCl, etc.) can also be produced.

BRINES

Mining of lithium brines is accomplished using a series of shallow wells and pumping the brines to evaporative ponds for further concentration of the lithium. A typical operation would extract a brine containing about 0.125 pct Li and concentrate it to about 4.3 pct Li (design strength). The first group of ponds receives the brine and concentrates it to 0.4 pct Li while precipitating the salt as halite (NaCl) and potash as sylvinite (KCl). These minerals are periodically harvested from these ponds. A second group of ponds is used mainly to remove the magnesium, and the last group of ponds is used for final evaporation and storage of the brine for shipment to the chemical plant for precipitation of the lithium.

Conversion to Li_2CO_3 from the LiCl brine is a relatively simple process. It consists of using hydrated lime ($Ca(OH)_2$) for final magnesium and calcium precipitation and soda ash (Na_2CO_3) for precipitation of Li_2CO_3. This is done under close pH and temperature controls and with attendant filtering and washing steps common to most chemical extraction processes. The 99.5-pct Li_2CO_3 is the product that is either shipped to other plants for making downstream products (i.e., butyllithium, lithium hydroxide, etc.) or sold as is.

PRODUCTION COSTS

PEGMATITES

Mining and milling costs for hard-rock mines are as weighted averages based on a per-metric-ton-of-ore basis over the life of the operation. Wide variations in product grade and purity of the spodumene concentrates makes reporting of production costs misleading.

At the time of this analysis, four surface mines and one underground mine were operating. In addition, eight pegmatite deposits were evaluated, six as potential underground mines and two as potential surface mines.

Surface mining costs for the four pegmatite producers ranged from \$4/mt to \$19/mt of ore, with a weighted average of \$13/mt. The weighted-average mining cost for the proposed surface mines is estimated to be \$5/mt of ore. Major factors causing the higher costs for producers include greater labor and energy costs, and higher stripping ratios among the producers.

Bernic Lake is the only producing underground mine evaluated in this study. Costs for this operation are not presented in order to protect the confidentiality of proprietary data. Underground operating costs for nonproducers are estimated to range from \$28/mt to \$36/mt of ore with a weighted average of \$30/mt. Higher costs result from differences in productivity owing to complex mining methods and low mine capacity.

Beneficiation costs for producers are estimated to range from \$10/mt to \$19/mt of ore, with a weighted average of \$13/mt. Nonproducer beneficiation costs are estimated to range from \$6/mt to \$15/mt of ore, with a weighted average of \$10/mt. These costs include the expense for the recovery of all byproducts. Again higher cost for producers are the result of greater labor and energy costs.

Most spodumene and petalite concentrates are sold from main ports, generally in Europe and the United States. Concentrates that are treated into downstream products on site incur little or no transportation cost. The average cost for shipping concentrates to port was estimated to be \$45/mt of concentrate.

Costs associated with postmill processing of spodumene concentrates are not reported because of their highly proprietary nature.

BRINES

Production costs for brine operations are discussed in terms of $LiCO_3$. The product of all brine operations is assumed to be 100-pct Li_2CO_3. The Silver Peak operation in Nevada began producing $LiCO_3$ from brines in 1966. Operating costs for the Silver Peak Mine are estimated to be the lowest among the evaluated brine deposits, since all initial capital has been depreciated. Estimated capacities and processing costs for the Salar de Atacama Mine and proposed Salar de Uyani operation were modeled after the Silver Peak, with the exception of slight differences because of brine chemistries. Overall production costs for the South American deposits carry the added burden of capital recovery and therefore are greater than costs for the Silver Peak Mine. Capital costs are composed of wells, piping, salt recovery equipment, pond liners, and trucks (27 pct); chemical plant (48 pct); and infrastructure (25 pct). Based on an estimated annual output of 6,350 mt Li_2CO_3 for the three brine operations, the cost to process the brine to product was estimated to be less than \$1/lb Li_2CO_3 (far below the cost of Li_2CO_3 from spodumene). Transportation costs are not evaluated for brines because the Li_2CO_3 (the downsteam product) is produced on site.

AVAILABILITY

An evaluation was performed on 16 mines and deposits located in seven market economy countries (MEC's) to determine the availability of spodumene and petalite concentrates and Li_2CO_3 from brines. Both the total and annual availability was determined for all evaluated properties.

TOTAL RECOVERABLE

Five pegmatite mines and two brine operations were producing at the time of the study. Of the total 2.2 million mt of recoverable lithium available from all deposits evaluated (producers and nonproducers), 86 pct is available from the producers alone.

Pegmatites

The 12 primary spodumene properties could produce an estimated 25.9 million mt of spodumene concentrate containing a total of 717,000 mt of recoverable lithium. Currently producing spodumene mines account for a total of 18.4 million mt of recoverable spodumene concentrate (513,000 mt of contained lithium). Of the 25.9 million mt of spodumene concentrate available, 35 pct is from U.S. deposits, 34 pct from Canadian deposits, and 29 pct from the Greenbushes deposit in Australia. Total availability curves for the 12 spodumene properties are not presented because of difficulties in calculating individual total production costs, owing to variations in product grade and purity.

In terms of recoverable spodumene from demonstrated hard-rock resources, Australia's producing Greenbushes property contains the single largest recoverable spodumene resource evaluated in this study of 7.6 million mt (247,000 mt of recoverable lithium). The second and third largest spodumene properties are Kings Mountain and Bessemer City, with recoverable lithium resources of 115,000 mt and 101,000 mt, respectively.

Approximately 8.8 million mt of spodumene concentrate containing 242,000 mt of lithium is potentially recoverable from eight Canadian deposits. A total of 1.7 million mt of concentrate at 6.2-pct Li_2O are available from the Bernic Lake Mine. In 1985, Bernic Lake went from a pilot-scale operation into production and began mining and crushing spodumene ores. Bernic Lake had previously operated as a tantalum producer but temporarily suspended production in 1982 because of poor demand for tantalum.

Kitotolo in Zaire contains 430,000 mt of recoverable spodumene (12,000 mt of recoverable lithium). The small production of concentrates from Kitotolo results from low mill recoveries owing to complex ore mineralogies.

The Bikita Mine in Zimbabwe is the only evaluated hard-rock property that does not produce spodumene concentrate as its primary mill product. The mine can produce approximately 38,500 mt/yr petalite concentrate at 4.1-pct Li_2O. Bikita also produces lesser amounts of spodumene and lepidolite concentrates.

Brines

The three brine properties, Salar de Atacama, Chile; Salar de Uyuni, Bolivia; and Silver Peak, NV, contain over 7.8 million mt of recoverable Li_2CO_3 (1.5 million mt lithium). Large additional lithium resources exist in these deposits and in other brine deposits in South America, but have not been adequately quantified to be classified as demonstrated.

The weighted-average total costs at 0- and 15-pct discounted-cash-flow rates of return (DCFROR's) for the three properties evaluated are \$0.70/lb and \$1.40/lb of Li_2CO_3, respectively (both below the January 1985 market price of \$1.54/lb). The large difference primarily results from the higher return required on newly invested capital at the Salar de Atacama Mine and the large investment required to bring the Salar de Uyani into operation. The Silver Peak operation has a significant competitive advantage over the South American deposits, benefiting from capital depreciation.

ANNUAL CAPACITY

Potential total annual production was analyzed for the seven producers of lithium evaluated in this study; four produce spodumene concentrate (Bessemer City, Kings Mountain, Greenbushes, and Bernic Lake), one produces a petalite concentrate (Bikita), and two produce Li_2CO_3 from brines (Salar de Atacama and Silver Peak). Production data could not be plotted on the same curves owing to different product types and values, plus the small number of data points. However, potential annual production in terms of lithium equivalent is tabulated in table 4.

Table 4.—Potential annual lithium production from producing mines, metric tons contained lithium

	1985	1990	1995
Spodumene	5,225	7,126	7,126
Brines	1,690	2,380	2,380
Petalite	770	770	770
Total	7,685	10,276	10,276

In 1985 (table 4), the evaluated lithium producers had a total lithium capacity of 7,685 mt of recoverable lithium. This total was composed of lithium in spodumene (68 pct), Li_2CO_3 (22 pct), and petalite (10 pct). A comparison of 1985 estimated production capacity with the estimated decrease in reported lithium consumption from 1984 to 1985 signals an oversupply position for lithium producers (*4*). The decline in lithium consumption during 1985 is due in large part to the continuing cutback in production by the U.S. aluminum industry to 65 pct of capacity in 1985 (*4*).

By 1990, the potential annual capacity from current producers is estimated to be 10,276 mt recoverable lithium. This increase results largely from anticipated expansions at Greenbushes from 15,000 mt of 7-pct Li_2O (487 mt Li) to 25,000 mt (812 mt Li) in 1987 and from Bernic Lake, which is anticipated to reach production capacity of 55,000 mt of 6.2 pct Li_2O (1,576 mt Li) by 1990.

At present there appears to be adequate resources of lithium to meet the world's lithium needs to the end of the century. The four producing spodume operations and two brine operations have sufficient demonstrated resources to produce at capacity through 1995 and beyond. Currently, annual Li_2CO_3 capacity from the brines is 30 pct of the total lithium production among the evaluated producers. An important consideration in anticipating future market conditions is the likely development of additional production from the Atacama Basin, such as that planned by AMAX. Such a scenario will greatly increase the importance of South American brine resources in the world lithium product market.

CONCLUSIONS

Lithium is used in many commercial applications because of its many unique and versatile properties. The principal end use of lithium is in the production of primary aluminum. $LiCO_3$ added to the cryolite bath in aluminum potlines serves to lower operating pot temperatures and increase the electrical conductivity and overall efficiency of the bath. Large amounts of lithium are also used in the production of glass and ceramics, and lubricants.

The 16 lithium MEC properties evaluated for this study represent a demonstrated recoverable tonnage of 2.2 million mt of lithium. Producing operations account for about 86 pct of the total recoverable resources. Currently, the majority of lithium is mined from pegmatite ore bodies in the mineral form of spodumene.

The largest known lithium resource is contained in the brine deposits in South America. Demonstrated recoverable resources for the brines in this region total 1.4 million mt of lithium or 64 pct of the total evaluated. Large additional brine resources apparently exist in South America; however, these resources have not been adequately explored and are classified as inferred resources for the purposes of this evaluation.

Historically, the United States has dominated the production and sale of lithium products and is self-reliant in this commodity. Pegmatite bodies in North Carolina and a brine deposit in Nevada have yielded all of the domestic lithium for many years. However, the U.S. lithium industry's dominance of the world lithium market is likely to be diminished in the near future by production from Australian and Canadian pegmatite deposits and from the vast South American brine resources. Production from the Salar de Atacama Mine in Chile is likely to increase as the Chilean Government issues permits for additional brine operations.

The cost of producing Li_2CO_3 from brine deposits is far below the cost of producing Li_2CO_3 from pegmatitic minerals. This cost advantage, coupled with a current world lithium production capacity much larger than current market demand, will likely weaken the U.S. position as the major supplier of lithium. In fact, the potential for low-cost recovery of lithium from brines could put Chile in the position to potentially control the MEC lithium market.

REFERENCES

1. Ferrell, J. E. Lithium. Ch. in Mineral Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 461-470.
2. ______. Lithium. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 92-93.
3. Searles, J. P. Lithium. Ch. in BuMines Minerals Yearbook, v. 1, 1985, pp. 645-651.
4. Norton, J. J. Lithium, Cesium and Rubidium—The Rare Alkali Metals. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, p. 367.
5. Bleiwas, D. I., and J. S. Coffman. Lithium Availability—Market Economy Countries. A Minerals Availability Appraisal. BuMines IC 9102, 1986, 23 pp.
6. Foote Mineral Co. 1982 10-K Report, p. I-3.
7. Greenbushes Tin Ltd. 1982 Annual Report, p. 7.
8. Gulf Resources Corp. 1982 10-K Report, p. 9.
9. Latin American Mining Letter. Amax Signs Chilean Salt Deposits Agreement. V. 5, No. 3., Feb. 5, 1986, p. 1.
10. ______. Amax in Chile Lithium Talks. V. 3, No. 17, Sept. 12, 1984, p. 1.
11. Industrial Minerals (London). Amax and Molymet Win Lithium Concession. Sept. 1984, p. 17.
12. ______. Prices. No. 207, Dec. 1984.
13. Alexander, J. H. Lithium. Eng. and Min. J., v. 187, No. 5, May 1986, pp. 48-49.
14. Jacc' son, R. N. (Foote Mineral Co.). Private communication, 1986; available upon request from R. G. Christiansen, BuMines, Denver, CO.

MAGNESIUM

INTRODUCTION

Magnesium, the eighth most abundant element in the Earth's crust, plays an important role in a wide variety of industrial and military applications. It is in an unusual position in that it has the capability of being recovered in both metallic and nonmetallic forms from multiple renewable sources from many countries of the world. This important commodity is marketed in many product forms. Its low density has encouraged its use in structural applications as an alloying agent with aluminum for use in machinery, aircraft, and automobiles. Refractory magnesia products play an important role in the iron and steel industries. Other markets for magnesium products include the chemical, agricultural, construction, and paper industries.

Both magnesium metal (Mg) and magnesium compounds are produced from magnesium-bearing ores (magnesite, dolomite, brucite, and olivine), seawater, and brines. The most important magnesium compound is magnesium oxide (MgO), commonly referred to as magnesia. Various magnesia products are made by calcining magnesium carbonate or hydroxide at different temperatures. Deadburned magnesia is a granular product obtained by calcining magnesia above 2,640° F to form a high-grade refractory product. Caustic calcined magnesia is a reactive MgO product formed at calcination temperatures less than 1,640° F and is used in lower grade refractories, chemicals, and other industrial applications.

This chapter updates information presented in Bureau of Mines Information Circular 9112 "Magnesium Availability—Market Economy Countries. A Minerals Availability Appraisal" (*1*).[1] This commodity availability update complements Mineral Commodity Summary data on magnesium metal and magnesium compounds (*2*) and updates past magnesium availability data reported by the Bureau (*3*).

GEOLOGY AND RESOURCES

GEOLOGY

Magnesium occurs in a variety of rock types, seawater, lake brines, and well brines. Although magnesium is found in over 60 minerals, only dolomite, magnesite, brucite, and olivine are minerals of commercial importance.

Dolomite [$CaMg(CO_3)_2$] is a sedimentary rock interbedded with limestone and occurs as massive or bedded deposits throughout the world. In the United States alone, dolomite deposits containing at least 37.5 pct $MgCO_3$ occur in over a dozen States. Magnesite ($MgCO_3$), the natural form of magnesium carbonate, occurs in four types of deposits: as crystalline masses replacing dolomite, as impure crystalline masses replacing ultramafic rocks, as cryptocrystalline masses in ultramafic rocks, and as sedimentary beds and lenses. In recent years, only deposits of crystalline magnesite that had replaced dolomite have been mined. Brucite [$Mg(OH)_2$], the natural form of magnesium hydroxide, is a secondary origin mineral found in association with other magnesium minerals, particularly magnesite. It is usually associated with carbonate rocks and serpentine. Additional geologic information can be found in U.S. Geological Survey Professional Paper 820 (*4*).

Magnesium occurs within seawater at an average concentration of 0.13 pct by weight. Magnesium recovery plants are located at sites of relatively high salinity, reflecting higher than average concentrations of dissolved salts. The world's oceans are estimated to contain 17.8 billion st of contained magnesium.

Lake brines commonly occur in enclosed drainage basins where water loss through evaporation exceeds water gained through precipitation. The Dead Sea brines in Israel and the Great Salt Lake brines in the United States are examples of natural brines. Well brines are extracted either from near-shore aquifers or naturally concentrated interstitial brines. Magnesium concentrations in interstitial brines are higher than those of seawater, but lower than the best lake brines.

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

RESOURCES

Magnesium and magnesium compounds can be produced from numerous sources in virtually unlimited quantities from many countries. The U.S. Geological Survey estimated in 1973 that identified world resources of magnesite total over 12 billion st; identified brucite resources total several million short tons; resources of dolomite and other magnesium-bearing evaporite minerals were enormous; brines constitute resource material containing billions of short tons; and seawater resources were considered unlimited (*4*).

Table 1 presents the 1985 demonstrated market economy country (MEC) magnesium resources available from those sources considered the most likely to produce; e.g., those currently producing, those with recent production, and those not currently producing but considered for development. Table 2 is a list of the properties evaluated in this study.

Resources have been classified into four primary sources: seawater, brines, magnesite, and dolomite. Discussion of dolomite sources is limited to those properties producing dolomite solely for its magnesium content, rather than for its numerous other uses. Resources from properties recovering magnesium products from brine or seawater sources were reported in terms of a 30-yr production life. Total demonstrated resources from these properties are estimated at 33,070 million st source material containing 427.5 million st MgO, of which 68 pct is recoverable, either as magnesium metal or compounds. Additional magnesium resources of 2,101 million st containing 351 million st MgO are available at the inferred resource level; inferred resources from brines and seawater are considered unlimited. Magnesite deposits account for 57 pct (165.7 million st) of the total available MgO included in the properties evaluated in this study. Brines account for 28 pct (80.6 million st) of total recoverable MgO, seawater sources account for 13 pct (38.9 million st), and dolomite deposits account for the remaining 2 pct (8.0 million st) total available MgO.

Figure 1 shows the source distribution of resources for principal producing MEC's. Those countries containing over 10 pct of demonstrated world magnesium resources are the United States and Mexico (53 million st recoverable MgO,

Table 1.—Summary of MEC demonstrated magnesium resources evaluated for this study, as of January 1985

Country/ Primary source	Current status[2]	Demonstrated: In situ ore, 10^6 st[3]	Demonstrated: MgO grade, pct	Demonstrated: Contained MgO, 10^6 st[4]	Demonstrated: Recoverable MgO, 10^6 st[5]	Identified[1]: In situ ore, 10^6 st	Identified[1]: MgO grade, pct	Identified[1]: Contained MgO, 10^6 st	Annual production capacity, 10^3 st product	Product type[6]
Brazil: Magnesite	P	W	46	W	W	W	46	W	330	DB,CC
Canada:										
Dolomite	P	11	21	2.4	2.4	11	21	2.4	11	MG
Magnesite	N	136	30	41	41	176	30	53	205	DB,CC
Greece: Magnesite	C	W	6.4	W	W	W	6.4	W	410	DB
India: Magnesite	P	334	9.5	32	2.2	1,583	9.8	155	133	DB,CC
Ireland: Seawater	C	2,473	.19	4.7	4.7	(7)	.19	(7)	200	DB
Italy: Seawater	P	1,031	.23	2.4	2.4	(7)	.23	(7)	72	DB
Japan: Seawater	P	17,007	.20	31	18	(7)	.20	(7)	691	DB,CC,MG
Mexico:										
Seawater	P	1,012	.20	2.0	2.0	(7)	4.5	(7)	77	DB
Brine	P	1,128	4.5	51	51	(7)	4.5	(7)	110	DB,CC
Netherlands: Brine	P	29	9.0	2.6	2.6	59	9.0	5.3	110	DB
Norway: Seawater	P	1,356	.20	2.7	2.7	(7)	.20	(7)	55	MG
Spain: Magnesite	P	W	32	W	W	W	32	W	105	DB,CC
Tunisia: Brine	N	59	6.8	4.0	4.0	1,102	6.8	75	110	DB
Turkey: Magnesite	P	437	27	118	49	1,105	26	287	205	DB,CC
United Kingdom: Seawater	P	1,936	.21	4.0	4.0	3,287	.21	6.9	193	CC,DB
United States:										
Seawater	P	3,636	.21	7.6	5.1	(7)	.21	(7)	468	DB
Brine	P	1,880	1.2	23	23	(7)	1.2	(7)	580	DB
Magnesite	C	W	26	W	W	W	26	W	123	DB,CC
Dolomite	C	W	39	W	W	W	39	W	27	DB,MG
Total or averages:										
Seawater	NAp	28,451	NAp	54.4	38.9	(7)	NAp	(7)	1,756	NAp
Brine	NAp	3,096	NAp	80.6	80.6	(7)	NAp	(7)	910	NAp
Magnesite	NAp	1,499	NAp	285.0	165.7	3,600	NAp	636	1,511	NAp
Dolomite	NAp	24	NAp	7.5	7.5	24	NAp	7.5	38	NAp
Grand Total	NAp	33,070	NAp	427.5	292.7	(7)	NAp	(7)	4,215	NAp

NAp Not applicable. W Withheld to avoid disclosing company proprietary data included in totals.

1 Identified resources include inferred resources; seawater and some brine resources are considered virtually unlimited.

2 C, combined producer and nonproducer; N, nonproducer; P, producer.

3 Resources for seawater and brines are effectively inexhaustible; for the purpose of this study a property life of 30 yr was assumed for estimates of resources from these sources.

4 Data report total MgO content of in situ ore.

5 Data report MgO recoverable from deposits, assuming current recovery rates.

6 CC, caustic calcined MgO; DB, deadburned MgO; MG, Mg metal. Principal products only; other products may be recovered in limited quantities.

7 Unlimited resource.

each), Turkey (49 million st MgO), and Canada (43.4 million st MgO). Other MEC's with large commercial magnesium resources are Brazil, Greece, India, Ireland, Italy, Japan, Netherlands, Norway, Tunisia, and the United Kingdom.

Significant magnesium resource potential, particularly from magnesite sources, is available from centrally planned economy countries (CPEC's) not included in table 1. Magnesite resources from China are reported to account for 29.3 pct of the world magnesite reserve base (*2*). North Korea accounts for 17.5 pct, the U.S.S.R. accounts for 25.7 pct, and Czechoslovakia accounts for 0.7 pct of world magnesite reserve base potential.

Table 2.—MEC magnesium properties included in this study

Deposit	Ownership	Current status[1]	Mining type[2]	Process[3]	Products[4]
Brazil: Magnesita	Magnesita S.A.	P	S	DB,LB,MT	DB,CC
Canada:					
Haley	Chromasco Ltd	P	S	PI	MG
Mt. Brussilof	Baymag Mines Ltd	N	S	LB	CC
Timmins	Canadian Magnesite Mines Ltd.	N	S	DB	DB
Greece:					
Fimisco	FIMISCO	P	S	DB	DB
Larco	Larco S.A.	N	S	DB	DB
India:					
Almora	Almora Magnesite S.A.	P	S	DB,LB	DB,CC
Tamilnadu	TamilNadu Magnesite (Government owned).	P	S	DB,LB	DB,CC
Burn Standard	Burn Standard Co. (Government owned).	P	S	DB,LB	DB,CC
Dalmia	Dalmia Cement Co	P	S	DB	DB
Ireland:					
Drogheda	Premier Periclase Ltd	P	SW	DB	DB
Quigley	Quigley Magnesite Co	N	SW,S	DB	DB
Italy: Cogema	COGEMA	P	SW	DB	DB
Japan:					
Ube	Ube Industries Ltd	P	SW	DB,LB,PI	DB,CC
Onahama	Asahi Chemical Industries	P	SW	DB,LB	DB,CC
Minamata	..do	P	SW	DB	DB
Mexico:					
Quimica del Rey	Industrias Penoles S.A.	P	S,B	DB,LB	DB,CC
Quimica del Mar	..do	P	SW	DB	DB
Netherlands: Veendam	Billiton International Metals B.V.	P	S,B	DB	DB
Norway: Norsk Hydro	Norsk Hydro A/S	P	SW,B,S	NH	MG
Spain: Zubiri	Magnesitas Navarras S.A.	P	S	DB,LB	DB,CC
Tunisa: Zarzis	Government of Tunisia	N	B	DB	DB
Turkey:					
Comag	Comag	P	S	LB	CC
Kumas	Kumas	P	S	DB	DB
Sumerbank	Sumerbank Genel Mudurlugu	P	S	DB	DB
United Kingdom: Hartlepool	Steetley Refractories Ltd	P	SW,S	LB,DB	CC,DB
United States:					
California: Moss Landing	National Refractories and Minerals Corp.	P	SW,S	DB	DB
Delaware: Barcroft	Barcroft Co	P	SW	HM	DB
Florida: Basic Magnesia	Combustion Engineering	P	SW,S	DB	DB
Michigan:					
Ludington-Harbison	Dow Chemical, USA and Harbison-Walker Refractories.	P	BW	DB	DB
Midland Magnesia	Dow Chemical, USA and Martin Marietta Basic Products.	P	BW	DB	DB
M-M Manistee	Martin Marietta Basic Products	P	BW	DB	DB
Morton Chemical	Morton Chemical Corp	P	BW	DB	DB
Nevada: Basic, Inc	Combustion Engineering	P	S	DB	DB
Texas: Dow Freeport	Dow Chemical, USA	P	SW,S	DO	MG
Utah: Amax G.S.L	Amax Specialty Metals	P	B	AM	MG
Washington:					
Northwest Alloys	Northwest Alloys Inc	P	S	MT	MG
Stevens County Deposits	Harbison Walker Inc	N	S	DB	DB

[1] N, not producing as of January 1984; P, producing as of January 1984.
[2] B, sea or lake brines; S, surface; SW, seawater; W, brine wells.
[3] AM, AMAX process; DB, deadburning process; DO, Dow process; HM, hydromettalurgical process; LB, lightburning process; MT, Magnatherm process; NH, Norsk Hydro process; PI, Pidgeon process.
[4] CC, caustic calcined MgO; DB, deadburned MgO; MG, Mg metal. This study assumes recovery of principal products; minor byproducts may be recovered but are not included.

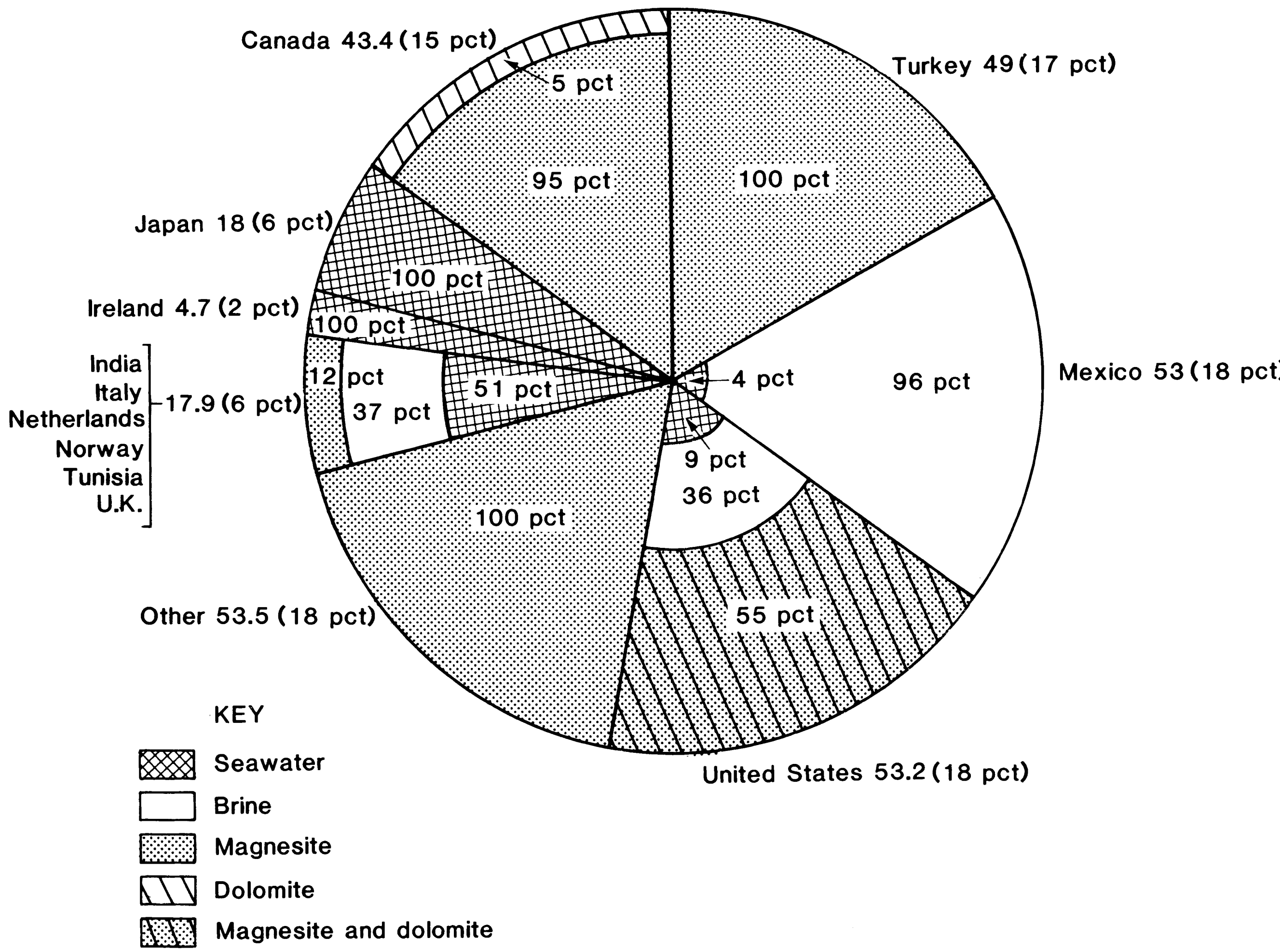

Figure 1.—Estimated 1985 demonstrated resource distribution of magnesium by country and source material (million metric tons of recoverable MgO product).

U.S. AND WORLD HISTORICAL PRODUCTION

Production of magnesium-related products is reported for magnesium metal and magnesite. Magnesium primary metal production in 1985 was estimated to be 357,000 st from nine countries (*2*). Magnesite production in 1985 was estimated to be 13 million st from 23 countries (*2*). Figure 2 shows world production for magnesium metal and magnesite in 5-yr intervals from 1950 to 1985. The United States leads the world in producing refined magnesium metal, but it is a relatively minor producer of raw materials for magnesium compound production.

Since 1950, magnesium metal production has increased sevenfold, with the United States assuming an increasingly dominant role in primary magnesium metal production. In 1984, the United States produced approximately 45 pct of the world's primary magnesium metal. Production from MEC's totaled 72 pct of the world total in 1984, primarily from the United States and Norway. Production from CPEC's, primarily the U.S.S.R., remained stable at 28 pct of world production.

Magnesite production since 1950 has roughly doubled, mainly as a result of increased capacity of CPEC's. In 1984, approximately 62 pct of world magnesite production occurred in CPEC's, primarily the U.S.S.R, China, North Korea, and Czechoslovakia. The remaining 38 pct came principally from the MEC's—Austria, Greece, Spain, and Turkey. Magnesite production from the United States has increased from the 7-pct share reported for 1950, but is still minor when compared with production from the other countries previously mentioned.

EXTRACTION AND PROCESSING TECHNOLOGY

Magnesium and magnesium products are recovered from a variety of sources. Each source requires specialized extraction and processing methods, with method selection dependent upon the desired marketable form. Methods employed at the operations evaluated in this study are reported in table 2.

SEAWATER AND BRINES

Extraction

The simplest source of magnesium to recover is seawater or magnesium-rich lake brines. Water is simply pumped from the shallow source area by means of centrifugal pumps. Pump intake is generally arranged in a series of weirs or sumps in order to prevent the pumps from being clogged with fish or other contaminants.

Magnesium-rich natural brines such as those recovered from the Michigan Basin are extracted by means of deep wells. Extraction methods and equipment are similar to that used in the oil industry. Well fields are usually put into production by contract drilling firms on a "turnkey" basis.

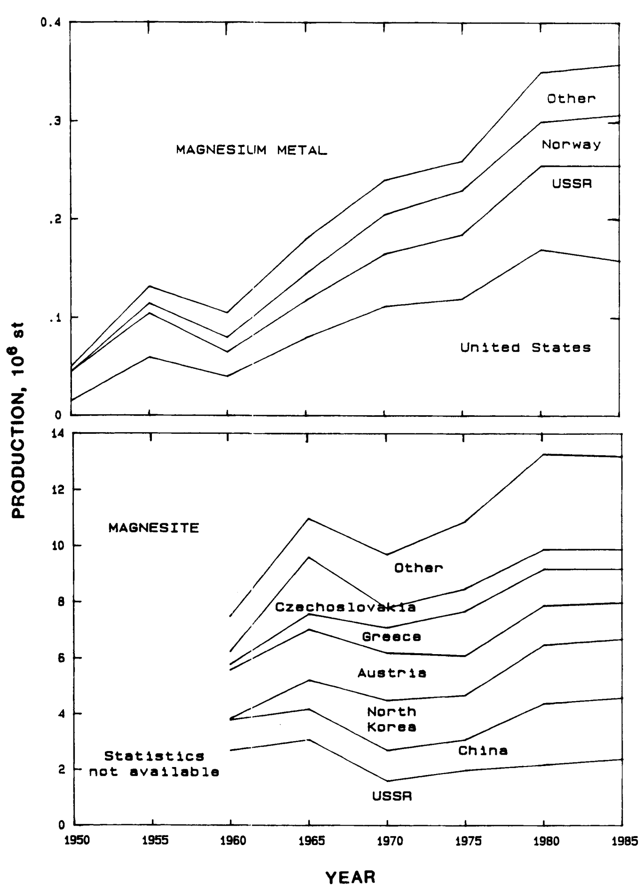

Figure 2.—World magnesium metal and magnesite production, 1950-85.

Michigan brine wells are sunk to depths ranging from 4,000 to 5,200 ft. Well casings are cement sealed from the surface to the top of the brine-producing aquifer. Pumping rates range from 20 to 120 gal/min. Brines are transported to processing facilities by a network of steel or fiberglass pipelines. Similar brine recovery techniques are employed worldwide.

Processing

Processing methods employed are selected based upon source material type and product forms desired.

Seawater or well brines are processed similarly, but minor differences occur at the initial stages of processing. If the solutions are used as feed for producing caustic calcined or deadburned MgO, carbonate levels are first reduced so insoluble calcium compounds do not precipitate with magnesium hydroxide in the subsequent reaction process.

Brines (accounting for 28 pct of the total recoverable MgO in this study) are first combined with slaked lime to precipitate soluble bicarbonates as calcium carbonate, while seawater (13 pct of recoverable MgO) is commonly treated with sulfuric acid to liberate carbon dioxide. Subsequent processing steps for these two sources are identical. The treated solution is then heated to approximately 131° F and placed into reaction vessels where calcined oyster shells, limestone, or dolomite are added. Magnesium ions in the water react with the slaked carbonates to precipitate $Mg(OH)_2$. This magnesium hydroxide slurry is concentrated in thickeners, washed with fresh water in a countercurrent system, then filtered. The resulting filter cake can either be dried and marketed at this point as magnesium hydroxide or it can undergo further calcining to attain the desired magnesia product. Different calcination temperatures are used for different products.

When dolomite is used as a reactant, approximately half of the resulting magnesium in the magnesia product comes from the dolomite; when limestone is used, all the product magnesia comes from the seawater or brine. In order to obtain an equivalent production rate in terms of marketable product, twice the volume of liquid must be processed when limestone is used as compared to when dolomite is used.

Magnesium sulfate (epsom salts) can be produced by dissolving magnesia in sulfuric acid with subsequent crystallization, or by reacting magnesium hydroxide with SO_2. Various grades of magnesium carbonate can be produced by combining solutions of magnesium sulfate and sodium carbonate followed by precipitation, filtration, and drying (*5*).

MAGNESITE, BRUCITE, DOLOMITE, AND OLIVINE

Extraction

Magnesite, brucite, dolomite, and olivine are the principal rock sources of magnesium. Open pit mining methods are most commonly used. Mining consists of stripping overburden and drilling, blasting, loading, and hauling ore.

Hand cobbing of broken material is employed at some locations, primarily in India and Turkey. Crushers located in the pit are utilized at other locations. Overburden and waste removal vary significantly for each operation. Waste-to-ore ratios range from zero to as much as 3:1.

Nonmetallic Magnesia Processing

The first step in processing magnesium-bearing ores is to crush either run-of-mine or hand-cobbed ore in multiple stages either at the mine site or the processing facilities. Dolomite (source for 2 pct of recoverable MgO in this study) at this point is simply directed to rotary or shaft kilns where it is calcined. Magnesite and brucite ores (57 pct of recoverable MgO) undergo more rigorous beneficiation before calcination. After undergoing a series of crushing stages, ore is screened, washed, and classified to remove slimes. Depending upon ore quality, flotation or heavy-media separation may be required to prepare the feed for calcination.

Two forms of magnesia are commonly produced, caustic calcined MgO or deadburned MgO. Caustic calcined MgO is obtained by heating the feed material in either rotary or shaft kilns to above 1,640° F. Deadburned material is produced by pelletizing and additional kilning at temperatures up to 3,450° F. Both brick and maintenance grade refractory magnesia products can be produced (*3*).

Magnesium Metal Processing

Two principal processes are currently utilized for magnesium metal recovery: electrolytic reduction of magnesium chloride and silicothermic reduction of magnesium oxide. Exact process methodology varies from operation to operation.

Electrolytic Processing

Several basic electrolytic cell designs are in use today; the three main types currently used are the Dow, the I.G., and the diaphragmless cells. Cell design differs in electrode positioning and utilization of a diaphragm for separating the electrodes. In each cell, direct current breaks down the magnesium chloride into chlorine gas and molten magnesium. The metal is formed at the cathode and rises to the surface of the bath where it is guided into storage wells prior to casting into desired forms.

AMAX utilizes solar evaporation to preconcentrate Great Salt Lake brines prior to electrolysis. The increased lake level in recent months has flooded many of the solar evaporation ponds, resulting in a need to adjust recovery methods.

Silicothermic Processing

Dolomite is presently used as the magnesium source material and ferrosilicon is used as a reducing agent. In this type of processing, calcined dolomite, ferrosilicon, and alumina are ground, heated, and briquetted, then heated in electric furnaces operated under a vacuum at temperatures up to 2,900° F. The alumina serves as a fluidizing agent and reduces the melting point of the slag produced as the result of the dolomite-silicon reaction. Magnesium vapors are condensed and magnesium metal is then cast in desired forms.

NEW TECHNOLOGY

The Bureau recently published results of research on producing refractory magnesia bricks from chemically modified periclase grains and magnesium hydroxide slurries. Strength values of the new bricks were found to be superior to commercially used 98 wt pct magnesia bricks. These refractories could potentially be substituted for magnesia chrome products containing high proportions of imported chromite (*6*).

Scientists at the General Motors Research Laboratories have developed an experimental electrolytic process to remove magnesium from scrap aluminum-based alloys. This process utilizes a three-layered electrolytic cell that reportedly could provide an alternative to the traditional chlorination process (*7*).

Utilizing a new single-step reactor process to produce primary magnesium metal from magnesite, Aluminum Company of America (Alcoa) and MPLC Holdings S.A. plan to construct a 55,000-st plant in three stages near Aldersyde, Alberta. All current Mg metal producers use either seawater, brine, or dolomite source materials (*8*).

PRODUCTION COSTS

Production costs vary greatly depending upon such factors as source material, processing methodology, size of operation, deposit location, ore characteristics, energy and labor costs, tax structure, and degree of product integration and diversification. The diversity of source materials and marketed products make economic comparisons between magnesium-producing properties difficult. For the purposes of economic analysis, properties are grouped in terms of primary product and source material. A detailed breakdown of costs is not possible because of proprietary constraints.

Capital investments for the production of magnesium metal range from $3,200/st to $4,500/st annual capacity, while capital requirements for refractory magnesia production range from $900/st to $1,300/st annual MgO capacity (1985 dollars).

Operating costs for magnesium metal from all sources range from $280/st to $2,070/st Mg metal product. The weighted-average operating cost for the production of magnesium metal was $1,020/st product. Operating costs for seawater operations tended to be less than those from dolomite or brine source material; operating costs from brine operations were found to be the highest.

Operating costs from evaluated properties producing MgO compounds ranged from $22/st to $386/st product. The weighted-average operating cost for nonmetal magnesium compounds was \$172/st product. Costs from seawater and magnesite properties were significantly higher than those from properties producing brines.

Figure 3 gives a breakdown, by percentage, of operating cost for major cost components. Labor costs range from 13 to 31 pct of the operating cost. Energy costs range from 22 to 71 pct, while materials-supplies costs make up 6 to 49 pct. Energy costs are the most significant component of most production costs.

A summary of total production costs for signficant operations producing Mg metal and selected MgO compounds is reported in table 3. Costs for both Mg metal and MgO compounds are reported in terms of cost per short ton of contained MgO for all magnesium products. Both cost ranges and weighted-average costs are reported. In addition to operating costs, total production costs include costs for capital recovery, taxes, and selected byproduct credits; the 15-pct discounted-cash-flow rate of return (DCFROR) cost also includes recovery on capital (profit) at a 15-pct DCFROR rate.

The average total cost of production at a 15-pct DCFROR to produce magnesium metal ranges from \$720/st to \$2,640/st (\$0.36/lb to \$1.32/lb) Mg metal product for the properties under consideration. The weighted-average total

Table 3.—Magnesium production costs for selected magnesium products

(January 1985 U.S. dollars per short ton primary product)

Product and source	0-pct DCFROR		15-pct DCFROR	
	Cost range	Wtd av cost	Cost range	Wtd av cost
Mg metal[1]	400-2,380	1,370	720-2,640	1,579
MgO COMPOUNDS				
Deadburned MgO:				
Seawater	153- 396	249	177- 573	290
Brines	122- 307	191	172- 330	302
Magnesite and brucite	92- 324	202	113- 467	228
Wtd Av	NAp	236	NAp	276
Caustic calcined MgO:				
Seawater	186- 296	193	198- 331	208
Brine	W	W	W	W
Magnesite and brucite	78- 273	147	96- 393	227
Wtd av	NAp	190	NAp	213

NAp Not applicable

W Withheld to avoid disclosing proprietary data included in totals.

[1] The proprietary nature of the data prevents reporting of actual production costs for Mg metal. Weighted-average costs from all source materials are reported.

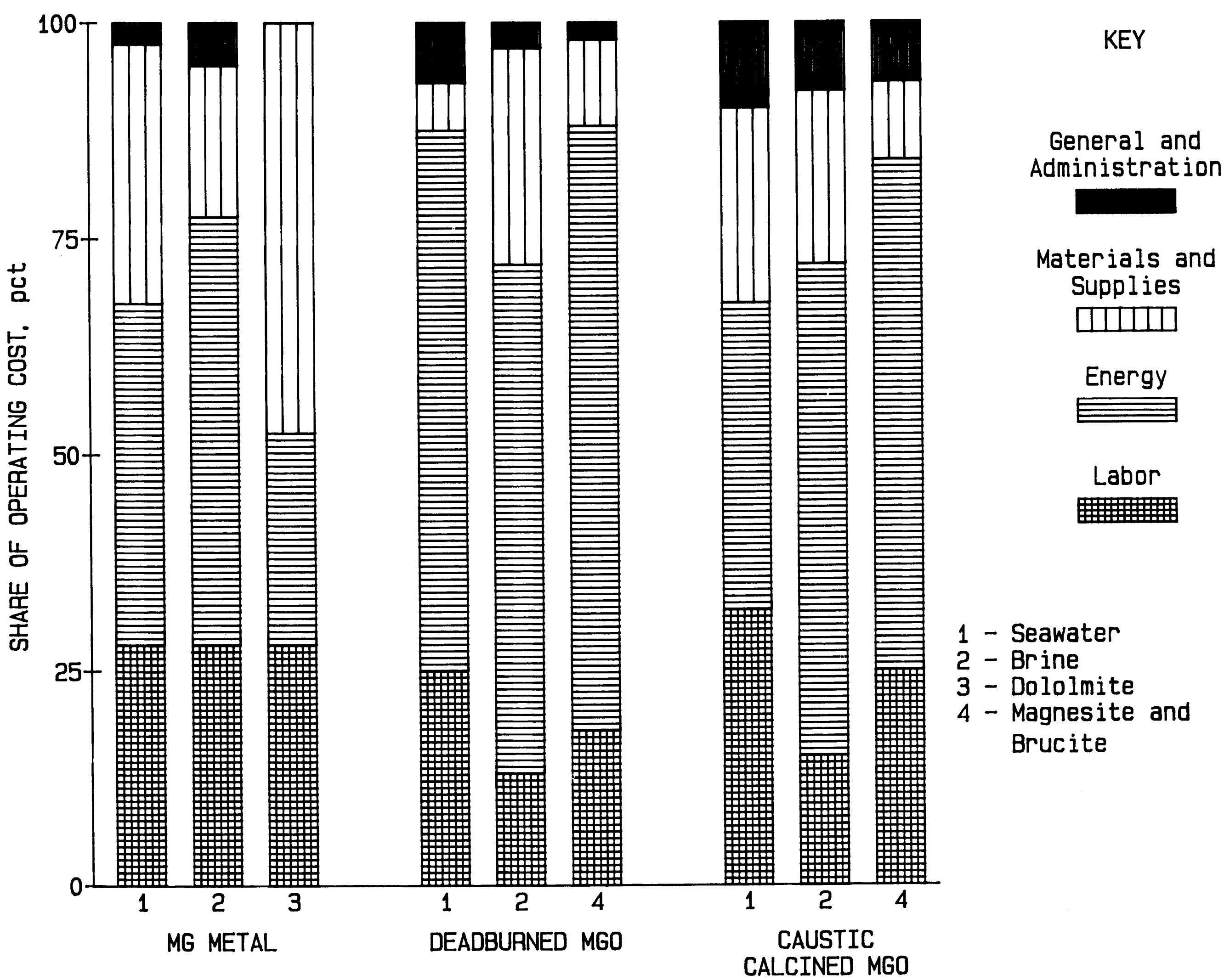

Figure 3.—Magnesium production costs, as a percent of total operating cost, for producing operations in MEC's.

production cost for magnesium metal producers is \$1,579/st (\$0.79/lb) Mg metal product. Given the proprietary nature of the data, actual costs could not be reported for each source material. On a relative basis, total production costs from seawater sources were lower than costs from dolomite or brine source materials; brine costs to recover Mg metal were the highest from the properties evaluated.

For producing operations, the average total cost (including a 15-pct DCFROR) to produce deadburned MgO ranges from \$177/st to \$573/st from seawater, \$172/st to \$330/st from brines, and \$113/st to \$467/st from magnesite or brucite. Costs average \$290/st from seawater, \$302/st from brines, and \$208/st from magnesite; with a weighted-average cost from all sources of \$276/st MgO. The January 1985 market price for deadburned MgO was \$392/st MgO product.

The average total cost (at a 15-pct DCFROR) for producing operations to recover caustic calcined MgO as a primary product ranges from \$198/st to \$331/st from seawater and \$96/st to \$393/st from magnesite or brucite. Average costs were \$208/st from seawater and \$227/st from magnesite or brucite properties. The weighted-average cost, including brine sources, was \$213/st MgO product. The January 1985 market price for chemical-grade caustic calcined MgO was \$330/st MgO product, well above the average total production cost.

Costs to recover caustic calcined MgO as the primary recoverable MgO product average 77 pct of the costs to recover deadburned MgO as the primary product. This is to be expected where process technology is similar and only an additional processing step is required to produce the higher grade deadburned MgO product.

Because of proprietary considerations, detailed costs for producing and nonproducing properties could not be provided. Nonproducing properties evaluated in this study have costs that average 57 pct more than costs from producing properties recovering from a similar source material.

AVAILABILITY

For the purpose of availability analysis, three principal recoverable products were assumed: Mg metal, deadburned MgO, and caustic calcined MgO. Total and annual availability curves were developed for each of these three products. Additional costs for further processing of these products to meet end-use requirements have not been included in this evaluation.

TOTAL RECOVERABLE

The total potential availability of selected magnesium products from properties included in this study is reported in the figures included in this section. Of the 38 properties evaluated, 5 operations recover magnesium metal, 22 recover deadburned MgO, and 11 recover caustic calcined MgO as the principal magnesium product. At the time of evaluation, 32 of these properties were producing. Because of the limited number of nonproducing properties evaluated, separate curves for nonproducers were not possible. Production costs for properties in CPEC's were not estimated owing to lack of information.

The tonnage of magnesium metal, deadburned MgO, and caustic calcined MgO potentially available from evaluated producing properties in MEC's is shown in figure 4. Costs are bracketed by lower and upper cost levels. The lower level reflects costs at a 0-pct DCFROR, while the upper level includes a 15-pct DCFROR on invested capital.

The January 1985 market price for magnesium metal ingots was approximately \$1.34/lb *(9)*. Approximately 13 million st Mg metal is potentially available at an average total cost equal to the reported market price. All properties included in this study are currently producing and incurred costs (including a 15-pct DCFROR) below the market price for magnesium metal.

The January 1985 market price for deadburned MgO was approximately \$392/st *(9)*. Approximately 107 million st MgO is potentially available from evaluated deposits (103 million st from producers), which have production costs (including a 15-pct DCFROR) at or below this price. An additional 23 million st would become available if the deadburned MgO price rose to \$500/st (at a 15-pct DCFROR).

The January 1985 market price for caustic calcined MgO was \$330/st *(9)*. At that price, approximately 44 million st MgO is potentially recoverable from evaluated deposits assuming a 15-pct DCFROR. All but one of these deposits are producers.

The domestic magnesium industry has the capability to produce approximately 9.6 million st Mg metal (201,000 st/yr) and 32 million st (1.2 million st/yr) deadburned MgO at the January 1985 market price for these products, assuming a 15-pct DCFROR. This equates to 73 pct of the magnesium metal and 24 pct of the deadburned MgO potentially available from evaluated world deposits. This is well above the projected domestic consumption level for magnesium products through the year 2000.

ANNUAL CAPACITY

Potential 1985 production at full capacity from operating properties in selected MEC's is shown in figure 5 for Mg metal, caustic calcined MgO, and deadburned MgO. The properties studied had the capacity to produce 275,000 st recoverable Mg metal and 2.8 million st recoverable deadburned MgO in 1985. Each of these curves is bracketed by a lower operating cost level and an upper cost level including a 0-pct rate of return. Most of the properties producing either Mg metal or deadburned MgO incurred estimated costs below the reported marked price of these products.

Analyses were also performed to estimate the annual production potential of the producing magnesium properties evaluated in this study. Since the general approach of this study was to evaluate the properties at full production capacity over the next 30 yr, the annual curves present total potential availability from producing properties for each year shown, rather than an assessment of future production.

Figures 6 and 7 show the potential annual availability for Mg metal, caustic calcined MgO, and deadburned MgO from producing operations based on a 15-pct DCFROR. At a production cost of $1.40/lb (the January 1985 market price was $1.34/lb), approximately 294,000 st Mg metal is potentially available annually between 1986 and 1989, increasing to 301,000 st between 1990 and 1995. This compares with 256,000 st primary magnesium metal produced in 1985 *(2)*. At a total production cost of $400/st (the January 1985 market price was $392/st), approximately 2.8 million st deadburned MgO is available annually between 1987 and 1995 from producing deposits. Approximately 430,000 st caustic calcined MgO is potentially annually available at total operating costs below the January 1985 market price of $330/st.

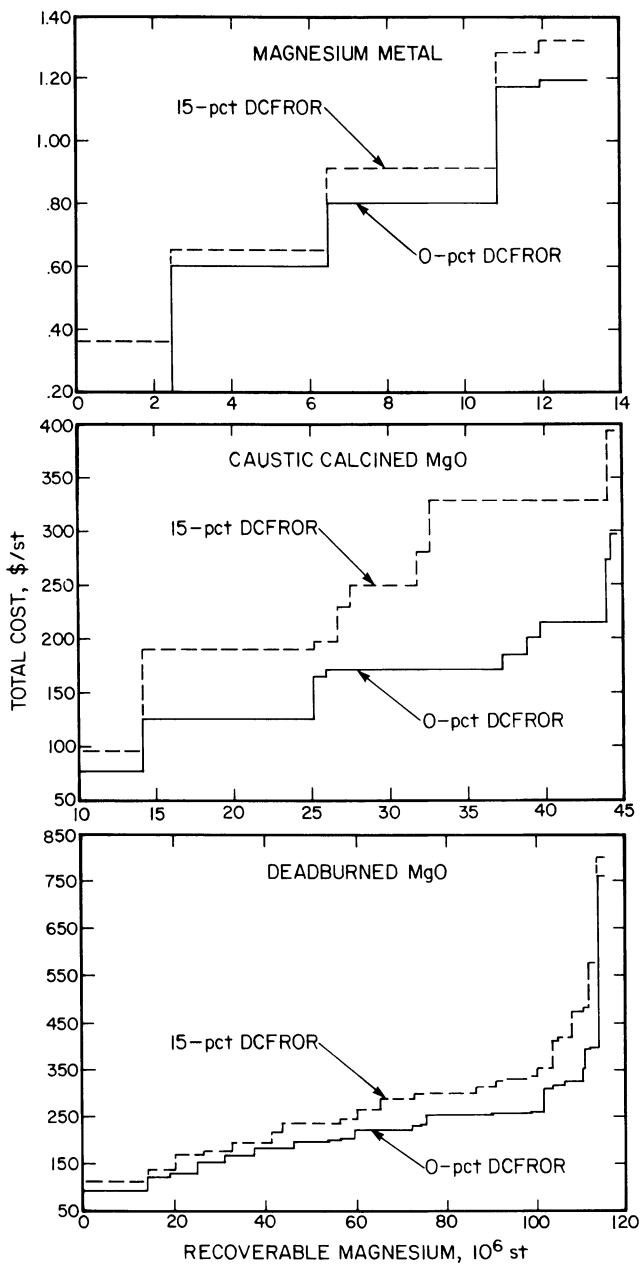

Figure 4.—Potential total magnesium metal, caustic calcined MgO, and deadburned MgO available from producing mines in MEC's (January 1985 U.S. dollars).

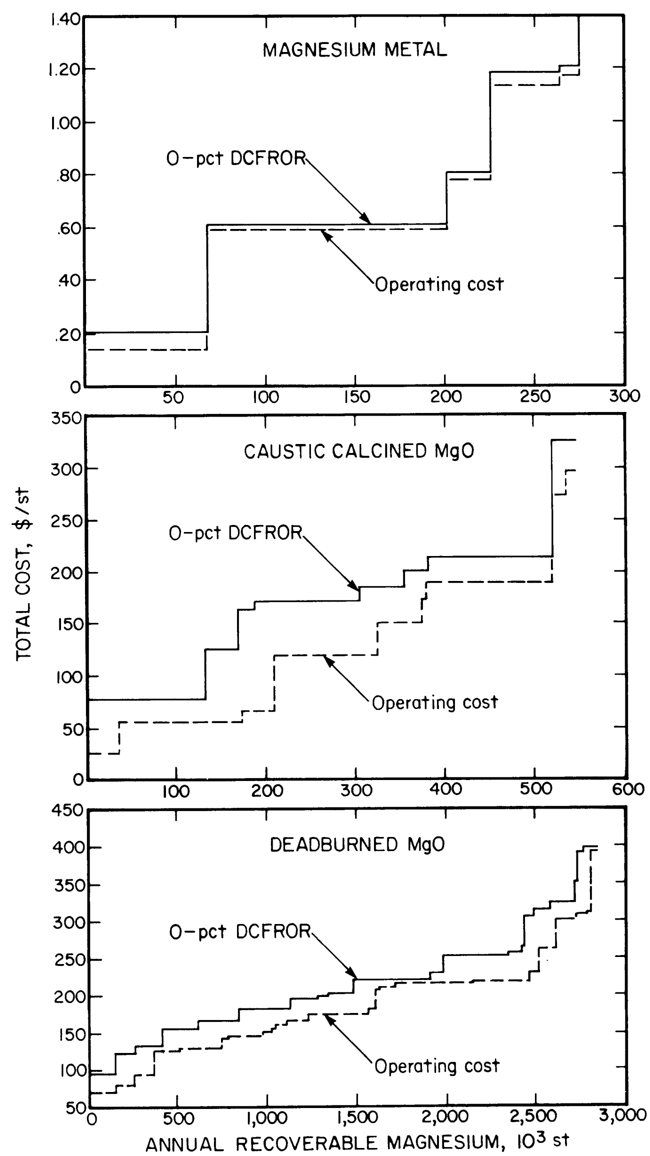

Figure 5.—1985 magnesium metal, caustic calcined MgO, and deadburned MgO capacities from producing operations in MEC's (January 1985 U.S. dollars).

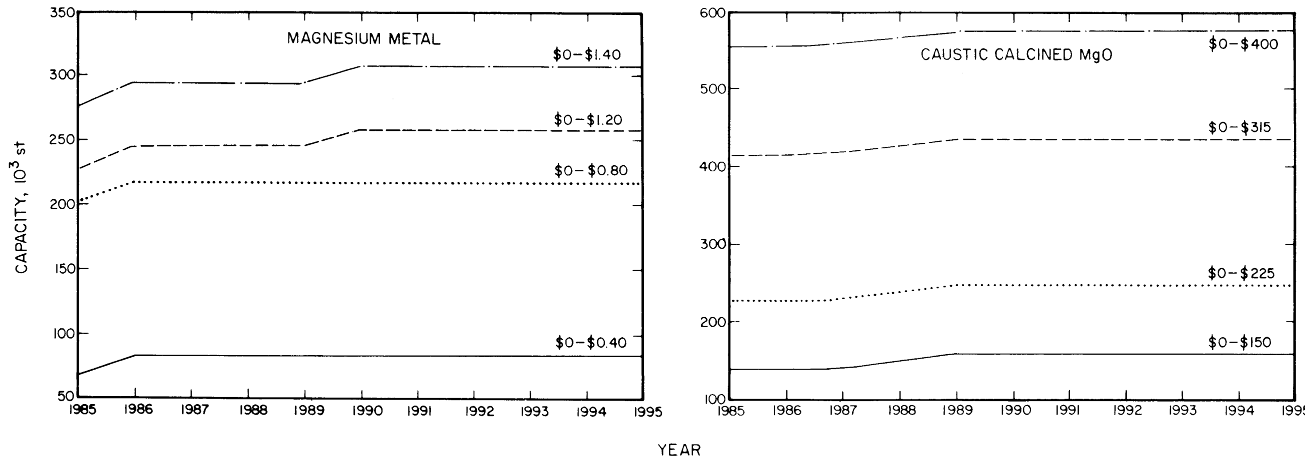

Figure 6.—Potential annual magnesium metal and caustic calcined MgO production from producing mines in MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

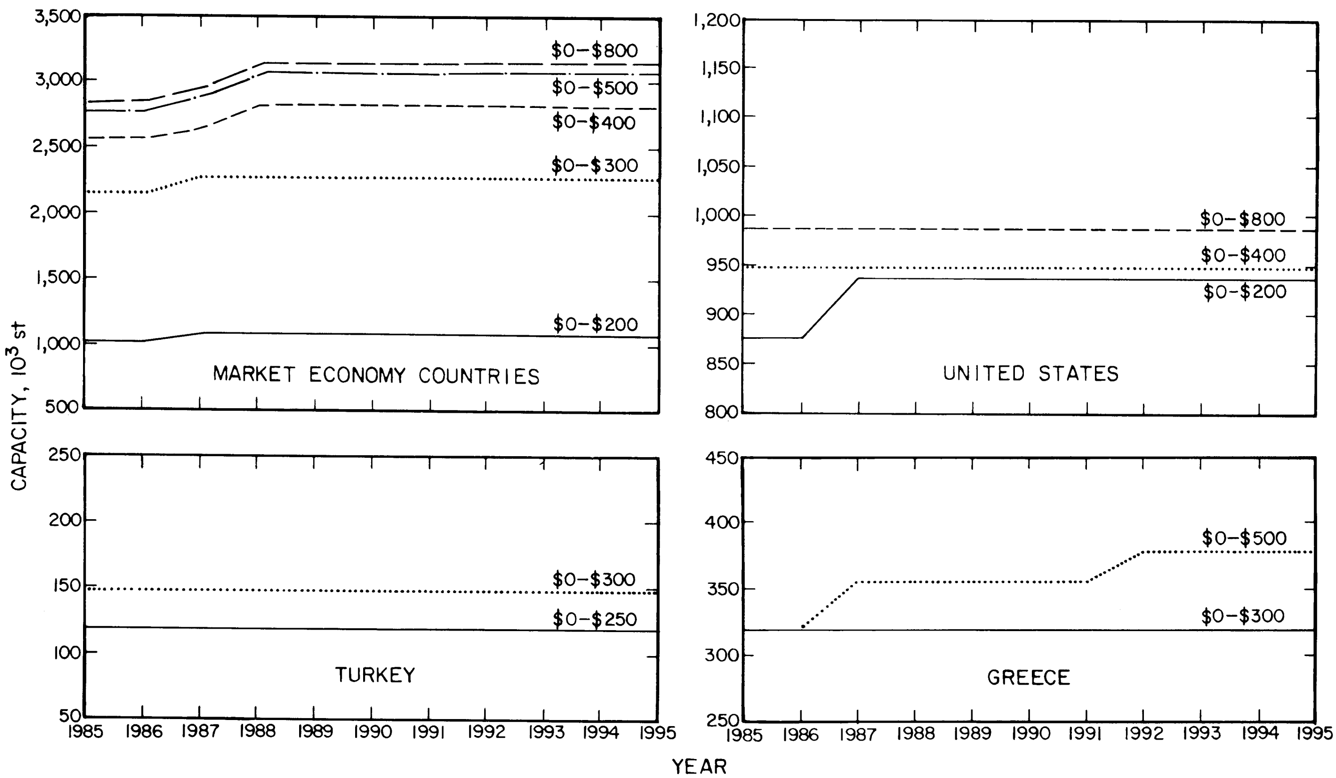

Figure 7.—Potential annual deadburned MgO production from producing mines in MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

CONCLUSIONS

Magnesium is in an unusual position in that it has the capability of being recovered from multiple renewable sources from almost any country in the world. High costs and processing technology complexity have historically restricted industrial production to technically advanced areas, although development in less industrialized areas is beginning to occur.

For this study, magnesium resources have been defined for the next 30 yr. Over this period, properties considered in this study contain an estimated 427.5 million st MgO, of which 68 pct is recoverable as either magnesium metal or MgO compounds utilizing current technology. There are sufficient magnesium reserves currently developed to sustain present MEC production levels at least through the end of this century.

Over 13 million st Mg metal is available from evaluated MEC properties at or below a total cost of $1.34/lb Mg, which was the January 1985 market price of magnesium metal. Approximately 107 million st deadburned MgO and 44 million st caustic calcined MgO were available at or below the January 1985 market prices for these products.

The United States leads the world in producing refined magnesium metal, but it is a relatively minor producer of raw materials for magnesium compound production. Domestic operations have the capability to supply 201,000 st Mg metal and 1.2 million st deadburned MgO annually at the January 1985 market prices of these products, assuming a 15-pct DCFROR. Domestic magnesium resources should continue to be adequate. The total domestic material available over the next 30 yr is well above the projected cumulative domestic needs for these products. As of January 1985, all domestic producers appear to be profitable.

Magnesium metal recovery from brines appears to be the most costly on a per ton basis, while costs to recover MgO appear to be highest from seawater operations. Energy costs account for a significant portion of processing costs; the magnesium industry is currently striving to reduce energy consumption to improve the market position of magnesium. If energy costs remain a prime consideration, imports from countries with lower energy costs could compete with domestically produced magnesium products. U.S. dominance in magnesium markets could well depend on the domestic magnesium industry's ability to reduce energy costs to a competitive level.

CPEC share of world magnesium markets in 1984 amounted to 28 pct for magnesium metal and 62 pct for magnesite. These countries remain relatively unaffected by the recent recession in the Western World, and have continued to increase magnesium production capability at the expense of MEC production. Should current trends continue, the CPEC share of world magnesium markets could be even greater in coming years.

REFERENCES

1. Wilburn, D.R. Magnesium Availability—Market Economy Countries. A Minerals Availability Appraisal. BuMines IC 9112, 1986, 24 pp.

2. Kramer, D.A. Magnesium Metal and Magnesium Compounds. Secs. in BuMines Mineral Commodity Summaries 1986, pp. 94-97.

3. Comstock, H.B. Magnesium and Magnesium Compounds. A Materials Survey. BuMines IC 8201, 1963, 128 pp.

4. Bodenlos, A.J., and T.P. Theyer. Magnesium Refractories. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 379-384.

5. Kramer, D.A. Magnesium. Ch. in Mineral Facts and Problems, 1985 Edition. BuMines 675, 1985, pp. 471-482.

6. Bennett, J.P., and T.A. Clancy. Magnesia Refractories Produced From Chemically Modified Periclase Grains and $Mg(OH)_2$ Slurries. BuMines RI 8848, 1984, 14 pp.

7. Tiwari, B.L., and R.A. Sharma. Electrolytic Removal of Magnesium From Scrap Aluminum. J. Met., v. 36, No. 7, 1984, pp. 41-43.

8. American Mining Congress Journal. Alcoa, MPLC To Build Magnesium Plant. V. 72, No. 9, 1986, p. 14.

9. Chemical Marketing Reporter. Current Prices of Chemicals and Related Materials. V. 224, Jan. 1985.

MANGANESE

INTRODUCTION

Manganese is a vital element in an industrial society, because it is virtually indispensable in enhancing strength, toughness, hardness, and hardenability in iron and steel. In nonferrous metallurgy, manganese is used as an alloying element in aluminum, magnesium, and copper. These metallurgical applications account for about 95 pct of total manganese consumption (*1*).[1] At present, no satisfactory substitute exists to replace manganese in its alloying role. A large share of the remaining 5 pct of consumption is used in the battery and chemical industries.

In 1985, the United States consumed an estimated total of 608,000 mt in manganese metal content (excluding ore containing less than 35 pct Mn) (*2*). Domestic production came from low-grade mines, representing a very insignificant amount relative to consumption. In light of the fact that domestic manganese resources are very uneconomical, the United States will remain dependent on imported manganese to satisfy the industrial demand. The Republic of South Africa, Gabon, Mexico, Brazil, Australia, and India would be the most probable major sources.

There is no standard definition and classification for manganese ores, although industry uses 48 pct Mn content as a standard for international trade pricing. The Bureau, however, has defined manganese ores as those ores containing 35 pct or more manganese. Manganiferous ores are defined as ores containing greater than 5 pct and less than 35 pct Mn. This term has been further subdivided into ferruginous manganese ore for those ores containing 10 to 35 pct Mn and manganiferous iron ores for those ores containing 5 to 10 pct Mn. In industry practice, the term manganiferous iron ores include ores grading as low as 2 pct Mn. The Bureau uses these terms in a broad and practical manner to avoid difficulties in classification purposes. Materials reported as ore may actually be in the form of concentrate, nodules, sinter, or a synthetic manganese ore (*3*). This study takes manganese to the concentrate form delivered to a smelter. All of the analyses are in terms of contained manganese in the concentrate.

Most of the information presented in this report was derived from the Bureau's Information Circular 8978 "Manganese Availability—Market Economy Countries. A Minerals Availability Program Appraisal" (*4*). Additional information on the domestic and foreign manganese industry is available from other publications (*5-7*).

GEOLOGY AND RESOURCES

GEOLOGY

Manganese deposits are classified into three geological types: hydrothermal, residual, and sedimentary (*8-9*). Hydrothermal manganese deposits are normally made up of carbonates and oxides of manganese minerals along with other hydrothermal minerals such as barite, fluorite, and sulfides. Examples of hydrothermal vein-type and replacement deposits include the rhodochrosite ore at Butte and Phillipsburg, MT.

Residual deposits are formed near the surface by weathering processes. Large deposits of economic significance include the Serra do Navio deposits in Brazil, Moanda in Gabon, Nsuta in Ghana, and several occurrences in Australia and India.

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

Sedimentary manganese deposits contain the largest portion of world economic manganese. These deposits are subdivided into the following subclasses:

1. Volcanogenic-related deposits are those in which the manganese can be related directly or indirectly to volcanic sources. The Nsuta Mine in Ghana is considered to be in this class.
2. Nonvolcanogenic sedimentary deposits include those where the manganese is not related to any volcanic source. The more important manganese deposits of this type include Groote Eylandt in Australia, the Morro do Urucum area in Brazil, and the Maharashtra-Madhya Pradesh area in India.
3. Metasedimentary manganese deposits associated with iron formations are identified in Brazil and the Republic of South Africa. The iron formation units are intensive and cover relatively great distances; however, the associated manganese beds within these formations vary in thickness and continuity.
4. Ocean floor nodules cover vast areas in the Pacific, Atlantic, and Indian Oceans. The nodules are found at all depths of the ocean, but higher grades are usually found within the deeper basins at great distances from land areas.

RESOURCES

Total demonstrated resources evaluated for this study amount to 2,191 million mt of manganese ore. Resources classified at the inferred level contribute an additional 668 million mt of ore. Manganese contained in ore and in concentrate is shown in table 1. Table 2 is a list of the market economy country (MEC) properties included in this study.

The Republic of South Africa contains the largest resource of manganese ore evaluated for this study, amounting to 783 million mt or 36 pct of the total. Large resources also exist in Gabon and Australia, included in table 1 under "others."

The world reserve base for manganese (fig. 1) totals 10,885 million mt of manganese ore, of which 8,495 is located in MEC's. In the Republic of South Africa, only a portion of the reserve base was evaluated, owing to limited cost information.

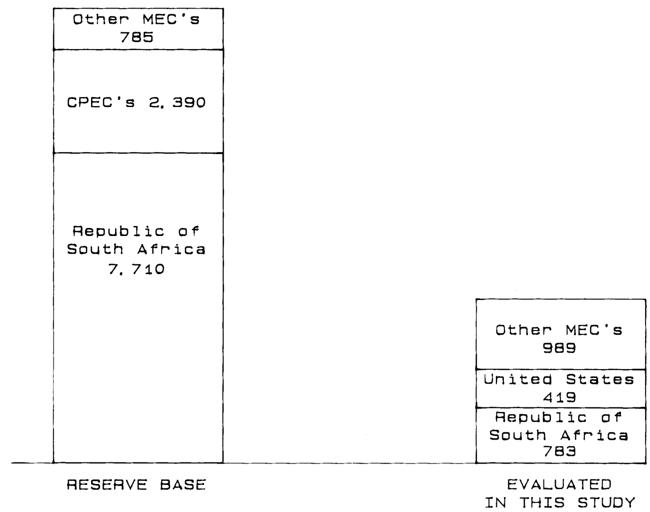

Figure 1.—Estimates of world manganese resources (million metric tons of ore).

Table 1.—Summary of MEC manganese resources evaluated for this study, as of January 1985

(Million metric tons unless otherwise specified)

Country	Demonstrated in situ		Mn in concentrate[1]	Identified in situ		Average grade, pct[1]
	Ore	Contained Mn		Ore	Contained Mn	
Brazil	151	65	41	220	95	43
India	20	9	6	42	18	43
South Africa, Rep. of	783	321	198	818	335	41
United States	419	38	23	533	48	9
Others	818	335	220	1,246	510	41
Total or av	2,191	768	488	2,859	1,006	35

[1] From demonstrated resources only.

Table 2.—MEC manganese properties included in this study

Country and mine or deposit	Owner and/or operator	Deposit status[1]	Type of operation[2]	Av feed grade, pct Mn	Annual product capacity, 10^3 mt
Australia: Groote Eylandt	Broken Hills Proprietary Co.	P	OP	44	2,500
Brazil:					
Azul, Buritirama, Sereno	Companhia Vale do Rio Doce (CVRD).	P[3]	OP	34	1,000
Santana	Companhia Paulista de Ferro Ligas.	P	UG	43	120
Serra do Navio	Industria Commercio e Minerios SA (ICOMI).	P	OP	44	1,200
Urucum	CVRD	P	UG	46	250
Gabon: Moanda	Cie Miniere de l'ogoooue (Comilog).	P	OP	44	2,300
Ghana: Nsuta	Ghana National Manganese Corp.	P	OP	31	300
India:					
Maharastra-Madya-Pradesh Area (includes Balaghat, Kandri, Munsar, Tirodi, Ukwa).	Manganese Ore India, Ltd.	P	OP, UG	42	320
Karnataka (Mysore) Bisgod.	Mysore Minerals Ltd	P	OP	42	45
Keonjhar District	Various owners	P	OP	40	300
Mexico: Tetzintla-Molango	Compania Minera Autlan	P	OP, UG	27	550
South Africa, Republic of:					
Black Rock area (includes Gloria, Nchwaning, Nchwaning West)	Associated Manganese Mines of S. Africa Ltd (AMMOSAL)	P	UG	40	2,000
Mamatwan	S. African Manganese Amcor Ltd. (SAMANCOR).	P	OP	35	2,200
Middleplaats	..do	P[4]	UG	36	1,000
Wessels	..do	P	UG	43	1,125
Lohathla	..do	P[4]	OP	32	500
United States					
Hardshell	Unknown	E	UG	15	100
Maggie	Arizona Manganese Corp.	E	UG	7.5	30
Sunnyside	Standard Metals Inc.	E	UG	10.0	30
Maple Mountain Hovey Mountain.	Various owners	E	OP	8.9	30
North Aroostook District (Dudley and Gelot Hills).	..do	E	OP	9.5	20
Cuyuna Range (southwest portion).	Unknown	E	OP	7.8	50
Butte District (Emma Mine).	The Anaconda Co.	E	UG	17.0	230
Three Kids	Income Investment Inc.	E	OP	13.2	190
Upper Volta: Tambao	Societe Miniere de Tambao	E	OP	54	500

[1] P, producers; E, explored.
[2] UG, underground; OP, open pit.
[3] Property not producing as of study date (January 1, 1985) but did start production in 1985.
[4] Shut down in 1985 but considered as producers for this study.

U.S. AND WORLD HISTORICAL PRODUCTION

Figure 2 shows the history of world manganese ore production in 5-yr intervals from 1950 to 1985 for the major producing countries and regions. The largest production in each year has been from centrally planned economy countries (CPEC's), which include the U.S.S.R., China, Bulgaria, and Hungary. The largest increase in manganese ore production among the MEC's is from the Republic of South Africa, where production rose from 791,000 mt in 1950 to a peak of 5.8 million mt in 1975. Although not shown on curve, there was a significant decrease in production during the late 1970's from the Republic of South Africa. The CPEC production increased from 2.2 million mt to 11.7 million mt over the 1950-85 period.

Total world production of manganese ore in 1985 was estimated at 23.4 million mt, of which 11.7 million mt (50 pct) was from MEC's (*2*). Approximately 85 pct of total CPEC production in 1985 was from the U.S.S.R. Currently, the largest producing region in the MEC's is Africa, which produced an estimated 5.5 million mt of manganese ore in 1985 (*2*). The largest share (62 pct) of the African production is from the Republic of South Africa, which in 1985 produced 3.4 million mt of manganese ore. The other principal producer in Africa is Gabon, which produced an estimated 2.1 million mt of manganese ore in 1985.

Recent significant developments in the U.S. manganese industry include a decrease in production of ferromanganese and a corresponding increase in imports. In addition, nearly all U.S. manganese ferroalloy plants have been acquired by foreign companies, some of which are current ore producers. There has also been a marked decrease in ore imports and ferromanganese production and an increase in ferromanganese imports. Of significance also is the increase in ferromanganese imports from non-ore-producing countries. On the other hand, during the 1973-78 period, when comparative data are available, ferromanganese imports of non-ore-producing countries other than the United States remained relatively stable (*1*). Apparently, the United States has absorbed much of the new ferromanganese production from the ore-producing countries. As a result, the manganese ferroalloy smelting capacity of the United States has significantly decreased to the point that in 1985 only two smelters were operating, one of which was disabled by a flood in November 1985.

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

Manganese deposits are mined by either open pit or underground methods. The physical character of the ore body influences the selection of the specific mining system for each deposit. The continuous character of the ore horizons in the major manganese resource areas permits highly mechanized operations in both open pit and underground mines. Room-and-pillar is the most widely used underground method.

Of the operating areas evaluated in this study, seven are being mined by open pit methods, six by underground methods, and two by combined open pit-underground mining methods. In terms of both production capacity and demonstrated resource, the open pit mines account for about 60 pct of the total quantities. Most mining operations are mechanized, except that operations in India are largely labor intensive. In some cases, a minor amount of mechanization is used, such as a bulldozer or shovel for removing overburden.

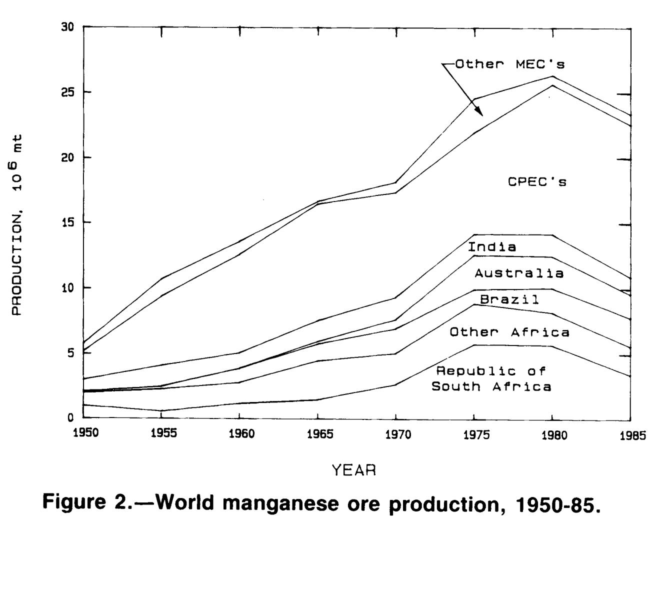

Figure 2.—World manganese ore production, 1950-85.

PROCESSING

Beneficiation

All manganese ores must be crushed and screened, and if the ore contains appreciable quantities of low-grade or barren clayey material, then washing is practiced. The adaptability of a particular beneficiation process depends upon the mineralogical character of the ore. In general, crushing, screening, and washing may upgrade the product by only 1 to 3 pct Mn. Washing can improve the product more if there is a significant clayey material, such as in the Groote Eylandt Mine in Australia. In most cases, however, the fines removed by washing contain relatively high values of manganese (approximately 20 to 30 pct Mn), and thus the recovery of manganese is lowered. Recoveries estimated for the operations in this study range from about 60 to 75 pct. Long-term recoveries may be higher, however, because some of the discarded material may be sold as low-grade blast furnace feed later, depending on demand.

Heavy-media separation can be used where there is a significant quantity of silica and alumina gangue in the ore and in some cases for upgrading chemical- and battery-grade ore. This process is currently in use at the Groote Eylandt Mine (Australia) and the Molango-Tetzintla Mine in Mexico, where silica gangue is encountered.

Pelletizing of manganese fines (about minus 1.5 mm) in conjunction with heavy-media separation and spiraling was done by Serra do Navio Mines of Brazil prior to 1984. The final process was reduction roast to convert hematitic gangue to magnetite. The magnetic fraction was removed, and the remainder of the material was pelletized.

The nodulizing process is normally used in upgrading manganese carbonate ores. The process involves calcining the crushed ore in a rotary kiln to liberate carbon dioxide and to form nodules. Carbon dioxide is first liberated, then further heating softens the ore and forms nodules of about 25-mm diameter. This process is used at the Molango Mine in Mexico. A nodulizing plant was installed at the Nsuta Mine of Ghana to upgrade carbonate ores from 31 to 44 pct Mn (*10*), however, this plant has not yet operated. The advantage of this process is that the manganese content of the carbonate ore can be increased by 12 to 15 pct with little recovery losses.

In all processes, a certain amount of fines are produced that may or may not be further upgraded or shipped direct as sinter feed. All fines (generally minus 6 mm) must be sintered, and this is done at the smelter installation.

Smelting

Smelting of manganese ore for the production of ferromanganese is carried out in either a blast furnace or a submerged arc electric furnace.

Blast furnace smelting can utilize iron blast furnaces converted to smelt ferromanganese. With the exception of the smelter at Boulogne, France, the primary use of the smelter is to supply the steel comlex in which it is located. Excess production can be sold on the open market.

Electric furnaces can be used for producing all forms of manganese ferroalloy and other ferroalloys as well, whereas usually the blast furnace is used to produce high-carbon ferromanganese. Recoveries in the electric furnace are higher when the slags, which may contain 30 to 35 pct Mn, are used to produce silicomanganese, which may be used as is or can be used to produce medium- and low-carbon ferromanganese. The production of silicomanganese in the electric furnace process from these slags is mandatory if the manganese in the slags is to be recovered. The slag is upgraded by the addition of manganese ore and resmelted to produce the silicomanganese.

It is estimated that manganese recoveries to final products in the electric furnace would be about 95 to 96 pct and in the blast furnace about 88 to 90 pct (*1*). If the slag in the electric furnace process were not used for silicomanganese production, electric furnace recoveries would be about the same as those for the blast furnace.

Because of manganese oversupply in recent years, a number of smelters in the ore-importing countries are shut down. These smelters are expected to remain closed even when demand improves because the ore-producing countries will attempt to expand their existing smelting facilities to meet the demand.

PRODUCTION COST

Comparative operating costs to produce manganese ore and concentrates are listed in table 3 and illustrated in figure 3. All costs are in terms of dollars per long ton unit (22.4 lb) of contained manganese in concentrate and therefore represent the effect of manganese grade on the cost elements. Transportation costs that are assigned to particular mines and deposits are based on assumed delivery patterns to areas of smelting.

The table and illustration show the comparative costs on a regional basis. For the producers, Asia incurs the highest mining cost; this is mostly due to inefficient and high-labor-intensive mining among the Indian operations, while Latin America compared to Indian operations enjoys the benefit of higher grade, mechanized operations, and cheap labor. In milling, Africa, because of its high mechanization, enjoys the cheapest milling cost compared to India, where the concentration process is limited to hand sorting. The transportation costs between regions do not vary significantly because distances to smelters are similar. The other costs incurred by the producers (recovery of capital, taxes, and return on investment) only account for 2 to 12 pct of the total production cost because most of these operations have been producing for many years and therefore have very little capital remaining to be recovered.

Among the nonproducers, the high mine and mill operating costs reflect mainly U.S. ores that are low in grade. However, these deposits would recover some byproducts, which would partially offset these high costs. Owing to large initial capital investments required to bring these deposits into production, capital recovery, taxes, and return on investments, are also high.

Table 3.—Manganese production costs for selected MEC's

(All costs are in January 1985 U.S. dollars per long ton unit of manganese contained in concentrate on a weighted-averaged basis)

Region	Operating cost: Mine	Operating cost: Mill	Transportation[1]	Net operating cost	Recovery of capital[2]	0-pct DCFROR: Taxes and royalties[3]	0-pct DCFROR: Total cost[4]	15-pct DCFROR: Taxes and royalties[5]	15-pct DCFROR: Return on investment[6]	15-pct DCFROR: Total cost[7]
Producers:										
Asia[8]	0.99	0.19	0.59	1.77	0.02	0.01	1.80	0.07	0.05	1.91
Latin America[9]	.23	.22	.70	1.15	.03	.02	1.20	.06	.06	1.30
Africa[10]	.25	.07	.69	1.01	.02	0	1.03	.02	.02	1.07
Nonproducers[11]	.44	2.81	.88	[12]3.09	1.53	.15	4.77	1.46	2.94	9.02

[1] Transportation costs to smelters that have been assumed as product destination points for this study.
[2] Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure, and reinvestments required over the life of the operation.
[3] Includes property, State, Federal, and severance taxes, and royalties, where applicable, calculated at a 0-pct DCFROR.
[4] Equal to the sum of net operating costs, taxation, and capital recovery determined at a 0-pct DCFROR.
[5] Includes property, State, Federal, and severance taxes, and royalties, where applicable, calculated at a 15-pct DCFROR.
[6] The revenue increase per ton necessary to obtain a 15-pct DCFROR.
[7] Equal to the sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery *plus* the per ton increase necessary to provide a 15-pct DCFROR after taxation.
[8] Includes 3 mines in India.
[9] Includes 3 mines in Brazil and 1 in Mexico.
[10] Includes 5 mines in the Republic of South Africa and 1 each in Ghana and Gabon.
[11] Includes 8 U.S. deposits and 1 each in Brazil and Upper Volta.
[12] Includes a $1.04 credit from byproduct revenues.

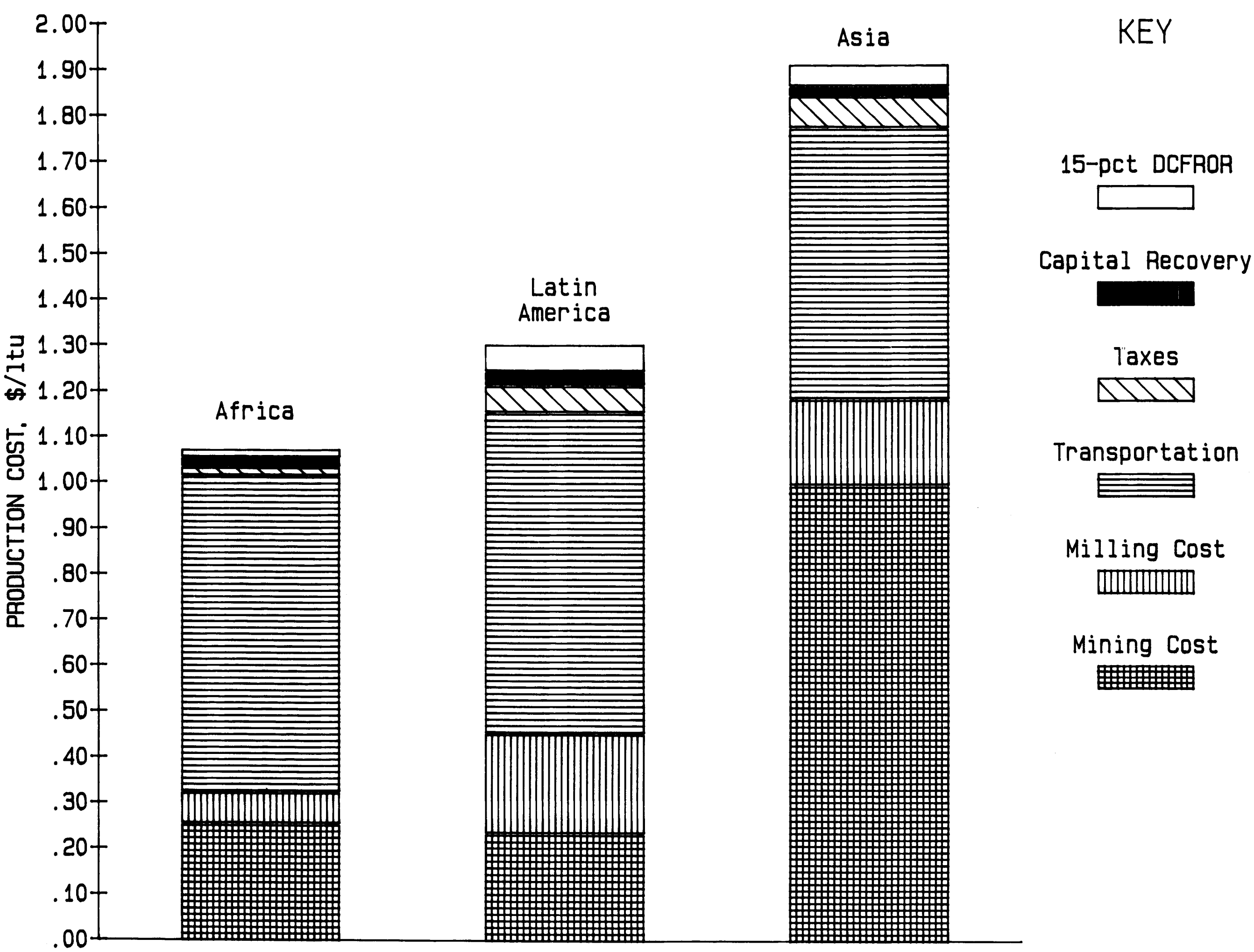

Figure 3.—Manganese production costs for producing mines in selected MEC regions (January 1985 U.S. dollars).

AVAILABILITY

This study analyzed potential manganese availability based upon the demonstrated resources of 25 manganese operations (some mines and mills were combined as districts) in nine MEC's. These deposits contain an in situ quantity of about 2.19 billion mt of ore with a total potential recoverable resource of about 488 million mt Mn in concentrates, 90 pct from producing mines.

TOTAL RECOVERABLE

The potential total availability of manganese for producing and nonproducing operations is illustrated in figure 4. The curves estimate the manganese that is potentially recoverable at certain production costs. At up to $1.43/ltu (the approximate 1985 market price for manganese contained in concentrate), with a 15-pct discounted-cash-flow rate of return (DCFROR), the total potential recoverable manganese from producers is about 353 million mt. At up to $1.85/ltu, total potential recoverable manganese would be about 439 million mt from the producers. As shown, the nonproducing deposits evaluated in this study have the potential to produce only a small quantity of manganese. Most would be economically available only at prices exceeding the January 1985 market level. Not illustrated on the curve is an additional 23 million mt of manganese from the nonproducers, which would have production costs exceeding $2.50/ltu. Most of the 23 million mt is in the United States.

ANNUAL CAPACITY

Potential 1985 production of manganese from producing mines is illustrated in figure 5. The curves are based on 1985 production capacity for producing mines at a 0-pct DCFROR as represented by the upper curve and a corresponding operating cost represented by the lower curve. Because most of the investments at the mines have been depreciated, there is little difference between the curves. As shown, at a cost of $1.43/ltu, the average 1985 market price for manganese, 4.8 million mt Mn in concentrate could have been mined from the producing mines. At a total cost of $1.85/ltu, 6.1 million mt was potentially available from the producers. Estimated 1985 MEC production of manganese in concentrate was approximately 5 million mt (*2*).

The estimated annual production potentially available from 1985 through 1995, at full capacity, from operating mines (including a 15-pct DCFROR) is shown in figure 6. During this period, the curves show only a small decrease in capacity because of the large resources of producing mines. This indicates that current producers can meet MEC requirements through the end of the century without any additional capacity expansion, assuming no significant increase in demand over 1985 levels.

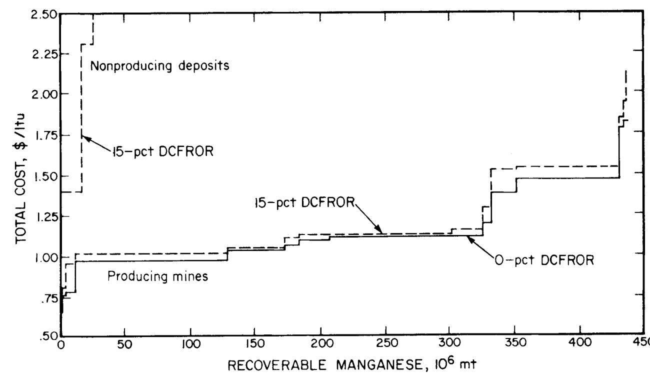

Figure 4.—Potential total manganese available from MEC's (January 1985 U.S. dollars).

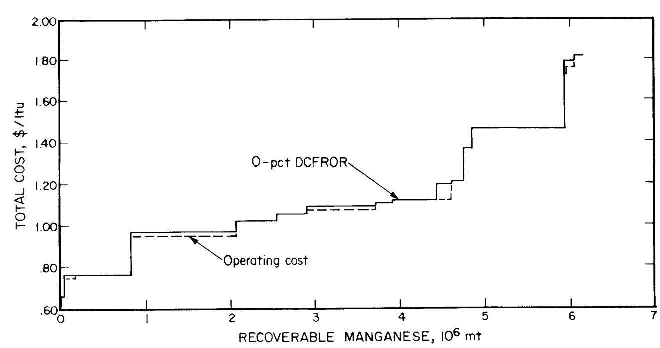

Figure 5.—1985 manganese capacity from producing mines in MEC's (January 1985 U.S. dollars).

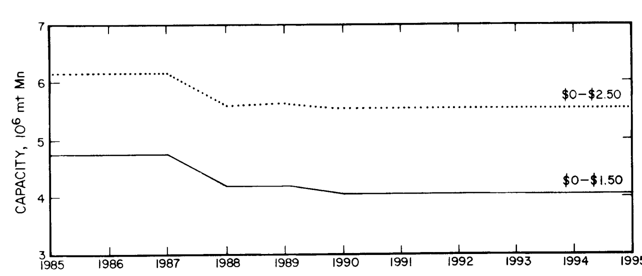

Figure 6.—Potential annual manganese production from producing mines in MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

CONCLUSIONS

The demonstrated in situ resources of the 25 operations or districts analyzed in this study amount to approximately 2.19 billion mt of manganese ore located in nine MEC countries. This resource is estimated to contain about 488 million mt of manganese in concentrate. Over 88 pct of the total is contained in eight mining areas: Groote Eylandt, Australia; Azul Buritirama and Urucum-Corumba areas, Brazil; Moanda, Gabon; and the Black Rock area, Middelplaats, Wessels, and Mamatwan, Republic of South Africa. Based on the 10-yr average MEC production of about 13 million mt of manganese ore, the current demonstrated ore resource would last more than 168 yr.

In recent years, the manganese ferroalloy smelting capacity of the United States has significantly decreased to the point that, in 1985, only two smelters were operating, one of which was disabled by a flood in November 1985. This decrease in capacity has been caused in part by the increase in smelting capacities in ore-producing countries to maximize their profit and employment. Undoubtedly, U.S. smelting capacity will remain small, even if demand increases. Should world manganese demand increase, it is expected the ore-producing countries would attempt to increase their smelting capacity rather than increase ore and concentrate exports.

The United States, without any minable resource above 35 pct Mn, is expected to rely on imported manganese. However, because of the large resources of producing mines, it is expected that the supply of manganese to the United States is well assured.

REFERENCES

1. International Iron and Steel Institute (Brussels, Belgium). Manganese and the Iron and Steel Industry. 1980, pp. 40-41, 59.
2. Jones, T.S. Manganese. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 98-99.
3. ______. Manganese. Ch. in Mineral Facts and Problems, 1985 Edition. BuMines B 675, pp. 483-498.
4. Coffman, J.S., and C.M. Palencia. Manganese Availability—Market Economy Countries. A Minerals Availability Program Appraisal. BuMines IC 8978, 1984, 26 pp.
5. Kilgore, C.C., and P.R. Thomas. Manganese Availability—Domestic. A Minerals Availability System Appraisal. BuMines IC 8889, 1982, 14 pp.
6. Jones, T.S. Manganese. Ch. in BuMines Minerals Yearbook 1985, v. 1, pp. 669-682.
7. DeYoung, J.H., Jr., D.M. Sutphin, and W.F. Cannon. International Strategic Minerals Inventory Summary Report—Manganese. U.S. Geol. Surv. Circ. 930-A, 1984, 22 pp.
8. National Materials Advisory Board, National Research Council—National Academy of Sciences. Manganese Reserves and Resources of the World and Their Industrial Implications. NAS, NMAB-374, 1981, 334 pp.; NTIS PB 82-117615.
9. Dorr, J.V.N. II, M.D. Crittenden, Jr., and R.G. Worl. Manganese. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 385-399.
10. World Mining. Ghana Manganese Will Extend Life by Nodulizing Carbonate Ore Reserve. V. 31, No. 12, 1978, p. 79.

MERCURY

INTRODUCTION

Mercury's unusual combination of physical and chemical properties gives it an industrial and economic importance. It is considered by the Bureau of Mines to be a critical commodity for the United States, owing to its extensive use in a variety of industrial, scientific, and military applications, many of which have few satisfactory substitutes. Significant production is geologically restricted to a limited number of areas, many of which have ceased production in recent years as a result of depressed market conditions.

Mercury, also known as quicksilver, is one of the few metals that is liquid at ordinary temperatures. Important properties that influence its marketability include high density, uniform volume of expansion, high electrical conductivity, ability to alloy readily, high surface tension, chemical stability, and toxicity of its compounds.

Historically, mercury's unique characteristics have enabled it to be used in a wide variety of applications, including electrical apparatus, industrial and control instrumentation, agriculture, pharmaceuticals, paints, pigments, electrolytic preparation of chlorine and caustic soda, and dental supplies. Owing to the toxic nature of mercury vapor and certain compounds, its use has been increasingly restricted in recent years.

The product of most mercury mines is cinnabar ore, which is commonly processed to recover 99.9 pct pure mercury metal (prime virgin). Mercury is sold on the basis of 76-lb flasks. This unit of measure originated in Spain and has been accepted as a worldwide standard because Spain has been the world's leading mercury producer for centuries.

There are over 1,300 known mercury occurrences throughout the world, and many more areas with possible mercury content (*1*).[1] Deposits considered for economic evaluation were limited to those known to have at least 600 flasks contained mercury (15 deposits). The Idria Mine in Yugoslavia was excluded because of the inaccessibility of detailed data. In this study, availability is presented in terms of flasks of pure mercury metal.

Most of the information presented in this chapter is an updated summary of Bureau of Mines Information Circular 9038"Mercury Availability—Market Economy Countries. A Minerals Availability Appraisal" (*2*). Additional information on the domestic and foreign mercury industry is available from other Bureau of Mines publications (*3-5*).

GEOLOGY AND RESOURCES

GEOLOGY

Traces of mercury can be found in most natural substances. Mercury is recovered primarily from the red sulfide mineral, cinnabar (HgS). Native mercury metal, metacinnabar, livingstonite, corderoite, and other mercury minerals are present in some ores, but are rarely present in sufficient quantity to be recoverable. Valuable metals such as gold and silver are generally present in most mercury occurrences in insufficient quantities for economical recovery. It is not uncommon to see mercury produced as a byproduct of gold and/or silver production. Most often, mercury from gold-silver operations is a product of pyrometallurgical process environmental control systems.

Mercury can be found in a wide variety of host rocks. Common host rocks include limestone, calcareous shale, sandstone, serpentine, chert, andesite, basalt, and rhyolite. Significant mercury deposits are found chiefly in regions of extensive Tertiary or Quaternary volcanic and tectonic activity in areas with a high degree of faulting or fracturing. These deposits are classed as epithermal, formed by the deposition of ore minerals from aqueous solutions at relatively low temperatures and shallow depths (*6*). For most deposits, the highest grade of ore is generally found in ore-bearing levels closest to the surface. Ore has been concentrated by replacement, open-space filling, and detrital concentration (*7*).

Mercury ore bodies are commonly small, irregular, and erratic (*6*). Three common forms are (1) distinct veins of very high grade cinnabar (the Almaden district of Spain), (2) disseminated ore occurring in fine-grained or brecciated ore zones (the Monte Amiata district of Italy), or (3) disseminated ore along highly fractured contact zones (the Turkish deposits). Higher grade disseminated ore is usually associated with open-textured rock types such as sandstone or coarse breccia (*8*). Lateral extent of ore zones is highly variable; generally, individual ore zone dimensions do not exceed 100 m. Vein thickness varies from less than 1 m to 21 m.

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

RESOURCES

Demonstrated market economy country (MEC) resources for the 15 deposits evaluated in this study have been estimated at 24 million mt of in situ ore at a weighted-average grade of 0.74 pct Hg. These deposits contain an estimated 5.3 million flasks of mercury, of which 4.4 million flasks, or 83 pct, is recoverable utilizing current technology. Table 1 gives the demonstrated mercury resources as of January 1985 for evaluated MEC's. Because of the sensitivity of mercury resource information in some countries, detailed information for individual deposits could not be reported. Table 2 lists the properties evaluated for this study.

Table 1.—Summary of MEC demonstrated mercury resources evaluated for this study, as of January 1985

Country	Number of deposits	In situ: Hg grade, pct	In situ: Ore, 10^3 mt	Hg, 10^3 flasks: Contained	Hg, 10^3 flasks: Recoverable
Algeria	2	1.00	300	90	80
Italy	2	.56	12,300	2,190	1,750
Turkey	4	.33	2,000	190	150
United States	4	.26	2,100	170	140
Other[1]	3	1.27	7,400	2,700	2,320
Total or av	15	.74	24,100	5,340	4,440

[1] Includes Canada, Philippines, and Spain. Owing to proprietary considerations, these countries could not be treated separately.

Table 2.—MEC mercury properties included in this study

Deposit and location	Ownership	Current status[1]	Mining type[2]
Algeria:			
Ismail	SONAREM (Govt. owned)	P	S
M'Rasma	..do	P	S
Canada: Pinchi Lake	COMINCO, Ltd	N	S,U
Italy:			
Abbadia S. Salvatore	SAMIM (Govt. owned)	N	U
Selvena	..do	N	U
Philippines: Palawan Quicksilver	Palawan Quicksilver Mines, Inc.	N	S
Spain: Almaden	Arrayanes, S.A. (Govt. owned).	P	S,U
Turkey:			
Halikoy	Etibank (Govt. owned)	P[3]	U
Karaburun-Izmir	Underdetermined[4]	N	S
Karareis	..do[4]	N	U
Konya Area	Etibank (Govt. owned)	P[3]	U
United States:			
B and B Mine	Private individual	N	S
Gibraltar	Undetermined[4]	N	U
McDermitt	Placer U.S., Inc	P	S
Study Butte	Sanger Investment Co.	N	U

[1] N, not producing as of January 1985; P, producing as of January 1985.
[2] S, surface; U, underground. For deposits not producing, mining type is proposed based on past history, geology, and technology.
[3] Operations producing at limited rate for internal use only.
[4] Ownership undetermined; property has either been abandoned or is involved in litigation.

Mercury is currently being recovered in a limited number of areas around the world. Over half of the total recoverable mercury at the demonstrated level in this study is derived from six producing deposits. Much of the mercury currently being recovered is produced from the Almaden District in Spain. In 1985, it is estimated that Spain produced 44 pct of the total mercury for MEC's or 23 pct of the world's total mercury (*5*).

Demonstrated resources from nonproducing deposits make up over 40 pct of the total recoverable resource available from MEC's. Nonproducing Italian deposits of the Monte Amiata District are the source for nearly 40 pct of the total recoverable mercury from MEC's. These deposits have the potential for supplying nearly 90 pct of the recoverable mercury from MEC's not currently producing. Additional demonstrated resources from nonproducing deposits are available from Canada, the Philippines, Turkey, and the United States.

Mercury production from centrally planned economy countries (CPEC's) has become increasingly influential on world markets, particularly from mines in the U.S.S.R., China, and Czechoslovakia. These resources are not included in the economic evaluations of this study because of the lack of detailed engineering, resource, and cost data.

Figure 1 illustrates the world mercury resources included in this study as well as the reserve base and identified resource values. The reserve base of 7.2 million flasks of recoverable mercury includes essentially the same resources as does this study, with the addition of reserves in Mexico, Yugoslavia, China, and the U.S.S.R. (*3*). At the identified level, resources are estimated to be 17 million flasks of recoverable mercury (*5*).

Not listed on table 1, or included in figure 1, is approximately 2 million flasks of recoverable mercury at the inferred resource level, which are additional resources from deposits included in this study.

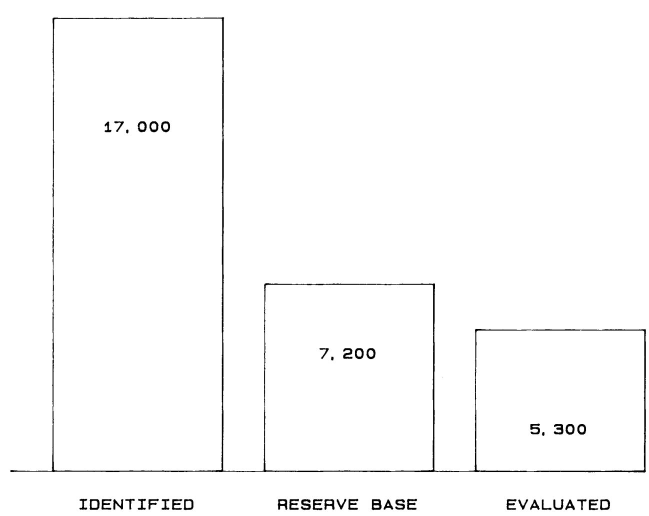

Figure 1.—Estimates of world mercury resources (thousand flasks of contained mercury).

U.S. AND WORLD HISTORICAL PRODUCTION

Mercury production in 1985 was estimated to be 196,250 flasks from 11 countries; only 55 pct from MEC's. Two countries, the U.S.S.R. (34 pct) and Spain (23 pct), accounted for over half of the world's 1985 mercury production. Figure 2 graphically illustrates world mercury production at 5-yr intervals for the 1950-85 period. Since 1950, when Spain and Italy dominated world mercury production, the market share has been lost primarily to CPEC's, particularly the U.S.S.R. and to a lesser extent China. The late 1960's and early 1970's saw world mercury production at all time high levels. During this period, several countries including Mexico, Canada, and Yugoslavia (included in "other MEC" on figure 2) produced at record high levels. In recent years, the world mercury market has seen the decline of Italian production and the emergence of production from Algeria (included in "other MEC"). The U.S.S.R. has now become the world's leader, accounting for one-third of production. Over the past 35 yr, Spain and the United States have maintained a fairly consistent share of world mercury production; Spain averaging 20 pct of market share and the United States 10 pct.

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

Mercury ore is mined by both surface and underground methods. The mode of occurrence of the mercury deposit determines the mining method. Table 2 lists the mining methods employed at the mercury mines included in this study. The bulk of mercury ore is currently recovered by conventional open-pit mining methods. The ore and overburden are separately drilled, blasted, loaded, and hauled. Broken ore is loaded by shovels and front-end loaders and is hauled by truck and rail to the beneficiation plant or to an on-site crusher.

Of the underground mining methods currently being utilized by the mercury industry, narrow vein mining methods predominate, with cut-and-fill mining the most commonly used. Vertical crater retreat (VCR) mining, a relatively new mining method that has proven much more economical than cut-and-fill, is gradually becoming the method used by industry.

All of the underground mercury mines evaluated in this study utilize conventional drilling, blasting, loading, and hauling. Drilling is done by jumbo drills in the larger mines and by hand-held jackleg drills in the smaller mines. Broken ore is loaded by mucking machines, slushers, or manually, and hauled out of the mine by various means.

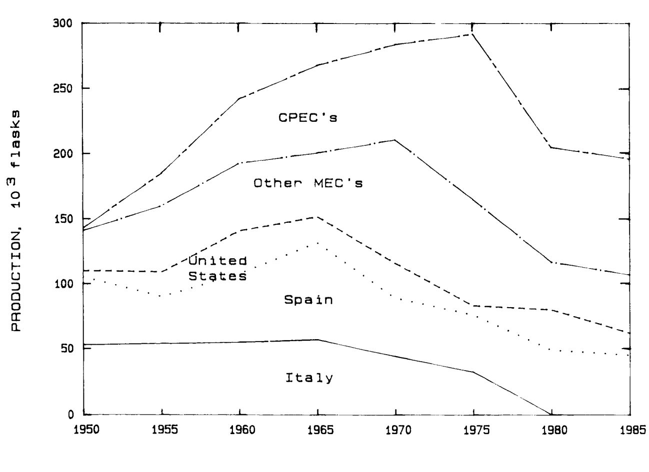

Figure 2.—World mercury production, 1950-85.

The Antiqua Mine, part of the Almaden operations of Spain, is presently the largest underground mine in the MEC's. The VCR mining method is replacing cut-and-fill methods at this operation.

PROCESSING

Typically, beneficiation of mercury ore begins with crushing and screening. Mercury-bearing ore crushes more readily than barren rock; therefore, a crude separation of ore and barren rock is conducted by screening. Primary crushing is done by jaw crushers; screening, by grizzlies. In recent years, this first step in the beneficiation process has moved from the beneficiation facility to the mine site. Efficient mercury beneficiation recovery can exceed 95 pct.

In some operations, mercury ore is further upgraded before roasting. In this process, ore is further reduced in size by use of rod mills, ball mills, or semiautogeneous mills. Ore is then concentrated by flotation, jigging, and tabling. Flotation has proven to be the most successful concentration method, producing concentrates from 25 to 75 pct Hg and recovering almost 90 pct of the contained mercury. This upgrading process has resulted in improved efficiency and considerable energy savings in the subsequent roasting step.

Roasting is essentially a distillation-oxidizing process and consists of heating of the concentrate, forming mercury vapor and SO_2 gas, followed by condensation of the mercury vapor. Either mechanical furnaces or retorts are used in roasting. Mechanical furnaces include both multiple hearth furnaces and rotary furnaces. Both types of furnaces exhibit advantages and disadvantages; the multiple hearth is more capital intensive than the rotary but much more energy efficient.

Gases from furnacing are passed through dust collectors. Dust is collected and further processed and the dust-free gas is directed to the condenser system, where mercury vapor is collected.

Retorts are also used in roasting mercury ore. They are relatively inexpensive compared to the mechanized roasters but exhibit several disadvantages: mercury ore must be batch loaded, and manually charged and discharged. In addition, retorts are relatively small in capacity.

PRODUCTION COSTS

A summary of estimated operating costs for producing and nonproducing mercury mines and deposits is given in table 3 and illustrated on figure 3 (in terms of cost per recoverable flask). Individual deposit costs are withheld to preserve proprietary data.

The table and illustration show that the mining costs for the nonproducers are higher than those for the producers. This is primarily a result of the producers being weighted very heavily towards the large-scale operations at Almaden, where a high degree of economy of scale exists, while the nonproducers primarily represent the Italian deposits where ore would be recovered by expensive underground methods.

The milling costs exhibit a lesser degree of variability because, generally, beneficiation techniques are similar for all operations considered. Although not shown, a small cost to transport the mercury to a local market is included in the mill cost. The table and figure also show that capital recovery costs and the return on investment is greater for the nonproducers. This is because most of the investments for the producers are already depreciated, and require less cost increase to attain the stipulated 15-pct discounted-cash-flow rate of return (DCFROR).

Table 3.—Mercury production costs for producing and nonproducing operations in MEC's

(All costs are in January 1985 U.S. dollars per flask of recoverable mercury on a weighted-averge basis)

	Operating cost		Net operating cost	Recovery of capital[1]	0-pct DCFROR		15-pct DCFROR		
	Mine	Mill			Taxes and royalties[2]	Total cost[3]	Taxes and royalties[4]	Return on investment[5]	Total cost[6]
Producers	43	73	116	15	2	133	3	16	150
Nonproducers	405	109	514	23	2	539	28	31	596

[1] Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure, as of January 1985, and reinvestments required over the remaining life of the operation.
[2] Includes property, State, Federal, and severance taxes, and royalties, where applicable, calculated at a 0-pct DCFROR.
[3] Equal to the sum of net operating costs, capital recovery, and taxation determined at a 0-pct DCFROR.
[4] Includes property, State, Federal, and severance taxes, and royalties, where applicable, calculated at a 15-pct DCFROR.
[5] The revenue increase per flask necessary to obtain a 15-pct DCFROR.
[6] Equal to the sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery *plus* the per flask increase necessary to provide a 15-pct DCFROR after taxation.

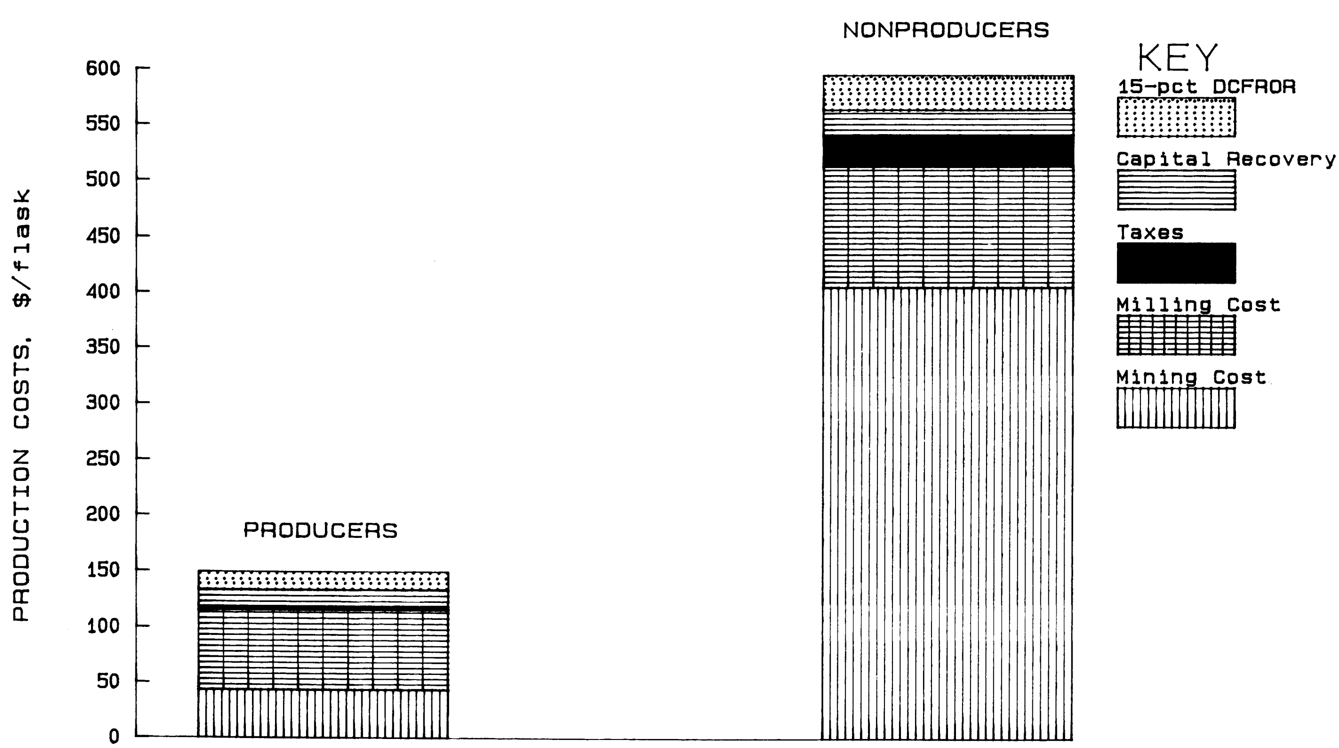

Figure 3.—Mercury production costs for producing and nonproducing operations in MEC's (January 1985 U.S. dollars).

AVAILABILITY

The following availability curves and table show the potentially recoverable mercury available from MEC's. Of the 15 properties analyzed in this study, two properties containing approximately 12,000 flasks of recoverable mercury have been excluded because of unusually high costs of production. Curves for this section have been limited to a total curve (which includes all studied properties) and an annual curve for the producers. Curves comparing total availability from the producers versus the nonproducers have been excluded because of the limited number of properties, therefore preserving proprietary data.

TOTAL RECOVERABLE

Table 4 and figure 4 depict the total availability of mercury from evaluated deposits in MEC's. The 1985 market price per flask for prime virgin mercury was approximately $300 (New York price was $315 and London $295) (*5*).

At a total production cost equivalent to $300/flask, approximately 2.4 million flasks of mercury is available, all from currently producing deposits. The total recoverable mercury from all evaluated properties, 4.4 million flasks, is available at costs less than $1,200/flask.

Table 4.—Potential total mercury available from producing and nonproducing MEC operations

Status	Cost range per flask[1]	Cumulative total recoverable Hg, 10^3 flasks
Producers	$0- $300	2,329
	0- 650	2,482
Nonproducers	0- 450	53
	0- 600	1,882
	0- 1,100	1,947
Total	0- 300	2,329
	0- 600	4,345
	0- 1,100	4,429

[1] Total cost including a 15-pct DCFROR; range includes all recoverable mercury up to the cost shown.

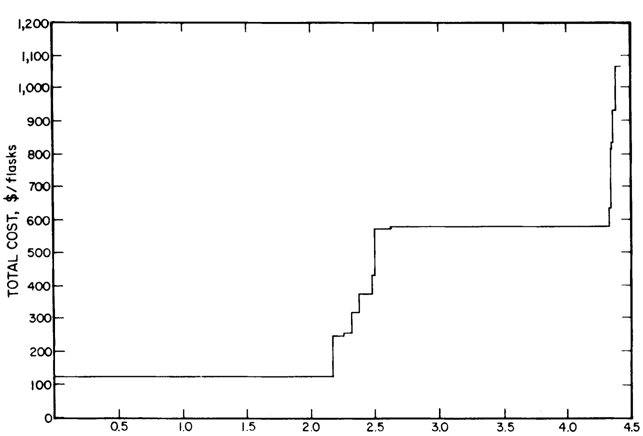

Figure 4.—Potential total mercury available from MEC's (January 1985 U.S. dollars).

Table 4 also indicates that a total of 2.5 million flasks of mercury is recoverable from producing properties at production costs ranging up to $650/flask, and 1.9 million flasks of mercury from currently nonproducing properties at production costs ranging up to $1,200/flask.

ANNUAL CAPACITY

Figure 5 shows potential annual mercury production from producing deposits from 1985 to 1995. At a production cost of $650/flask, approximately 126,000 flasks of mercury could potentially be produced annually from 1985 to 1988. This compares with approximately 100,000 flasks of mercury actually produced from MEC's in 1985 (*5*). By 1992, annual production could decrease to 85,000 flasks of mercury as resources from producing mines are depleted. This production decrease may be offset if resources currently considered inferred are further explored and proven, thereby increasing the demonstrated reserve base. From 1992 through 1995, annual mercury availability from producing MEC mines is projected to remain stable at 85,000 flasks.

Similarly, approximately 114,000 flasks of mercury is potentially available between 1985 and 1988 at a production cost of less than $300/flask. This is sufficient to meet the 1985 production rate of approximately 100,000 flasks from MEC's. Annual availability could decrease to 80,000 flasks by 1990 and remain at this level until at least 1995. Much of this decrease could be offset by increased sales from stockpiles or use of mercury from secondary sources.

In order to meet anticipated future demand, additional mercury production is required to supplement those properties currently producing. Assuming a demand of 241,000 flasks in 1990 (*3*), producing operations could potentially supply 37 pct of this demand at costs up to $450/flask; deposits not currently producing could supply an additional 28 pct at costs up to $1,000/flask. The remaining one-third would have to be supplied from either CPEC's, from Government or industry stockpiles, or from recycled mercury.

Actual annual availability could vary from anticipated levels as market conditions change. Factors that would affect such change include varied production of existing mines, nonproducing deposits coming into production, discovery of additional deposits, and reclassification of inferred resources as demonstrated.

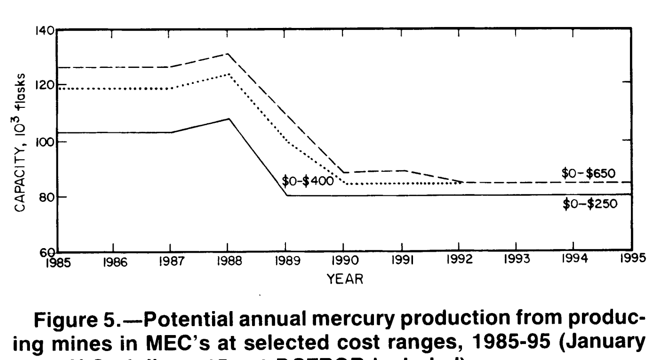

Figure 5.—Potential annual mercury production from producing mines in MEC's at selected cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

CONCLUSIONS

There has been little change in mercury mining and processing technology in recent years. However, significant changes have occurred in mercury use patterns where technological changes in mercury battery design, more efficient chlor-alkali mercury cells, and improvements in the substitute diaphragm and membrane cells, have begun to reduce primary mercury consumption.

Environmental concerns have provided impetus for more efficient recycling practices. Recycled mercury will undoubtedly make up a significant portion of mercury supply in the future.

The mercury industry appears to be stabilizing after a period of low demand and depressed market price brought about in part by increased environmental concerns and use of substitutes. Although prime virgin mercury consumption appears to be decreasing slightly, there are few indications that producers are curtailing output. Excess production from principal MEC's is being stockpiled; most production from CPEC's is being consumed internally. Production is reduced or curtailed for those operations that repeatedly incur costs above \$300/flask; a market price well above this level would encourage increased mercury production.

Demonstrated in situ resources of mercury from MEC's amount to approximately 24 million tons of ore from which 4.4 million flasks of mercury is potentially recoverable utilizing present technology. Mercury from producing properties operating at costs less than \$650/flask would be sufficient to meet market economy production needs at current production rates until 1988. Mercury availability could be greatly increased as known mercury occurrences, with identified tonnages containing over 17 million flasks, are further explored and resources upgraded to the demonstrated level.

Based upon forecast demand levels, mercury from the producing mines in MEC's, recoverable at a total production cost of \$650/flask, would supply only 37 pct of total world demand in 1990. To meet demand requirements, additional properties would need to come on line to supplement producers at higher mercury costs or a greater percentage of mercury could be purchased from CPEC's. Mercury production from these countries has increased in recent years. These demand projections could change significantly, however, if the projected growth rate does not come about because of a decrease in mercury demand, or should new technologies emerge that require greater mercury consumption.

REFERENCES

1. Pennington, J.W. Mercury. A Materials Survey. BuMines IC 7941, 1959, 92 pp.

2. Mishra, C.P., D.R. Wilburn, D.G. Hartos, C.D. Sheng-Fogg, and R.C. Bowyer. Mercury Availability—Market Economy Countries. A Minerals Availability Appraisal. BuMines IC 9038, 1985, 18 pp.

3. Carrico, L.C. Mercury. Ch. in Mineral Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 499-508.

4. ———. Mercury. Ch. in BuMines Minerals Yearbook 1985, v. 1, pp. 683-689.

5. ———. Mercury. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 100-101.

6. Davis, F.F., and E.H. Bailey. Mercury. Ch. in Mineral and Water Resources of California. CA Div. Mines and Geol. Bull. 191, 1966, pp. 247-254.

7. Bailey, E.H., A.L. Clark, and R.M. Smith. Mercury. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 401-414.

8. Drake, H.J. Mercury. Ch. in Mineral Facts and Problems, 1980 Edition. BuMines B 671, 1981, pp. 563-574.

MOLYBDENUM

INTRODUCTION

Molybdenum is a metallic element whose versatility has assured it a significant role in contemporary technology and industry. It is used principally as an alloying agent in steels, cast irons, and superalloys, either alone or in combination with other alloy metals. It also finds significant use as a refractory metal and in numerous chemical applications.

Basically, nonintegrated mine operations (mostly small and byproduct producers) sell molybdenum in concentrate form to independent companies for roasting. For this reason, all prices and costs in this chapter are quoted on the basis of contained molybdenum in concentrate, f.o.b. mill, excluding costs of transportation and conversion to marketable products. Although product prices are also based on contained molybdenum, they reflect, at least partially, the cost of conversion and vary slightly depending on purity, type of packaging, and whether briquetted.

Availability in this report is reported in terms of pounds of contained molybdenum in concentrate. The major portion of concentrate production is roasted to technical-grade molybdenum trioxide (MoO_3) the intermediate marketable material from which almost all molybdenum products are made; a small portion of the concentrate is used directly in mineral form (MoS_2). Any technical-grade oxide not directly used in the industry is further processed into other marketable forms: high-purity molybdenum trioxide and ferromolybdenum. The high-purity molybdenum trioxide in turn is converted either into chemical form (as pure salts), metal form, or alloy form (as master alloys for high-temperature metals).

Most of the information presented in this chapter is an updated summary of Bureau of Mines Information Circular 9044 "Molybdenum Availability—Market Economy Countries. A Minerals Availability Appraisal" (*1*).[1] Additional information on the domestic and foreign molybdenum industry can be found in other Bureau publications (*2-5*).

GEOLOGY AND RESOURCES

GEOLOGY

Molybdenum has not been found free or as a native metal in a natural environment. Even as a compound it is relatively scarce, comprising only about 1.0 to 1.5 ppm concentration in the continental crust. This concentration is evenly distributed throughout igneous rocks, though higher concentrations have been noted in basaltic rocks.

Molybdenite (MoS_2) is the only molybdenum mineral of commercial importance. The mineral is lead-gray with a metallic sheen and characteristically occurs in thin, tabular, hexagonal plates and is disseminated in fine specks. The plates have an eminent basal cleavage, and are soft and flexible but not elastic. Molybdenite is commonly mistaken for graphite because of its similarity in structure and softness. It frequently occurs in pneumatolytic contact deposits associated with cassiterite, scheelite, wolframite, fluorite, etc.

The following discussion is largely taken from the molybdenum chapter in U.S. Geological Survey Professional Paper 820 (*6*). Molybdenum deposits are classified into five genetic types: (1) primary porphyry or disseminated deposits, including stockworks and breccia pipes in which metallic sulfides are dispersed through relatively large volumes of altered and fractured rocks; (2) contact metamorphic zones and tactite bodies of silicated limestone adjacent to intrusive granitic rocks; (3) quartz veins; (4) pegmatites and aplite dikes; and (5) bedded deposits in sedimentary rocks. The first three genetic types are determined to be hydrothermal in origin. As such, nearly all resources in the world fall under these types. The remaining two types currently do not possess any economic importance.

Primary porphyry deposits, the major source of molybdenite, occur as small disseminated grains in veins and veinlets of hydrothermally altered igneous rocks. The mineralization normally is associated with intrusions along fault intersections. Generally, the intrusive rocks are siliceous with composition ranging from granodiorite to granite. The average mineral concentration in the resources identified for this study ranges from 0.1 to 0.5 pct MoS_2.

Copper-molybdenum porphyry deposits are somewhat similar in origin to that of primary molybdenum porphyry deposits. The deposits are related to silicic intrusions and hydrothermal alteration. Molybdenite concentrations are much lower, ranging from 0.015 to 0.1 pct MoS_2, and therefore can only be economically recovered as a byproduct.

In contact-metamorphic and tactite deposits, molybdenite is widely distributed along the contact between granitic intrusives and lime-rich sedimentary rocks. In many places, the size, shape, and occurrence of the ore bodies indicate that the mineralization was controlled by pre-ore fracturing of the lime-rich host rock. The molybdenite is usually associated with scheelite, bismuthite, and

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

copper sulfides. The deposits are generally small, but contain relatively high molybdenite concentration, up to 0.6 pct MoS_2.

Molybdenite also occurs in fissure or quartz vein deposits. The vein filling is mainly quartz and molybdenite with minor amounts of pyrite, chalcopyrite, and chlorite and varies in size from a fraction of an inch to several feet. The mineral concentration also varies widely, and most of the molybdenite is fairly fine grained and appears to be concentrated along vein walls or joints and seams.

RESOURCES

Most molybdenum resources occur in the western mountain regions of North and South America, extending from Alaska and British Columbia through the United States and Central America to the Andes Mountains of Chile. These regions, roughly bordering the western margin of the Pacific Basin, are also regarded as most promising for future discoveries.

Notably, the deposits that are presently mined or anticipated to be mined, primarily for molybdenum, occur only in Canada, the United States, and Mexico. Those three countries also contain large deposits where molybdenum is produced as a byproduct, chiefly of copper mining. All other world molybdenum resources are of the byproduct type, most importantly the deposits in Chile and Peru.

The world molybdenum reserve base, including primary and byproduct sources, is estimated to be 26 billion lb (*4*). Approximately 17.1 billion lb is recoverable from the demonstrated resources of the deposits evaluated for this report (table 1). (The reserve base is in terms of contained molybdenum in ore, whereas the tonnage evaluated in this study is in terms of molybdenum contained in concentrate net of recovery losses.) It was not possible to evaluate centrally planned economy country (CPEC) properties in this study owing to a lack of cost information.

Table 1.—Summary of world molybdenum resources evaluated for this study, as of January 1985

Country	In situ ore tonnage, 10^6 mt	In situ grade, wt pct	Recoverable, 10^6 lb
Canada	NM	NM	2,047
Chile	13,065	0.0228	3,897
Mexico	NM	NM	676
Peru	2,868	.0176	635
United States	NM	NM	9,064
Other countries	NM	NM	785
Total	NAp	NAp	17,104

NAp Not applicable.
NM Not meaningful. In situ tonnage and grade not totaled or averaged because it would combine primary and byproduct molybdenum properties or different countries.

Figure 1 graphically displays estimates for identified resources, the reserve base, and the resources evaluated in this study. The United States accounts for 50 pct of the analyzed demonstrated resources (9.1 billion lb), followed by Chile with 23 pct (3.9 billion lb). Canada, Peru, and Mexico have smaller but nonetheless important resources. The other analyzed deposits are scattered over the Americas, particularly in Argentina and Panama, southern Asia, and islands of the southwestern Pacific Ocean. Table 2 is a list of the properties evaluated.

Classified by type of deposit, properties primarily producing molybdenum contain 55 pct of the analyzed resources (fig. 2). Only 29 pct of those properties are currently operating. Properties producing molybdenum as a byproduct contain the remaining 45 pct of the analyzed resources, and 67 pct of those are currently operating. This sharp difference in operational status between primary and byproduct properties is a crucial factor in the industry, which will be discussed later.

An additional 20 billion lb of molybdenum is potentially recoverable at the inferred level (*4*). Additional hypothetical, speculative, and undiscovered molybdenum resources are available, although quantities are not known.

Table 2.—MEC molybdenum properties included in this study

Country and deposit name	Owner	Type[1]	Current status[2]	Mining method[3]
Argentina:				
Bajo la Alumbrera	Yacimientos Agua del Dionisio.	B	E	OP
El Pachon	Compania Minera Aguilar SA.	B	E	OP
Paramillo Sur	Fabricaciones Militares O/OP.	B	E	OP
Canada:				
Adanac	Adanac Mining & Exploration.	P	E	OP
Ajax	Newmont Mining Corp.	P	E	BC
Berg	Kennco Explorations Ltd.	B	E	OP
Bethlehem	Bethlehem Copper Ltd.	B	T	OP
Boss Mountain	Noranda Mines, Ltd.	P	T	SU
Brenda	Brenda Mines, Noranda Mines, Ltd.	B	P	OP
Casino	Casino Mining Co.	B	E	OP
Copper Mountain, Needle Mountain.	Mines Gaspe, Noranda Mines Ltd.	B	T	SU
Endako	Placer Development Ltd.	P	T	OP
Gibraltar	Gibraltar Mines Ltd.	B	P	OP
Glacier Gulch	Amax, Inc.	P	E	OS
Highmont	Highmont Mining Corp.	B	T	OP
Huckleberry Mountain.	Kennco Explorations Ltd.	B	E	OP
Island Copper	Utah Mines Ltd.	B	P	OP
JA Zone	Bethlehem Copper Ltd.	B	E	BC
Kitsault	Amax, Inc.	P	T	OP
Lornex	Lornex Mining Corp. Ltd.	B	P	OP
Maggie	Bethlehem Copper Ltd.	B	E	OP
Mount Thomlinson	Kidd Creek Mines Ltd.	P	E	OS
Poison Mountain	Long Lac Copper Giant	B	E	OP
Red Bird	Phelps Dodge Corp.	P	E	OP
Schaft Creek	Teck Corp., Liard Copper Mines.	B	E	OP
Trout Lake	Newmont Mining Corp.	P	E	OS
Valley Copper	Valley Copper Mines Ltd.	B	P	OP
Chile:				
Andacollo	Noranda Mines Ltd.	B	E	OP
Andina	Codelco	B	P	BC
Cerro Colorado	Cerro Colorado Mine Development Corp.	B	E	OS
Chuquicamata	Codelco	B	P	OP
El Abra	..do	B	E	OP
El Salvador	..do	B	P	BC
El Teniente	..do	B	P	BC
La Escondida	Utah International, Getty Oil.	B	E	OP
Los Bronces	Exxon Minerals Co.	B	P	OP
Los Pelambres	Anaconda Co.	B	E	OP
Quebrada Blanca	Chilean Government Compania Exploracion Dona Ines.	B	E	OP
Fiji: Namosi	Viti Copper Ltd.	B	E	OP
Iran: Sar Cheshmeh	National Iranian Copper.	B	P	OP

See explanatory notes at end of table.

Table 2.—MEC molybdenum properties included in this study—Continued

Country and deposit name	Owner	Type[1]	Current status[2]	Mining method[3]
Mexico:				
Cumobabi	Frisco SA de CV	P	P	OP
La Caridad	Mexicana del Cobre	B	P	OP
Opodepe	Minera Opodepe S de RL de CV.	P	E	OP
Pakistan: Saindak	Resource Development Corp.	B	E	OP
Panama: Cerro Colorado.	Empresa de Cobre Cerro Colorado.	B	E	OP
Papua New Guinea:				
OK Tedi	Dampier, My Fugilan, Kupfer.	B	P	OP
Yandera	Trialco, Buka, Broken Hill.	B	E	OP
Peru:				
Antamina	Minero Peru	B	E	OP
Cerro Verde, Santa Rosa.	..do	B	P	OP
Cuajone	Southern Peru Copper Corp.	B	P	OP
El Aguila	Empresa Minera El Aguila.	B	P	OP
Michiquillay	Minero Peru, Michiquillay Copper Co.	B	E	OP
Quellaveco	Minero Peru	B	E	OP
Toquepala	Southern Peru Copper Corp.	B	P	OP
Toromocho	Centromin	B	E	OP
Philippines:				
Basay	Southern Star Mining & Industrial Corp.	B	T	SU
Copper Shield, Kennon.	Various claim holders.	B	T	BC
Sipalay	Marindugue Mining & Industrial Corp.	B	T	OP
United States:				
Alaska:				
Orange Hill, Bond Creek.	Bear Creek Mining Co.	B	E	OP
Quartz Hill	Pacific Coast Molybdenum.	P	D	OP
Arizona:				
Copper Basin	Phelps Dodge Corp.	B	E	OP
Cyprus Bagdad	Cyprus Mining Corp., Amoco.	B	P	OP
Miami East	Newmont Mining Corp.	B	D	FS
Mineral Park	Duval Corp.	P	T	OP
Mission, San Xavier	Asarco	B	T	OP
Morenci, Metcalf	Phelps Dodge Corp.	B	P	OP
New Cornelia	..do	B	T	OP
Palo Verde	Anamax	B	T	OP
Pinto Valley	Newmont Mining Corp.	B	P	OP
Ray	Kennecott Mining Co.	B	P	OP
San Manuel	Newmont Mining Corp.	B	P	BC
Sierrita, Esperanza	Duval Corp.	B	P	OP
Twin Buttes	Anamax	B	T	OP
Vekol Hills	Newmont Mining Corp.	B	E	OP
Colorado:				
Climax	Amax, Inc.	P	P	SU
Henderson	..do	P	P	BC
Mount Emmons	U.S. Energy, Santa Fe Mining.	P	E	BC
Idaho:				
Thompson Creek	Cyprus Mines Corp.	P	P	OP
White Cloud	Asarco	P	E	OP
Montana:				
Big Ben	Amax, Inc.	P	E	OP
Heddleston	Asarco	B	E	OP
Nevada:				
B & C Springs	Sharon Steel Co.	P	E	OP
Buckingham	Amax, Inc.	P	E	OP
Hall, Tonopah	Anaconda Co.	P	T	OP
New Ruth	Kennecott Mining Co.	B	T	OP

See explanatory notes at end of table.

Table 2.—MEC molybdenum properties included in this study—Continued

Country and deposit name	Owner	Type[1]	Current status[2]	Mining method[3]
United States—Con.:				
New Mexico:				
Chino	Kennecott, Mitsubishi	B	P	OP
Copper Flat	Quintana Minerals, Philbro.	B	T	OP
Questa	Molycorp	P	T	BC
Utah: Bingham Canyon.	Kennecott Mining Co.	B	P[4]	OP
Washington:				
Mount Tolman	Colville Confederated Tribe.	P	E	OP
Sunrise	International Brenmac Development Corp.	B	E	OS
Wyoming: Kirwin	Amax, Inc.	B	E	OP

[1] P, primary; B, byproduct.
[2] P, producer; D, developing; E, explored, T, temporarily shut down. The status is as of March 1986.
[3] BC, block caving; FS, filled stope; OP, open pit (including bench-berm); OS, open stope; SU, combined surface and underground.
[4] Although Bingham Canyon Mine was closed as of March 1986, it is classed as producing because there are firm plans to reopen it.

NOTE.—All molybdenum ores are milled by flotation and all commercially exploitable molybdenum deposits are of hydrothermal type.

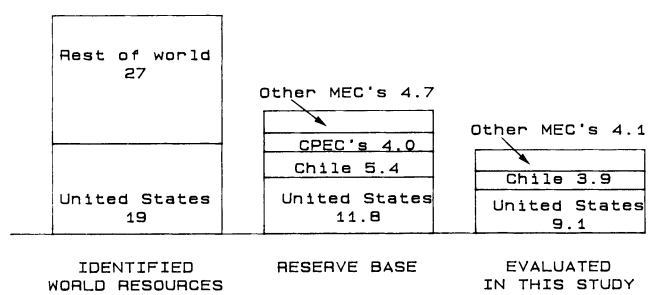

Figure 1.—Estimates of world molybdenum resources (billion metric tons).

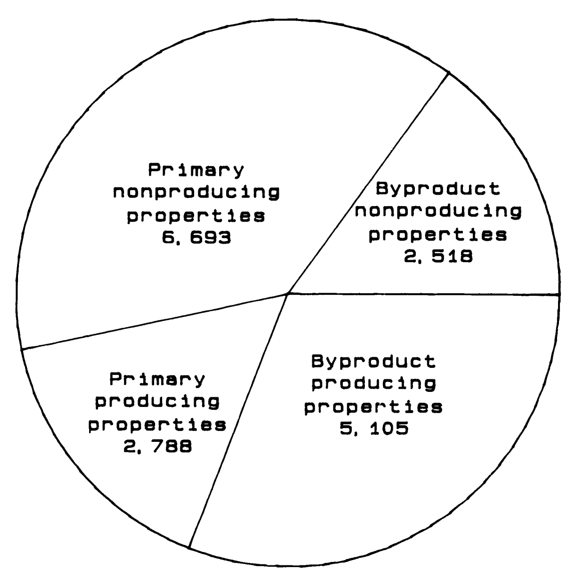

Figure 2.—Total recoverable molybdenum from primary and byproduct sources in MEC's (million pounds of molybdenum in concentrate).

U.S. AND WORLD HISTORICAL PRODUCTION

Molybdenum production in 5-yr intervals from 1955 to 1985 is shown in figure 3. Molybdenum production in 1985 was estimated to be 215.1 million lb (*3*). The United States, Chile, and the U.S.S.R. accounted for 80 pct of the 1985 production. Of the 183.2 million lb produced in market economy countries (MEC's), an estimated 43 pct came from primary molybdenum mines, mostly in the United States, which continued to lead the world in production. However, although not shown on figure 3, production in the United States dropped from 151 million lb in 1980 to slightly under 34 million lb in 1983, as both the Climax and Henderson Mines were temporarily shut down because of the slump in world steel demand. In the last 2 yr, domestic production has recovered to 103 million lb in 1984 and 108 million lb in 1985, but the 1985 output was still 28 pct below the 1980 peak. Inventories remained high in 1985, representing 40 pct of reported domestic consumption plus net exports. Much of domestic molybdenum production is exported; an estimated 59 pct of domestic output was exported in 1985.

Trends in the last few years have shown an increase in percent of world production from Latin America and CPEC's and a corresponding decrease from other MEC's. Chile, Peru, and Mexico have increased their combined share of world output from 14 pct in 1980 to 27 pct in 1985, while the share of CPEC production has gone from 12 pct in 1980 to 15 pct in 1985. At the same time, the U.S. share has dropped from 62 pct in 1980 to 50 pct in 1985, and that of other MEC's from 12 pct in 1980 to 8 pct in 1985.

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

Ore deposits are mined by both underground and open pit methods. The selection of a mining method for a given deposit is controlled by the grade, size, shape, and attitude of the ore body. Because primary molybdenum deposits occur as low-grade, high-tonnage porphyry deposits, low-cost underground block-caving and open pit operations are most commonly used to extract the ore economically.

Table 2 shows specific deposit data for the mines and deposits included in this study. Currently, of the 10 primary molybdenum producers and past producers, 2 use underground block-caving methods (Henderson and Questa), 6 use open pit methods (Cumobabi, Endako, Hall, Kitsault, Mineral Park, and Thompson Creek), and 2 use combined surface and underground methods (Boss Mountain and Climax). Of the producing and nonproducing byproduct operations, 6 are mined by underground block-caving methods, 3 by underground open stope and filled stope methods, 2 use combined surface and underground methods, and the remaining 57 are open pit.

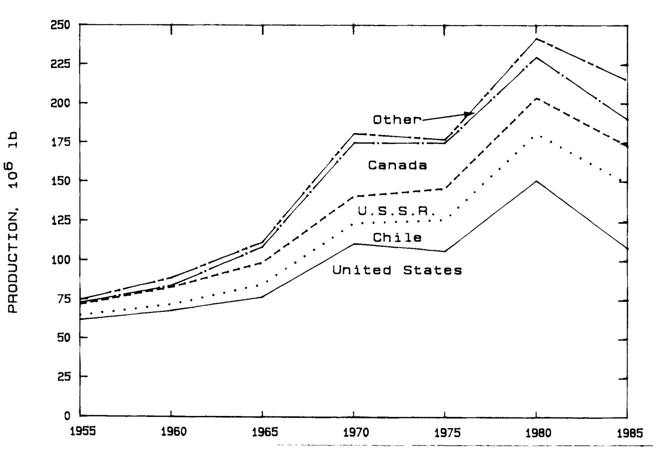

Figure 3.—World molybdenum production, 1955-85.

PROCESSING

In the beneficiation of primary molybdenum ores, to separate the molybdenite from the gangue minerals, the ore is crushed and ground to a size where the mineral is liberated. Molybdenite is a relatively easily floatable mineral, so concentration always involves crushing and grinding followed by flotation. Crushing of molybdenite ore consists of primary, secondary, and tertiary crushing systems. After crushing, the ore is fed to ball mills that are in closed circuit with classifiers. Most of the flotation reagents are added at the grinding stage. Molybdenite flotation practices are very similar for all operations. The only noticeable difference is in the equipment sizes, where new plants employ much larger, more efficient equipment than old plants. Both old and new plants produce a final concentrate of about 54 pct Mo (90 pct MoS_2).

Where molybdenum is a byproduct of copper, the rougher cells are designed to handle bulk flotation at the highest overall recovery of copper and molybdenite minerals at the coarsest grind possible. A much stronger collector is used for bulk flotation in the rougher cells, and a more selective reagent is used in the cleaner cells. A nonselective frother is also used in the rougher cells. Copper-molybdenum separation varies from one plant to another. However, the most commonly used technique to recover molybdenite from copper is the sodium hydrosulfide-sodium sulfide process. Although the hydrosulfide process can treat all copper-molybdenum concentrate separations, the economics of the process can vary because of differing mineral contaminants in the concentrate.

PRODUCTION COSTS

Table 3 and figure 4 illustrate long-term, weighted-average production costs by country for the primary molybdenum operations included in this study (expressed in January 1985 U.S. dollars per pound of molybdenum contained in concentrate). To maintain confidentiality, cost data on the two Mexican deposits are withheld. For the same reason, data for the temporarily shut down operations (Hall, Mineral Park, and Questa in the United States; Boss Mountain, Endako, and Kitsault in Canada) were combined with those for nondeveloped properties.

Mining costs for U.S. producing mines average \$2.17/lb, considerably higher than the \$1.57/lb average mining costs estimated for U.S. nonproducing deposits. Higher average mining costs for producing mines are attributed to the fact that about 50 pct of U.S. production is from underground operations, while six of the seven U.S. nonproducing deposits would be lower cost surface operations. The same reasoning explains the much higher estimated mining costs for Canadian nonproducing deposits, about half of which would be underground operations.

Average milling costs for U.S. producing mines, at \$1.04/lb, are lower than those estimated for both U.S. and Canadian nonproducing operations, at \$2.64/lb and \$2.23/lb, respectively. For milling, U.S. producing mines have a cost advantage over Canadian mines owing to a higher grade ore (0.182 pct for U.S. producers versus 0.078 pct in Canada), and economies of scale of the larger U.S. milling operations (13 million mt average capacity for U.S. producers and 6 million mt in Canada). Estimated milling costs for U.S. nonproducing deposits would be higher because of their lower average ore grades.

Because costs of primary molybdenum operations are evaluated f.o.b. mill, transportation costs are excluded. Estimated capital recovery costs for nonproducing deposits (excluding the temporarily shut down operations) are much higher, particularly in Canada, since none of the capital costs have been depreciated, as they have for producing mines. Estimated taxes and royalties per pound are also higher for nonproducers because of the higher revenues that would have to be earned in order to recover capital costs.

Given current prices of around \$3/lb of molybdenum, it is evident why few mines are able to remain in production and still fewer nondeveloped deposits are under development. Present market forecasts offer little prospect for an early resumption of production and development. Even at the high prices of 1979-81 (\$7.50/lb to \$9.70/lb), a number of the nonproducing deposits would be unable to operate and obtain a 15-pct discounted-cash-flow rate of return (DCFROR).

Costs for both Mexican properties (one producing mine and one nondeveloped deposit) are lower on average than equivalent properties in the United States and Canada.

Table 3.—Molybdenum production costs for producing and nonproducing primary molybdenum operations in selected MEC's

(All costs are in January 1985 U.S. dollars per pound of molybdenum contained in concentrate on a weighted-averaged basis)

	Operating cost		Less byproduct credit	Net operating cost	Recovery of capital[1]	0-pct DCFROR		15-pct DCFROR		
	Mine	Mill				Taxes and royalties[2]	Total cost[3]	Taxes and royalties[4]	Return on investment[5]	Total cost[6]
United States:										
Producers	2.17	1.04	0.06	3.15	0.31	0.08	3.54	0.19	0.33	3.98
Nonproducers	1.57	2.64	.33	3.88	.95	.18	5.01	1.35	2.86	9.04
Canada: Nonproducers	2.66	2.23	.04	4.85	1.26	.14	6.25	4.47	3.21	13.79

[1] Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure as of January 1985, and investments required over the life of the operation.
[2] Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 0-pct DCFROR.
[3] Equal to the sum of net operating costs, taxation, and capital recovery determined at a 0-pct DCFROR.
[4] Includes property, State, Federal and severance taxes and royalties, where applicable, calculated at a 15-pct. DCFROR.
[5] The revenue increase per pound necessary to obtain a 15-pct DCFROR.
[6] Equal to the sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery *plus* the per pound increase necessary to provide a 15-pct DCFROR after taxation.

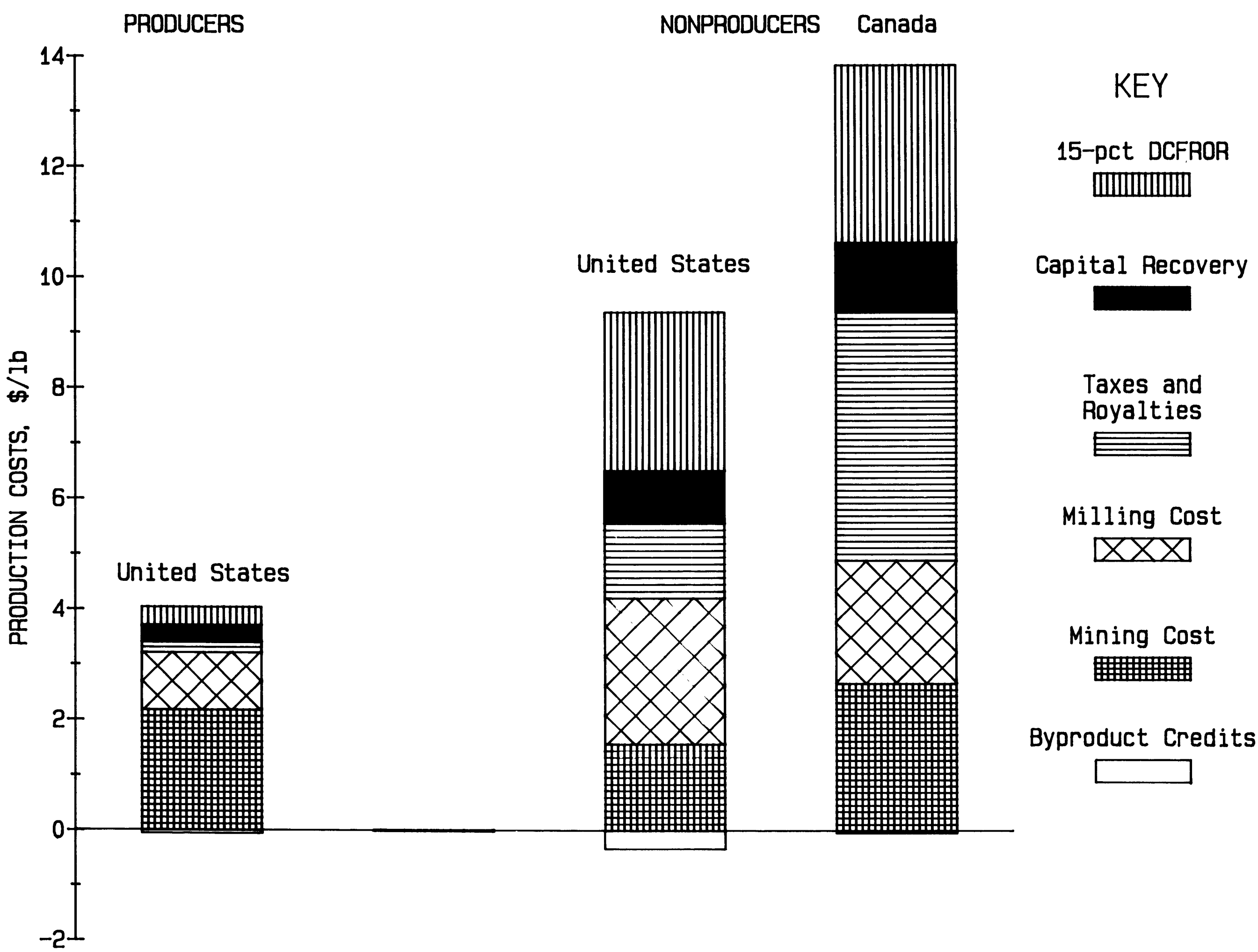

Figure 4.—Molybdenum production costs for primary molybdenum mines in the United States and Canada (January 1985 U.S. dollars).

AVAILABILITY

This analysis is limited to molybdenum available from MEC's. Lack of information precludes the inclusion of most CPEC's. It should be noted, however, that the U.S.S.R. is a major world producer (fig. 2). All costs are based on contained molybdenum in concentrate, f.o.b. mill.

A total of 92 mines and deposits in MEC's were analyzed, 34 domestic plus 58 foreign (table 2). These mines and deposits account for most of the molybdenum production and resources in MEC's. Of the domestic mines and deposits, 13 have molybdenum as the primary commodity, whereas 21 would produce molybdenum as a byproduct. The corresponding breakdown for the foreign mines and deposits is 11 primary and 47 byproduct. Table 4 shows the number of deposits of each type by production status.

TOTAL RECOVERABLE

The amount of molybdenum potentially available from the primary nonproducing deposits analyzed in MEC's is shown in figure 5; potential availability from byproduct producing and nonproducing deposits appears in figure 6. Confidentiality reasons precluded a graphic presentation of molybdenum availability by production cost level for primary producing mines owing to the small number of mines, but their data are included in the following discussion. (Total availability from primary producing mines, as shown in figure 2, is 2.8 billion lb.) Production cost levels are explained in the introduction to the Bulletin.

A total of 9.5 billion lb of molybdenum is potentially recoverable from the demonstrated resources of the evaluated MEC primary molybdenum deposits. Of the total, 29 pct occurs in mines that were producing in March 1986. At the 15-pct DCFROR level, a total of 2.7 billion lb is potentially recoverable from producing and nonproducing mines and deposits at a cost of under $5/lb. At a cost of $3.15/lb, which was approximately the average 1985 U.S. market price, no mines could produce and earn a 15-pct DCFROR. At the breakeven point (0-pct DCFROR), a total of 7.3 billion lb is potentially recoverable at a cost of under $5/lb, but

Table 4.—Evaluated MEC molybdenum deposits, by type and production status

	Producing		Nonproducing[1]	
	Primary	Byproduct	Primary	Byproduct
Argentina	0	0	0	3
Canada	0	5	9	10
Chile	0	5	0	6
Fiji	0	0	0	1
Iran	0	1	0	0
Mexico	1	1	1	0
Pakistan	0	0	0	1
Panama	0	0	0	1
Papua New Guinea	0	1	0	1
Peru	0	4	0	4
Philippines	0	0	0	3
United States	3	8	10	13
Total	4	25	20	43

[1] Includes temporarily shut down mines; Canada, 3 primary and 3 byproduct; Philippines, 3 byproduct; United States, 3 primary and 6 byproduct.

less than 1 billion lb would be available at a cost of $3.15/lb.

An additional 7.6 billion lb of molybdenum is potentially recoverable as a byproduct from the demonstrated resources of the evaluated MEC copper deposits. Of that amount, 67 pct occurs in mines that were producing in March 1986, most of which is potentially recoverable at a copper price of $0.75/lb or less (fig. 6). At the 15-pct DCFROR level, a total of 3.5 billion lb is potentially recoverable from producing mines at a cost of $0.75/lb of copper. At the average 1985 London Metal Exchange (LME) price of $0.64/lb, 2.6 billion lb is potentially recoverable from producing mines earning a 15-pct DCFROR. Only negligible amounts would be available from nonproducing byproduct deposits at those cost levels and rates of return. At the breakeven point (0-pct DCFROR), a total of 4.0 billion lb is potentially recoverable at a copper cost of $0.75/lb, and 3.5 billion lb would be available at a cost of $0.64/lb. The contrast between these numbers and those for molybdenum availability from primary sources explains the increased byproduct penetration in today's depressed market.

ANNUAL CAPACITY

Potential 1985 production at full capacity from producing MEC mines studied is shown in figure 7. The United States, Canada, Mexico, Chile, and Peru account for 98 pct of the 255 million lb of full-capacity molybdenum production potentially available from MEC producing mines in 1986. Table 5 compares availability and estimated production at average 1985 prices under different assumptions regarding the types of costs covered. Figure 8 gives the trends in annual availability at selected price levels based on the assumption of long-range profitability at a 15-pct DCFROR.

At the average 1985 price for molybdenum in the United States of $3.15/lb, 45 million lb could have been produced from primary molybdenum properties whose total costs (including capital recovery at a 0-pct DCFROR) could be covered at that price (table 5). Restricting coverage to operating costs would have made no additional amounts available. However, estimated primary production amounted to nearly 80 million lb, indicating that some mines may have remained in operation even while incurring longrun losses. In fact, two primary producers were shut down in 1985 and early 1986, and others were temporarily closed at times during the year. Most operated below capacity levels.

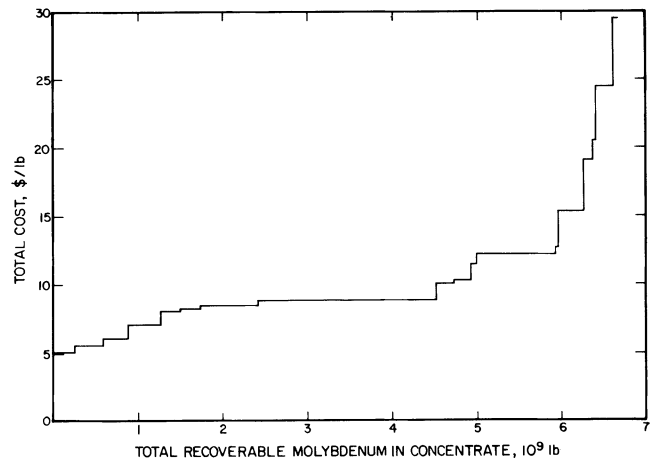

Figure 5.—Potential total molybdenum available from nonproducing primary molybdenum deposits in MEC's (January 1985 U.S. dollars).

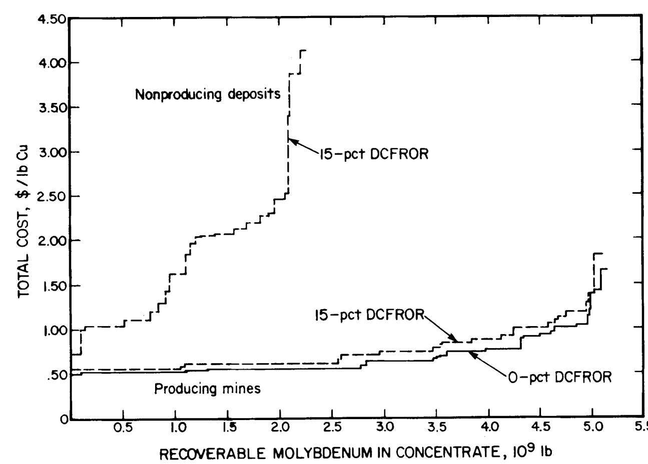

Figure 6.—Potential total molybdenum available from primary copper operations in MEC's (January 1985 U.S. dollars).

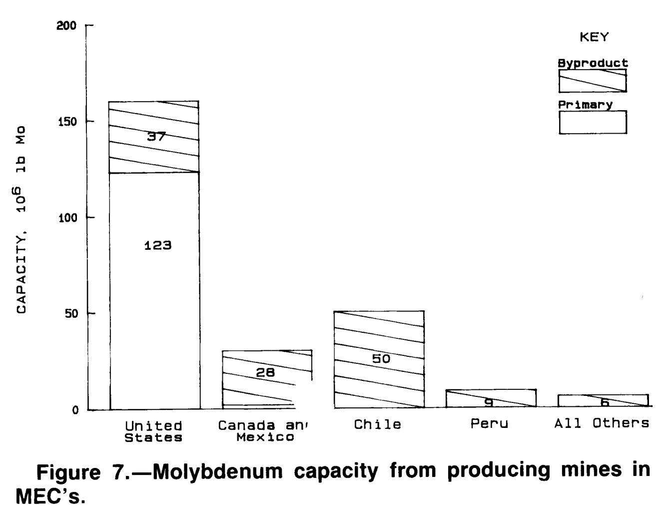

Figure 7.—Molybdenum capacity from producing mines in MEC's.

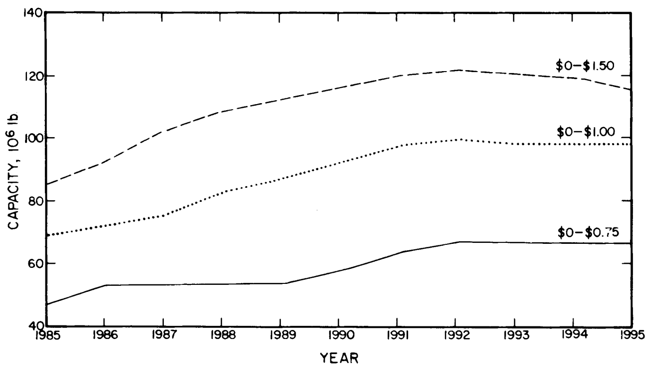

Figure 8.—Potential annual molybdenum production from producing primary copper operations in MEC's, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

Table 5.—1985 molybdenum capacity from primary and byproduct producing mines in selected MEC's, million pounds recoverable molybdenum

	Capacity		1985
	At 0-pct DCFROR	At operating cost[1]	estimated production
Primary, $3.15/lb Mo:			
Mexico	2.7	2.7	2.1
United States	42.3	42.3	77.3
Total	45.0	45.0	79.4
Byproduct, $0.64/lb Cu:			
Canada	0	0	16.7
Chile	43.3	43.3	40.5
Mexico	9.8	9.8	6.1
Peru	5.0	5.0	8.4
United States	6.3	29.7	30.7
Other MEC's	1.8	1.8	1.0
Total	66.2	89.6	103.4

[1] Includes mine and mill operating costs less byproduct credit.

Several byproduct molybdenum mines were shut down temporarily in 1985, mainly in the United States, and other U.S. and Canadian byproduct mines remained on care and maintenance. However, most mines in Chile and Peru operated at full capacity. At the average 1985 copper price on the LME of $0.64/lb, amounts varying from 42 to 90 million lb could have been produced from byproduct properties depending on the degree of capital and operating costs covered. In fact, slightly over 100 million lb was produced, again indicating the possibility that operations were not covering costs, mainly in Canada and the United States.

The level of byproduct molybdenum production is governed by conditions in the copper market, not the molybdenum market, and the incremental cost of molybdenum separation is small. Some of the foreign copper-molybdenum properties, particularly in Chile and Peru, are among the lowest cost copper producers in the world. They have an incentive to continue, and even increase, production because they can do so profitably and because they earn needed foreign exchange. Primary molybdenum producers are therefore in the position of filling the remaining demand above that supplied from byproduct sources.

The Bureau has forecast molybdenum demand to grow at annual rates of 0.8 pct in the United States and 3.3 pct in the rest of the world through the year 2000 (*2*). At those growth rates, U.S. demand in 2000 would be 71 million lb and demand in the rest of the world would be 190 million lb. If the producing primary and byproduct mines analyzed were to operate at full (100 pct) capacity from 1985 through 2000, including planned expansions, potential annual production of molybdenum in 2000 could be 144 million lb in the United States and 88 million lb in the rest of the world. These figures suggest a possible shortage of molybdenum in the rest of the world by that time and an increased dependence on U.S. sources. However, the existence of known, large, potentially low-cost deposits in Chile and Peru, plus several U.S. and Canadian primary and byproduct mines that are currently on care and maintenance (table 2), makes such a shortage highly unlikely. Inclusion of these evaluated nonproducing mines and deposits would make available an additional 166 million lb annually in the United States and 144 million lb in the rest of the world. These figures indicate adequate world production capacity and continued U.S. overcapacity through that period. Consideration of the large additional identified resources (fig. 1), which are divided about evenly between the United States and the rest of the world, suggests that world molybdenum supplies will be sufficient to meet demand in the foreseeable future, and that the U.S. primary molybdenum industry is likely to remain under pressure.

CONCLUSIONS

The U.S. molybdenum mining industry is currently suffering from a prolonged slump in demand, and market prices that are below the operating costs of most mines. In addition, South American operations, which produce molybdenum as a low-cost byproduct of copper mining, have been able to increase their share of the market. As a result, three of the six U.S. primary molybdenum mines have been temporarily shut down, the remaining producers are operating well below capacity, and most new developments have been postponed indefinitely.

Little change is likely to occur in the foreseeable future. Molybdenum will continue to be supplied from byproduct sources that are little affected by conditions in the molybdenum market. The capacity of current primary U.S. producers is more than sufficient to supply the remaining domestic and export demand.

REFERENCES

1. Palencia, C.M. Molybdenum Availability—Market Economy Countries. A Minerals Availability Appraisal. BuMines IC 9044, 1985, 30 pp.

2. Blossom, J.W. Molybdenum. Ch. in BuMines Mineral Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 521-534.

3. ______. Molybdenum. Ch. in BuMines Minerals Yearbook 1985, v. 1, pp. 701-708.

4. ______. Molybdenum. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 106-107.

5. Rosenkrantz, R.D., E.H. Boyle, Jr., and K.E. Porter. Copper Availability—Market Economy Countries. A Minerals Availability Program Appraisal. BuMines IC 8930, 1983, 30 pp.

6. King, R.V., D.R. Shawe, and E.M. MacKevett, Jr. Molybdenum. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 425-435.

NICKEL

INTRODUCTION

Nickel is a strategic and critical metal to modern industry. Nickel's strength and heat and corrosion resistance make it an important component in alloys, especially stainless steel. More than half of all primary nickel production is used in the manufacture of stainless steel and its proportion is growing. The United States has traditionally been dependent on Canadian mining operations for about 50 pct of its nickel imports (including ore mined in Canada and refined in Norway for import to the United States).

Nickel is derived from both sulfide and laterite ores. The bulk of nickel production originates from sulfide ores. Generally, sulfide ore is converted to nickel metal whereas laterite ore is processed to ferronickel. In this study, analyses were taken to refined nickel or nickel contained in the final product (i.e., ferronickel).

Most of the information presented in this chapter is an updated summary of Bureau of Mines Information Circular 8995 "Nickel Availability—Market Economy Countries. A Minerals Availability Program Appraisal" (*1*).[1] The Bureau publishes additional reports on the nickel industry describing important events, production and consumption, resource information, and other industry related data and statistics (*2-3*). Additional information on international deposits is available from a recent U.S. Geological Survey publication (*4*).

GEOLOGY AND RESOURCES

GEOLOGY

Sulfide

Nickel-bearing sulfide ores are formed by several geologic processes; however, the great majority are of igneous origin. The largest known primary nickel sulfide resource and nickel-producing district among market economy countries (MEC's), is located at Sudbury, Ontario, Canada. The Sudbury Complex consists of a generally differentiated magmatic body that contains a unit called the nickel irruptive. The nickel mineralization is primarily in the mineral pentlandite, but chalcopyrite and pyrrhotite are also contributors. Nickel feed grades generally range between 0.9 and 2.5 pct. Besides production of nickel, the district ranks third in world platinum-group metals (PGM) production and also produces important amounts of copper, cobalt, and precious metals.

The Thompson Nickel Belt (Thompson mines), in Manitoba, contains several ore bodies currently being worked. Nickel feed grades generally reach up to 2.5 pct. The ore bodies consist of uniformly disseminated sulfides and sulfide veinlets in peridotites and as mineralized breccia zones in metasediments. Although the ore grades are higher in the breccia deposits, they are less common.

The Duluth Gabbro (which contains the Birch Lake deposits, Dunka River, Ely Spruce, Minnamax, and Spruce pit), in Minnesota, is one of the world's largest known basic igneous intrusions and, although low grade, is also the largest single potential nickel resource in the United States. Nickel feed grades would probably range between 0.16 and 0.21 pct. Although there has been extensive exploration, there are no active mining operations.

Australia has numerous large nickel deposits. The majority of the deposits are associated with Precambrian ultramafic rocks, generally within metavolcanic belts. Nickel feed grades among the producing mines average about 2 pct.

Nickel is also recovered as a byproduct from PGM mining of the Merensky Reef (Impala, Rustenberg, and Western Platinum) in the Bushveldt Igneous Complex of the Republic of South Africa. Other important sources of nickel from polymetallic sulfide deposits are located in Botswana and Zimbabwe.

Laterites

Laterite ores are derived as a weathering product of ultrabasic rocks formed in tropical to subtropical environments. As nickeliferous olivine decomposes, nickel and other metals are released and mobilized into solution by the downward percolation of rainwater and/or the movement of ground water. Nickel, and sometimes cobalt, are redeposited at depth by chemical precipitation. This repeated action results in a zone of enrichment and, in some cases, can be mined as a resource.

There are basically two types of laterite ores, siliceous and limonitic. The silicate ores (garnierites) contain less than 30 pct Fe, about 30 pct silica, and a relatively high nickel grade, generally exceeding 1.5 pct. Garnierites are desirable for ferronickel production. The limonitic ores are higher in iron, about 45 to 55 pct Fe and about 1 pct Ni. Limonitic ores are amenable to leaching, by which process byproduct cobalt can sometimes be recovered. Important nickel-bearing laterites, based on size of resource and production history, are located in Greece (1.3 pct Ni), Indonesia (1.9 pct Ni), New Caledonia (2.4 pct Ni), and the Philippines (1.3 pct Ni). Laterites are also mined in Australia, Brazil, Colombia, the Dominican Republic, and the western United States.

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

RESOURCES

Important primary MEC nickel resources are located over several regions of the world, predominantly Australia, Canada, the Caribbean, the equatorial region of South America, Greece, Indonesia, the Philippines, the United States, and Yugoslavia. Figure 1 shows estimates of identified resources, the reserve base, and resources of the properties evaluated in this study. The Australian, Canadian, and U.S. resources are primarily sulfides, while most of the remainder in MEC's are laterites.

Nickel resource data, grouped by countries within regions, are presented in table 1. Table 2 is a list of the properties evaluated. Demonstrated MEC world resources (for the properties analyzed in this study and based on January 1981 data updated to January 1985) in terms of recoverable nickel, are estimated to be nearly 64 million mt. As shown, the Philippines, New Caledonia, and Canada have the largest recoverable nickel resources. Canadian resources, especially in the Sudbury District, are likely conservative owing to restricted access to data. Combined, New Caledonia and the Philippines contain nearly 55 pct of the total recoverable nickel resource, but only a relatively small portion can be recovered at a profit. Although the United States represents about 8 pct of the total, only a relatively small portion of this resource is developed to production status (about 182,000 mt or 0.2 pct).

As previously mentioned, nickel resources are divided into two broad categories based on geologic type, nickel from sulfide ore (mostly pentlandite) and laterite ore (limonitic or garnieritic). Sulfide ores compose about 30 pct of the recoverable nickel evaluated in this study. Sulfide resources are generally processed to elemental nickel products and are commonly associated with byproduct copper, cobalt, and precious group metals. The largest demonstrated sulfide resources, by far, are located in Canada, followed by the United States and Australia. None of the significant U.S. nickel resource in sulfide ores is developed to the extent where production will occur in the foreseeable future. The largest laterite resources, in terms of recoverable nickel, are in the Philippines, New Caledonia, and Indonesia.

Nickel-enriched ocean nodules and crusts represent a huge nickel resource. Although several designs for recovering the material from the ocean floor have been considered, none have entered commercial operation. Nodules typically contain 25 to 30 pct Mn, 1 to 1.5 pct Ni, 0.5 to 1.0 pct Cu, and 0.25 pct Co. In addition, testing and selection of a hydrometallurgical or pyrometallurgical process that can economically recover most of the valuable elements of the nodules, has not been completed. Legal issues over ownership also present complications.

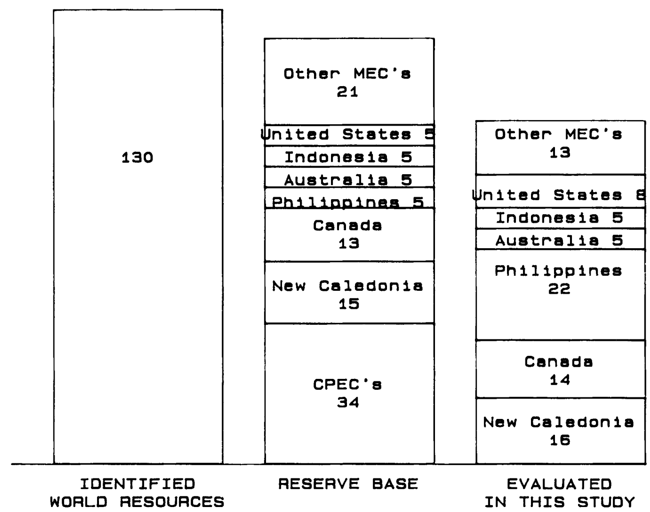

Figure 1.—Estimates of world nickel resources (million metric tons of contained nickel).

Table 1.—Summary of MEC demonstrated nickel resources evaluated for this study, as of January 1985,[1] thousand metric tons

Region and country	Producing mines: Number of deposits	Producing mines: In situ nickel	Producing mines: Recoverable nickel	Nonproducing mines: Number of deposits	Nonproducing mines: In situ nickel	Nonproducing mines: Recoverable nickel	Total: In situ nickel	Total: Recoverable nickel
Africa:								
Botswana	1	300	132	0	0	0	300	132
Zimbabwe[2]	4	120	81	1	23	17	143	98
Others[3]	0	0	0	4	4,308	3,759	4,368	3,759
Southwest Pacific:								
Australia	4	1,898	1,386	5	2,896	1,959	4,794	3,345
Indonesia	2	3,092	2,512	1	1,450	1,212	4,542	3,724
New Caledonia	7	5,500	1,657	6	10,519	9,030	16,019	10,687
Philippines	2	285	138	10	21,307	19,288	21,592	19,426
Europe:								
Finland	2	24	16	0	0	0	24	16
Greece	2	2,316	1,778	0	0	0	2,316	1,778
South America:								
Brazil	3	700	589	6	4,148	3,678	4,848	4,267
Others[4]	2	1,396	1,215	2	851	737	2,247	1,952
North America:								
Canada	15	8,182	6,119	14	5,417	3,712	13,599	9,831
United States[5]	1	192	182	20	7,718	4,899	7,910	5,081
Total	45	24,005	15,805	69	58,697	48,291	82,702	64,096

[1] Based on January 1981 resource data and updated by subtracting estimated production from operating mines to January 1985.
[2] Additional recoverable nickel resources of approximately 1.3 million mt are present in the Great Dyke PGM prospects but are very high cost.
[3] Includes India, the Ivory Coast, and Madagasgar.
[4] Includes Colombia, the Dominican Republic, and Guatemala.
[5] Includes Puerto Rico.

Table 2.—MEC nickel properties included in this study

Country and deposit name	Owner name	Ore type	Current status[1]	Mining method[2]	Product class[3]
Australia:					
Agnew	Agnew Mining Corp	Sulfide	P	U	1
Greenvale	Metals Expl. Ltd., Freeport Exp	Laterite	P	S	1
Kambalda	Western Mining Corp	Sulfide	P	U	1
Mt. Keith	Metals Expl., Freeport Exp	do	N	S	
Ora Banda (Siberia)	Western Mining Corp	Laterite	P	S	2
Sherlock Bay	Australian Inland Exp. Ltd	Sulfide	N	U	1
Wannaway	Western Mining Corp	do	P	U	1
Mt. Windarra, S. Windarra	Western Mining Corp	do	P	C	1
Wingellina	Texas Gulf	Laterite	P	S	1
Botswana: Selebi-Phikwe	BCL Ltd	Sulfide	P	U	Matte
Brazil:					
Barro Alto	Baminco-Mineracao	Laterite	N	S	Do.
Jussara	Companhia Minerada	do	N	S	Do.
Montes Carlos	Vatorantin Financial Group	do	N	S	Do.
Morro do Engenho	Companhia do Pesquiso	do	N	S	Do.
Morro do Niquel	Mineracao Sartenejo S.A	do	P	S	Do.
Niquelandia (CNT)	Companhia Niguel de Tocantins	do	P	S	1
Niquelandia (Codemin)	Codemin	do	P	S	1,2
Sao Feix do Xingu	Mineracao do Sul	do	N	S	2
Sao Joao do Piaui	Rio Doce Geologica e Mines	do	N	S	2
Canada:					
Birchtree	Inco	Sulfide	N	U	1
Bowden Lake	Falconbridge and others	do	N	U	1
Bucko Lake	Bowden Lake Nickel Mines	do	N	U	1
Copper Cliff North	Inco	do	P	U	1
Copper Cliff South	do	do	P	U	1
Crean Hill	do	do	N	S	1
Creighton	do	do	P	U	1
Dumont Nickel	Boliden	do	N	S	1
Expo Ungava	Expo Ungava Mines Ltd	do	N	S	1
Falconbridge	Falconbridge	do	N	U	1
Falconbridge East	do	do	P	U	1
Fraser	do	do	P	U	1
Frood	Inco	do	P	U	1
Garson	do	do	P	U	1
Levack	do	do	P	U	1
Little Stobie	do	do	P	U	1
Lockerby	Falconbridge	do	P	U	1
Moak	Inco	do	N	S	1
Mystery Lake	do	do	N	S	1
Onaping-Craig	Falconbridge	do	P	U	1
Pipe Underground	Inco	do	P	U	1
Raglan	Falconbridge	do	N	U	1
Shebandowan	Inco	do	SB	U	1
Soab	do	do	N	U	1
Stobie	do	do	P	U	1
Strathcona	Falconbridge	do	P	U	1
Texmont	New Texmont	do	N	U	1
Thompson	Inco	do	P	U	1
Totten	do	do	N	U	1
Colombia:					
Cerro Matoso	Econiquel, Billiton, Hanna	Laterite	P	S	1,2
Ure district	Colombian Government	do	N	S	2
Dominican Republic: Falcondo Bonao	Falconbridge	do	P	S	2
Finland:					
Hitura	Outokumpu Oy	Sulfide	SB	S	1
Kotalahti	do	do	P	U	1
Greece:					
Euboea	Greek National Bank (Prev. LARCO)	do	P	C	2
Aghios Ioannis	do	do	P	C	2
Guatemala: Exmibal (Elestor)	Exmibal	do	N	S	1
India: Sukinda	Tata Iron & Steel	do	N	S	1
Indonesia:					
Gag Island	P.T. Pacific Nickel	do	N	S	1
Pomalaa	P.T. Aneka Tambang	do	P	S	2
Soroako	P.T. Inco	do	P	S	2
Ivory Coast: Biankouma region	Ivory Coast Government	do	N	S	2
Madagascar:					
Ambatovy, Analamy	Malagasy Government	do	N	S	2
Valozoro	do	do	N	S	2
New Caledonia:					
Goro	Inco	do	N	S	1
Ile Art	Cofremni	do	N	S	1
Kouaoua	Societe le Nickel	do	P	S	1,2
Moneo	CGMC, Ballande	do	P	S	1,2
Nakety	Societe le Nickel	do	P	S	1,2
Nepoui	do	do	SB	S	1,2
Poum	Cofremni	do	N	S	1,2
Poro	Societe le Nickel	do	P	S	1,2
Prony	Penamex	do	N	S	1
Quaco	Societe le Nickel	do	N	S	1
Quinne	do	do	P	S	1
Tiebaghi	Cofremni	do	N	S	1
Thio	Societe le Nickel	do	P	S	1,2

See footnotes at end of table.

Table 2.—MEC nickel properties included in this study—Continued

Country and deposit name	Owner name	Ore type	Current status[1]	Mining method[2]	Product class[3]
Philippines:					
Acoje	Acoje Mining Company	Sulfide	N	S	1
Borongan	G.Y. Ornopia and Associates	Laterite	N	S	2
Dinagat	Nonoc Mining and Industrial Corp.	..do	N	S	2
Infanta (Ipilan)	Philippine Government	..do	N	S	2
Ipilan	De Lara Mining Corp	..do	N	S	2
Makambal	LBC & Greenfield Mining Corp	..do	N	S	2
Mount Kadig	Horizon Minerals and Oil	..do	N	S	2
Nonoc Mine (Surigao)	Nonoc Mining and Industrial Corp	..do	SB	S	1
Rio Tuba	Rio Tuba Mining Co	..do	P	S	2
Sablayan	Anglo Phil. Oil Corp	..do	N	S	2
Santa Cruz	Benquet Consolidated	..do	N	S	2
Soriano	Soriano Corp.	..do	N	S	1,2
United States:					
Birch Lake area	Hanna, Inco, Duval	Sulfide	N	S	1
Birch Lake (6 deposits)	Inco, Duval, Hanna	..do	N	U	1
Brady Glacier	Newmount Mining Corp	..do	N	U	1
Crawford Pond	Knox Mining Corp	..do	N	S	1
Dunka River area	AMAX	..do	N	U	1
Ely Spruce	Inco	..do	N	U	1
Gasquet	California Nickel Corp	Laterite	N	U	1
Guanijibo	Puerto Rico Government	..do	N	U	1
Madison	Anschutz Corp	Sulfide	N	U	1
Minnamax	Bear Creek Mining	..do	N	U,S	1
Nickel Mountain[4]	Hanna Mining	Laterite	P	S	2
Partridge River	AMAX	Sulfide	N	S	1
Pine Flat area	Hanna Mining	Laterite	N	S	1
Red Flat	Hanna Mining, Red Flat Nickel	..do	N	S	1
Spruce Pit	Inco	Sulfide	N	S	
Yakobi Island	Inspiration Development	..do	N	S	1
Zimbabwe:					1
Empress	Rio Tinto Mining	..do	N	U	1
Epoch	Trojan Nickel Mining Ltd	..do	P	U	1
Madziwa (Shamva)	Madziwa Mines Ltd	..do	P	U	1
Shangani	Trojan Nickel Mine Ltd	..do	P	U	1
Trojan	..do	..do	P	U	1

[1] As of January 1986: P, producer; N, nonproducer; SB, standby (those mines which have not produced in 3 yr or less, but are on a maintenance program).
[2] S, surface; U, underground; C, combined.
[3] Class 1 products include cathode nickel, nickel pellets, etc. Class 2 products include nickel matte and ferronickel.
[4] Closure pending.

U.S. AND WORLD HISTORICAL PRODUCTION

Primary nickel production, from 1950 to 1985, is shown in 5-yr intervals on figure 2. In 1985, production was estimated to be about 745,000 mt from mines in more than 21 countries. Nearly 70 pct (502,500 mt) was from MEC's; Canada accounting for 24 pct of total world production, a large decrease from the position it held in the early to mid-1970's. Production from the centrally planned economy countries (CPEC's) was estimated to be 242,000 mt, 75 pct of which was from the U.S.S.R. Other important producers, beside Canada, include the U.S.S.R. (24 pct), Australia (10 pct), Indonesia (9 pct), New Caledonia (5 pct), and the Dominican Republic, the Republic of South Africa, and the Philippines, with 3 pct each. Production from several of these countries (Indonesia, Greece (Europe) and the Philippines) was significantly lower in 1975 because their mining industries were just beginning dèvelopment. Canada's market share of world nickel in ore production has dropped dramatically. In 1975, Canada produced about 30 pct (245,000 mt) of the world's nickel, but in 1985 had dropped to 24 pct (177,000 mt) or about 70 pct of its 1975 production. This change was brought about by several elements such as air quality restrictions affecting the smelting and refining sector, displacement by new suppliers, and poor market conditions. New Caledonia's production share dropped from 15 pct (133,000 mt) in 1975 to 5 pct (40,000 mt) in 1985. Most of this change was brought about by poor market conditions and labor problems associated with the country's movement towards independence from France.

World nickel production could increase significantly, if there is a sudden increase in demand resulting from major

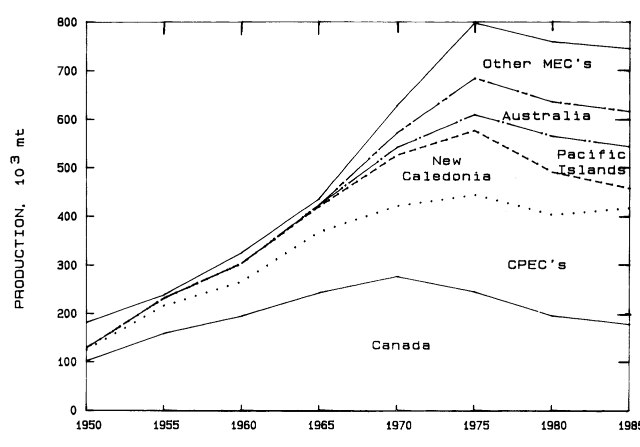

Figure 2.—World nickel production, 1950-85.

capital investments in industries requiring corrosion-resistant metals and alloys. Of great importance to the mine producers is the growth in demand for stainless steel and what percentage of that new demand could be met by scrap. In the United States, approximately 50,000 mt of nickel in scrap was recovered in 1985. In 1985, U.S. primary nickel mine production was reported to be 6,750 mt, from one company that produced ferronickel. MEC production in 1986 is projected to be about 542,000 mt (*5*).

As a result of a supply glut, MEC nickel smelter-refiner capacity has been operating at between 55 and 65 pct of capacity since 1983. Much of this unused capacity originates in Canada, where production dropped from over 235,000 mt of nickel in 1974 to about 89,000 mt in 1982, and rebounded to about 163,000 mt in 1984.

The last availability study undertaken relating to nickel was published in 1984 and contained comprehensive data describing the nickel industry up through part of 1981 (*1*). During the interim, there have been widespread changes throughout all levels of the nickel mining industry. These include major concessions sought from labor and utilities, as well as technological advances brought about by high labor and energy costs. Where concessions by management were not initially acceptable, strikes occurred. The plants affected in 1984 were Brazil's Cia Niquel Tocantins refinery, Australia's Western Mining operation, and Greece's Larco ferronickel refinery. The most important technological advance has been the continued development of more efficient bulk-mining techniques (vertical crater retreat or VCR) at Canada's Sudbury and Thompson Districts.

Poor market conditions, underutilized capacity, and already defined large high-grade deposits, have discouraged nickel exploration and development activities, except where cost cutting can be realized.

EXTRACTION AND PROCESSING TECHNOLOGY

The nature of the geologic occurrence of sulfide and laterite ores usually dictates underground mining for sulfides and surface mining for laterites. Additionally, differences in ore chemistry require different processing methods to produce a final marketable product.

Following is a brief discussion of the mining and processing methods for sulfides and laterites.

SULFIDES

The majority of nickel sulfide ores are taken through four basic steps in order to recover nickel and associated coproducts and byproducts: mining, beneficiation, smelting, and refining. In some cases, concentrates can be processed directly by hydrometallurgical methods. Class 1 products, consisting mostly in the form of nickel cathodes and carbonyl pellets, are the most common product from sulfide ores.

Mining

The mining of nickel sulfide ore is accomplished primarily by underground methods, although surface methods are also used. Surface or open-pit mining is based on standard engineering practices using power shovels and trucks and are generally accomplished without any special mining problems.

About 80 pct of the evaluated sulfide resource is, or would likely be, mined by underground methods. The underground nickel operations at Sudbury represent the state of the art mining methods. VCR, mechanized cut-and-fill, and large-diameter underground blasthole mining are methods used by Inco. Radio-controlled load-haul-dump (LHD) vehicles are used. The bulk mining methods are a major change from historical mining methods and, in some cases, have resulted in productivity improvements of over 300 pct. Presently, about 80 pct of Inco's mine production originates from bulk mining, and it is projected to comprise more than 95 pct by the end of the century.

Processing

Beneficiation

Beneficiation of sulfide ores is generally more complicated than laterite ores. Sulfide ore requires size reduction by crushing and grinding, followed by initial separation (usually by magnetic means) and flotation. The flotation circuits are designed to maximize the recovery of specific minerals or metals. In Canada, high copper with nickel and high nickel with copper concentrates are recovered. Nickel recoveries may reach 90 pct and concentrates may grade 12 pct Ni. Depending on ore chemistry, the concentrates may contain, in addition to nickel and copper, recoverable cobalt, gold, silver, and PGM's.

Postmill Processing

Concentrates produced from sulfide ores are generally treated by pyrometallurgical and/or hydrometallurgical methods.

The pyrometallurgical methods process concentrates to a matte in three stages: roasting, smelting, and converting.

Roasting is performed on concentrates that contain more iron than nickel and have a high sulfur content, and is accomplished by applying heat to the concentrate in an oxygen-enriched atmosphere resulting in the oxidation of contained metal sulfides and the removal of nearly 50 pct of contained sulfur (an important aspect for air quality).

Smelting is used to melt the concentrate and reduce it chemically to produce a separation of desired metal(s) or other constitutents. Smelting of most roasted concentrates is accomplished by use of a flash or electric furnace. The flash smelting process involves injection of concentrate and flux in an oxygen-enriched atmosphere, which produces an exothermic reaction. The electric furnace operates by passing an electric current through the concentrate, which melts the concentrate. A low-cost energy source is required to use this method economically.

The conversion process oxidizes iron and sulfur, in the presence of high temperature, in order to eliminate residual iron sulfide. The final matte contains about 50 pct Ni with a 90-pct recovery of the nickel contained in the original concentrate. The matte can be cast into anodes in preparation for direct electrolytic refining to nickel metal or it can be crushed and dissolved for electrowinning. Recovery of other metals contained in the matte is accomplished by processing the resulting slimes by several methods including chemical precipitation and electrolytic methods.

Hydrometallurgical methods can be used to process nickel concentrates into nickel metal plus recover other desired byproducts. The method involves preferential leaching, by chemical means, of the desirable metals. Either sulfuric acid or ammonia solutions can be used. Ammonia solutions are an effective leaching agent because they can selectively put nickel, copper, and cobalt into solution without reacting with carbon steel equipment, as do acid leach chemicals. The leaching process can usually be used without a roasting step. Despite the efficient recoveries of the system, high energy requirements, process controls, and effluent cleaning result in relatively high costs.

Several methods are used to produce refined nickel metal. They include electrowinning, electrorefining, and evapometallurgy. Electrowinning is a process for recovering a metal previously dissolved in an electrolyte. An electric curent passing through the solution causes the metal to deposit on a cathode. The cathode nickel sometimes requires additional electrorefining steps to purify the metal. Electrorefining of nickel anodes is accomplished by placing the anodes in a cell or tank containing an electrolyte solution. An electric current dissolves the anodes and reacts to form a plating on the cathode. Impurities at the bottom of the tank are removed and processed further to recover other products, including precious metals. The evapometallurgical process, or carbonyl gas process, requires impure nickel granules as feed. The nickel is converted to a gas in the presence of heat in a nickel-iron carbonyl reactor. The vapor is then condensed to a carbonyl liquid from which high-purity nickel is precipitated.

LATERITES

Laterites are generally mined by open-cut and modified open-pit methods. The generally soft nature of most laterite ores allows for mining with power shovels, draglines, front-end loaders, and bulldozers, with a minimum of blasting. In some cases, laterite mining is a simple earth moving procedure. One of the largest problems confronting some laterite mining operations is the relatively high moisture content (up to 30 pct), which adds weight and reduces equipment traction. Other problems relate to grade and ore chemistry control and mining difficulties associated with topography.

Underground mining of laterites is rarely undertaken because of high costs associated with ground control; however, in Greece, some laterite ores are mined by cut-and-fill methods.

Ore chemistry determines whether the laterite will be processed into nickel or ferronickel. In general, ores in the limonitic zone are treated by hydrometallurgical methods, and garnieritic (silicate) ores are processed by pyrometallurgical methods.

The most commonly applied pyrometallurgical methods consist of direct reduction to ferronickel, the major class 2 nickel product, and smelting to a nickel matte. Over 80 pct of the installed nickel capacity for laterites is for producing ferronickel. After the ore is dried, crushed, and screened, it is directed to calciners in order to remove or decompose carbonates and moisture. The hot calcine is transported to the smelter and is fed, along with reducing agents, to furnaces to produce an impure ferronickel. The molten ferronickel is then refined by dephosphorizing and deoxidizing. Final ferronickel assay is usually about 36 pct Ni and 62 pct Fe, with the remainder composed of cobalt, sulfur, carbon, and silicon. Approximately 90 pct of the nickel contained in the laterite is recovered. The Nickel Mountain ferronickel operation, in Oregon, produces a product that assays about 49 pct Ni.

Only a small amount of laterite ore is converted to matte. A mixture of ore, coke, and limestone and a sulfur source are fed into a smelter. The resulting impure material is then fed to a refiner that removes much of the remaining iron and sulfur. The refined nickel sulfide matte ideally assays about 75 pct Ni and 25 pct S, but there are usually some impurities. New Caledonia and Indonesia employ this method. The matte is later refined to nickel and associated byproducts. Cobalt and copper are the most commonly recoverable metals.

Limonitic ores, and in some cases mixed ores, are amenable to hydrometallurgical methods. Two methods are used: reduction roasting followed by ammoniacal leaching or sulfuric acid leaching.

The ammoniacal leaching process consists of a series of steps of solution and precipitation resulting in a separate nickel and cobalt product. The nickel product assays between 90 and 99.8 pct Ni with an overall estimated recovery of about 75 pct. This process, with some variations, is used at Greenvale, Australia, and at operations in the Philippines.

The sulfuric acid process requires leaching of the ore with hot sulfuric acid. After the metals are put into solution they are precipitated and refined. Overall recovery ranges between 60 and 90 pct. This process is used at Moa Bay, Cuba.

PRODUCTION COSTS

Economic pressures in the nickel industry over the last 5 yr have brought major changes to the industry. Most mining companies that operate underground sulfide operations have made important strides in operating cost reductions. Historically, sulfide operations (which are generally hard rock and underground) have a high labor cost component in their operating cost while laterite operations, because of high energy consumptive pyrometallurgical processing, have relatively large fuel and energy cost components. Figure 3 illustrates this relationship estimated for the 1984-85 period.

The mines in Sudbury, Canada, in particular, have been able to decrease their costs by more than 25 pct (mostly in the labor sector) since 1982, while at the same time have increased their productivity by reducing the workforce by about 30 pct. From 1983 to 1984, productivity increased by about 18 pct. The unit cost of producing nickel increased by nearly 30 pct from 1980 to 1982 but by mid-1985, was 5 pct below the 1980 level. The productivity increase and reduction in labor requirements resulted primarily from the development of bulk mining techniques and the introduction of continuous mining equipment. Canada is the lowest cost nickel-producing country among MEC's. It is estimated that it can currently produce nickel for less than $1.80/lb at a 0-pct discounted-cash-flow rate of return (DCFROR).

Nearly 50 pct of current MEC nickel capacity is in laterites. Efforts for cost reductions at laterite operations have also met with some success. Laterite operations have a much smaller labor portion to their operating costs owing to their simple surface mining techniques, relatively small labor requirements, and low wages. The laterite operations are most sensitive to energy-intensive treatment costs and require three to four times more energy per pound of nickel produced than sulfide operations. About 60 pct of the total operating cost in a laterite deposit originates from energy costs. Low metal prices, high costs of energy, and large underutilized capacities were directly responsible for some closures or continued financial losses among laterite operations.

In general, laterite operating costs change about $0.05/lb of nickel for every $1/barrel change in oil costs, whereas Australian and Canadian sulfide operations are affected relatively little by changes in oil costs. The recent reduction in energy costs, particularly oil, could provide laterite operations a temporary opportunity for increased competition with sulfide operations. Currency devaluations in some countries have also effectively lowered costs to the Western market. The Falconbridge operation, in the Dominican Republic, has benefited from lower fuel costs and is now a marginal cost producer.

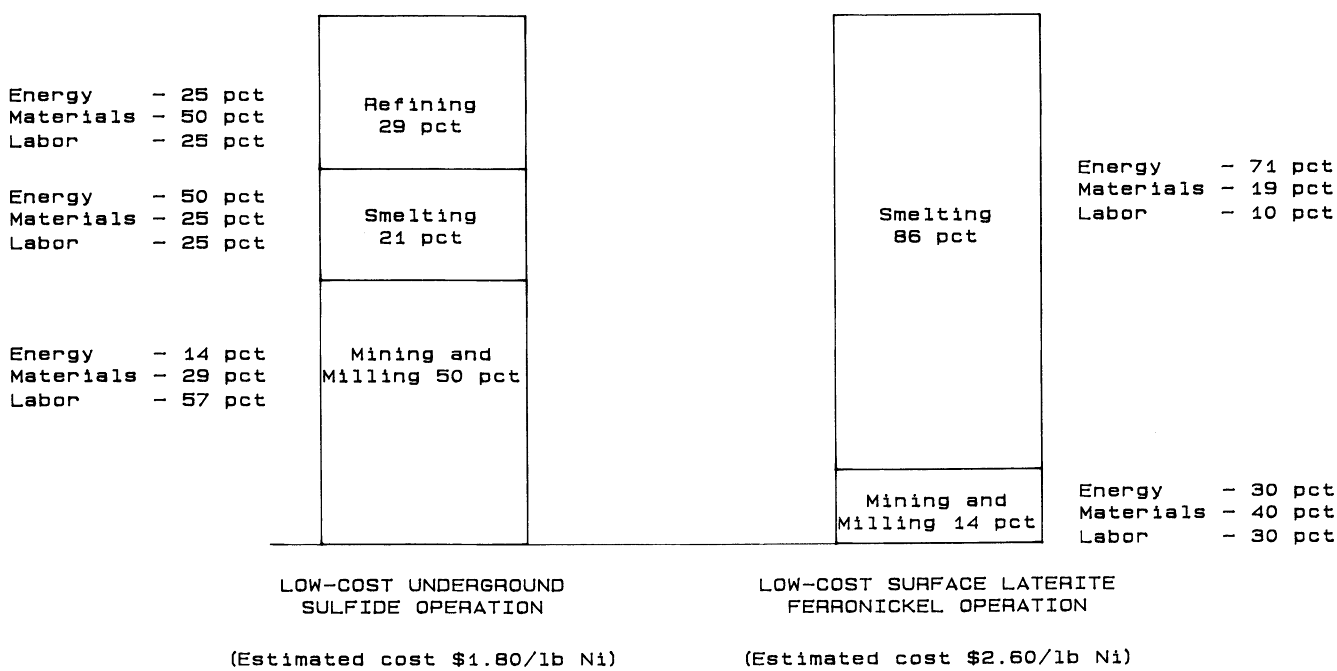

Figure 3.—Production cost comparison of low-cost underground sulfide operation and surface laterite ferronickel operation.

AVAILABILITY

During the first half of 1985, nickel achieved a modest recovery from 1984 prices. The average price for nickel in 1984 was approximately $2.22/lb, while by May of 1985 the price had reached nearly $2.60/lb. The increase in price was a direct result of increased consumption, thereby reducing the amount of refined nickel on hand. During the second half of 1985, however, the price of nickel fell, and by December the price of the metal had fallen back to below $2.00. Nickel sold for $1.70/lb to $1.80/lb by October 1986 and there were indications that it would fall further. The cause of the downturn has been attributed to a slowdown in world consumption, as well as nickel exports from the Soviet Union. In light of the sudden decline in prices, Inco has announced extended closures or reduced production at several of its Sudbury operations; Falconbridge, as well as other companies, are following suit. The nickel industry is currently operating at below capacity; any increase in demand can be met, and at a low cost, from Canadian or other producers.

A total of 114 mines and deposits in MEC's were analyzed in this study—21 domestic and 93 foreign. Approximately 64 million mt of nickel is potentially recoverable from these mines and deposits, and they account for nearly all of the primary nickel production in the MEC's. Forty-six of the mines evaluated, which account for approximately 30 pct of the estimated recoverable resource, were assumed to be in production (one in the United States) or temporarily on standby for 3 yr, or less. A total of 69 nonproducing deposits or mines were also evaluated, 20 in the United States (including Puerto Rico). Some of these properties are only explored or raw prospects. Resource and production cost data for mines or deposits in CPEC's were not available and were therefore not analyzed.

Most of the low-cost nickel from producing operations (less than $2/lb at a 0-pct DCFROR) originates from sulfide deposits in two countries, Australia and Canada (which in 1985 produced 85,000 mt and 152,000 mt of nickel in ore, respectively). Canada has over 50 pct of the recoverable nickel among sulfide deposits evaluated and over 40 pct of recoverable nickel from all evaluated MEC producers. Despite the relatively low cost for Canadian operations, refined nickel production is approximately 50 pct of estimated capacity. A relatively small amount of nickel originates from the PGM ore in the Republic of South Africa (approximately 25,000 mt annually), and is produced at essentially only the cost of refining the product.

Despite the huge, high-grade nickel laterite resources of New Caledonia, production of ferronickel, matte, and shipments of raw ore have dropped dramatically. New Caledonia has an estimated capacity of over 45,000 mt of nickel contained in ferronickel, plus 16,000 mt of nickel recovered from matte processed in France, and shipments of raw ore (averaging about 50,000 mt of contained nickel during the 1970's). In 1984, ferronickel production totaled 29,000 mt of nickel in ferronickel, less than 6,000 mt of nickel in matte, and shipments of raw ore, about 25,000 mt of contained nickel. An eventual return to the 1975 peaks of 71,000 mt of nickel in matte and ferronickel is possible. Production of refined nickel and cobalt is also feasible with construction of a facility or by supplying ore to the Yabulu refinery in Australia. The three most important elements for a return to competitiveness are lower energy costs (perhaps achieved by conversion to less expensive Australian coal); political stability, hopefully solved through peaceful negotiation; and infusion of new capital.

Total evaluated U.S. nickel resources, at the demonstrated level, amount to approximately 5.1 million mt of recoverable nickel and less than 5 pct is in the producing Nickel Mountain Mine. Except for Hanna's producing Nickel Mountain Mine in Oregon, only two additional U.S. resources, the Gasquet and the Madison, may see development to production this decade, but is less likely for other resources. Development of the Gasquet and Madison may well depend on cobalt prices, which are significant byproducts. The Hanna operation is experiencing difficulty remaining profitable and has instituted extended closures because of low metal prices and technical problems. The large available capacity, well-developed infrastructure, plus the relatively low costs at nearby Sudbury, hinder development of additional nickel mining operations in the United States.

Despite the poor economic environment for profit among many nickel producers, specifically the laterite operations, production continues, even at significant losses. Some of these mines continue to operate for several reasons: hopes for market recovery, the desire for currency exchange, to maintain employment levels, to pay interest on loans, etc. Much of the uneconomic production originates from laterites in Indonesia, the Philippines, and New Caledonia, where total costs, at a 0-pct DCFROR, exceed the market price of nickel.

CONCLUSIONS

Approximately 64 million mt of nickel is potentially recoverable from the deposits and properties evaluated in this study, of which about 25 pct is in properties considered producers. Although the largest evaluated demonstrated nickel resources are located in New Caledonia and the Philippines, there has been relatively little new development or continued growth in production owing to high production cost and political problems. Canada, the largest MEC nickel producer and the third largest resource, plus Australia, are probably in the best economic position among the large producers. In 1985, laterites represented about 45 pct of nickel in ore production among MEC's. Although the majority of laterite operations were unprofitable during the last 5 yr, recent reductions in fuel costs along with high grading have provided an opportunity for some laterite mines to return to profitability. The opportunity could be short term because of demands for higher worker wages and benefits plus a return to higher fuel costs.

Despite the fact that the United States has large undeveloped nickel resources, it may be many years before there is any large-scale development. Since 1980, New York dealer prices have fallen from a high of about $3.25/lb to less than $1.80/lb. Prices did not improve in 1986. Poor market conditions have resulted in about a 55- to 65-pct utilization of estimated smelter-refinery capacity. Nickel prices could increase significantly if there is a sudden increase in demand resulting from major capital investments in industries requiring corrosion-resistant metals and alloys.

Canada is in the best position to benefit from a recovery owing to its high-grade ore bodies, large swing capacity, and a well-developed marketing network. The United States has traditionally been reliant on Canada to meet the majority of its nickel requirements and will likely continue to be for the foreseeable future.

REFERENCES

1. Bleiwas, D.I. Nickel Availability—Market Economy Countries. A Minerals Availability Program Appraisal. BuMines IC 8995, 1984, 33 pp.

2. Chamberlain, P.G. Nickel. Ch. in BuMines Minerals Yearbook 1985, v. 1, pp. 709-724.

3. ______. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 108-109.

4. DeYoung, J.H., D.M. Sutphin, A.B.T. Werner, and M.P. Foose. International Strategic Mineral Inventory Summary Report—Nickel. U.S. Geol. Surv. Circ. 930-D, 1985, 62 pp.

5. Engineering and Mining Journal. Nickel. V. 187, 1986, pp. 52-53.

PHOSPHATE ROCK

INTRODUCTION

Phosphate rock, the only significant commercial source of the element phosphorus, is of vital importance to an expanding agricultural sector worldwide. Phosphorus, nitrogen, and potassium are the three primary nutrients necessary for plant growth. When these elements are either lacking or depleted from the soil, their addition is necessary to reestablish high agricultural yields. The growth of world agricultural production partially depends on the availability of phosphate fertilizers.

The United States has traditionally been the world's largest producer and net exporter of phosphate rock and related products. However, the U.S. producers are facing the challenge of foreign competition for export markets (primarily from Morocco), and rising production costs for Florida phosphate will make it more difficult to meet foreign competition in future years.

In this study, availability is presented in terms of phosphate rock. Following industry practice, the term "phosphate rock" is defined as the beneficiated product of phosphate ore rather than the in situ material. After beneficiation, phosphate rock ranges from 26 to about 34 pct P_2O_5 (phosphorus pentoxide). Phosphate rock can be converted to phosphoric acid and other fertilizers, converted to elemental phosphorus in an electric furnace, or applied directly to acidic soils as direct-application fertilizer.

Most of the information presented in this chapter is an updated summary of Bureau of Mines Information Circular 8989 "Phosphate Rock Availability—World. A Minerals Availability Program Appraisal" (*1*).[1] Additional information on the domestic and foreign phosphate industry is available from other publications (*2-5*).

GEOLOGY AND RESOURCES

GEOLOGY

The element phosphorus is widely abundant in the Earth's crust, composing approximately 0.11 pct. Phosphate (P_2O_5) concentrations exist throughout the world in both igneous and sedimentary rocks, primarily in the form of the mineral apatite. In igneous rocks it is generally as fluorapatite, $Ca_5(PO_4)_3F$, and in sedimentary rocks generally as hydroxy fluorapatite, $Ca_5(PO_4)_3OH,F$ or as carboxyapatite, $Ca_5(PO_4, CO_3OH)_3F$.

The majority of the phosphate resources throughout the world are classified as sedimentary marine phosphorite deposits. The two most significant of these types are deposits formed in areas of upwelling water and those formed in warm climates, particularly along eastern coasts. The Cretaceous and Eocene deposits of western and northern Africa and the Middle East, as well as the Permian Phosphoria Formation of the Western United States, are the best examples of deposits formed as a result of upwelling waters. These deposits formed because of chemical and biological precipitation of phosphate in areas of upwelling, phosphate-rich marine waters. The phosphate ore typically occurs in thin, marine sediment sequences in these deposits and the phosphate itself is typically carbonaceous, consisting of pellets and nodules. The Miocene deposits along the southeastern coast of the United States are the best examples of deposits formed in warm climates. These deposits are economically minable if they have been reworked by submarine currents and/or subjected to weathering. The phosphate ore typically consists of phosphatic limestone or sandstone.

In igneous rocks, apatite usually occurs as intrusive masses or sheets, as hydrothermal veins or disseminated replacements, as marginal differentiations, or as pegmatites. Intrusive masses are the most common occurrence of igneous apatite, usually associated with alkaline igneous rocks including carbonatite, ijolite, nepheline syenite, and pyroxenite. Examples of these types of deposits are in the Kola Peninsula in the U.S.S.R., Palabora complex in the Republic of South Africa, and the carbonatites of southern Brazil.

A final type of phosphate deposit is the island phosphate deposits that are formed through the large accumulation of guano from sea birds. The composition of these deposits varies with the degree of leaching by surface waters. Decomposed guano is primarily calcium phosphates. The deposits on Nauru and Christmas Islands in the Pacific and Indian Oceans, respectively, are the best examples of guano deposits.

RESOURCES

Demonstrated world resources in terms of recoverable phosphate rock are estimated to be nearly 36.6 billion mt (table 1) for market economy and centrally planned economy countries (MEC's and CPEC's). As shown on figure 1, north Africa accounts for 60 pct of the total (22 billion mt) followed

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

by the United States with 17 pct (6 billion mt). An additional 20 billion mt of phosphate rock is potentially recoverable at the inferred level from the deposits analyzed for this study (not shown on the table). Figure 1 also illustrates the reserve base value for world phosphate (*4*), which is slightly lower than the resource value used in this study, because potential operations having production costs exceeding \$100/mt phosphate rock have not been included in the base.

In addition to the demonstrated resources evaluated for U.S. deposits in this study, an estimated 7 billion mt of potentially recoverable phosphate rock exists at the inferred level (over 80 pct in the southeast), and over 24 billion mt exists at the hypothetical resource level (over 60 pct in the southeast). Also, new deposits will likely be discovered (particularly offshore deposits along the eastern seaboard); low-grade material could become economically minable; or technological advances could enable economic processing of high-magnesium oxide material or the mining of deep deposits. Each of these factors could greatly increase the amount of domestic phosphate available in the future, although most likely at higher cost levels.

Table 2 is a list of the properties evaluated in this study.

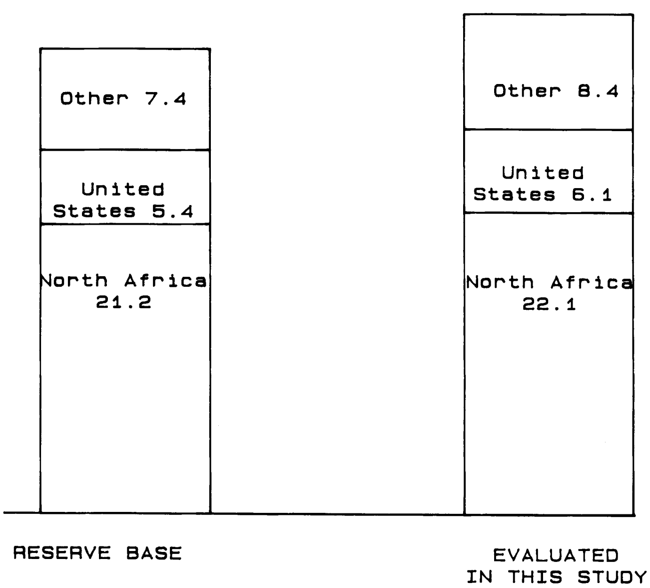

Figure 1.—Estimates of world phosphate rock resources (billion metric tons).

Table 1.—Summary of world demonstrated phosphate resources, as of January 1985

Region and country	In situ ore tonnage, 10^6 mt	In situ grade, wt pct P_2O_5	Rock product: Recoverable, 10^6 mt	Rock product: Grade, wt pct P_2O_5
MEC's				
North America:				
Canada and Mexico	1,385	8	199	34
United States	26,625	9	6,104	30
Total or av	28,010	9	6,303	30
North Africa:				
Morocco and Western Sahara	39,005	28	21,559	31
Tunisia and Algeria	1,247	22	545	31
Total or av	40,252	28	22,104	31
Middle East:				
Egypt	1,755	26	1,006	28
Israel	357	26	190	32
Jordan	1,169	26	511	33
Syria	447	24	204	30
Iraq, Saudi Arabia, and Turkey	739	21	304	33
Total or av	4,467	25	2,215	30
Oceania:				
Australia (including Christmas Island)	1,588	18	611	33
Nauru	22	38	14	39
Total or av	1,610	18	625	33
South America:				
Brazil	2,590	9	387	34
Colombia, Peru, and Venezuela	2,613	10	415	30
Total or av	5,203	9	802	32
West Africa: Senegal and Togo	834	27	237	34
Southern Africa:				
Angola and Zimbabwe	39	16	11	34
South Africa, Rep. of	21,426	6	2,544	37
Total or av	21,465	6	2,555	37
East Africa: Uganda	186	12	35	42
Europe: Finland	1,120	6	114	37
Asia: India, Pakistan, and Sri Lanka	107	25	65	32
MEC total or av	103,254	17	35,055	31
CPEC's[1]				
China	337	26	208	28
U.S.S.R.	5,024	15	1,333	33
CPEC total or av	5,361	16	1,541	32
World total or av	108,615	17	36,596	31

[1] Values not updated from previous world study owing to lack of new information; they remain in terms of January 1981.

Table 2.—MEC phosphate properties included in this study

Country and deposit name	Owner	Current status[1]	Mine method[2]	Mill method[3]	Deposit type[4]	Initial year of production[5]	Feed grade, wt pct	Mill capacity,[6] 10^3 t	Product: Grade, wt pct	Product: Capacity,[6] 10^3 t
Algeria: Djebel Onk	Algerian Government (SONEREM).	P	OP	S	SED	1965	24.6	3,500	31.8	2,000
Angola: Lacunga River	Angolan Government (Fosfang).	E	SL	Wa	SED	—	19.4	500	34.0	200
Australia:										
Christmas Island	Phosphate Mining Co. of Christmas Island.	P	SL	S	SED	1965	33.4	2,200	36.0	1,500
D-Tree	IMC Development Corp.	E	SL	F	SED	—	18.6	8,600	34.0	4,000
Duchess	Western Mining Corp.	T	SL	S	SED	1975	18.4	200	31.0	200
Lady Annie, Lady Jane	..do	E	SL	F	SED	—	17.0	9,400	34.0	4,000
Laverton	Australian Govt. and Union Oil.	E	OP	F	SED	—	20.0	2,100	38.0	1,000
Northern Deposits	Western Mining Corp.	E	SL	F	SED	—	15.1	7,900	34.0	3,000
Brazil:										
Anitapolis	ILM Ind. Luchsinger Madorin	D	SL	F	IGN	1986	6.5	3,800	35.1	600
Araxa-Arafertil	Arafertil	P	OP	F	IGN	1977	14.0	3,100	36.0	500
Araxa-Camig	..do	P	OP	F	IGN	1976	14.0	900	26.3	200
Catalao-Fosfago	Fosfago de Goias, S.A.	P	OP	F	IGN	1980	10.0	2,800	38.0	500
Catalao-Goiasfertil	Goias Fertilizantes	P	OP	F	IGN	1976	8.9	4,400	38.0	600
Ipanema I & II	Serrana, S.A.	E	OP	F	IGN	—	7.0	4,100	38.0	500
Itataia	Nuclebras	D	SL	F	HYD	—	11.0	2,000	33.0	500
Jacupiranga	Serrana, S.A.	P	OP	F	IGN	1965	5.0	4,300	35.0	500
Olinda-Paulista, Igarassu	FOSPERSA	E	SL	F	SED	—	20.0	700	28.0	300
Patos de Minas	Fertilizantes Fosfatos, S.A.	P	SL	F	SED	1977	13.0	3,200	30.0	1,000
Tapira	Fosfertil, S.A.	P	OP	F	IGN	1979	7.3	10,000	36.0	1,200
Canada:										
Cargill	Sherritt Gorden Mines	E	OP	F	IGN	—	20.0	1,900	39.0	700
Martison Lake	Campbell, New Venture	E	OP	F	IGN	—	17.5	1,300	39.0	400
Columbia: Pesca, Conejera, Sardinata.	Columbian Government (Ecominas).	PP	R&P	F	SED	—	18.0	400	29.0	100
Egypt:										
Abu Tartur	Arab Republic of Egypt	E	LW	F	SED	—	25.3	3,600	31.0	1,500
Hamrawein	Misr Phosphates	P	R&P	Wa	SED	1978	10.1	1,200	30.0	600
Quseir	Red Sea Phosphate Co.	P	R&P	Wa	SED	1980	21.0	200	26.0	100
Sabaiya East	Abu Zaabel Fertilizer Chemical Co.	P	R&P	S	SED	1977	26.0	100	26.0	100
Sabaiya West	..do	P	SL	F	SED	1977	22.0	900	28.0	500
Safaga	Red Sea Phosphate Co.	P	R&P	S	SED	1979	26.0	200	29.0	200
Finland:										
Siilinjarvi	Kemira Oy (Government of Finland).	P	OP	F	IGN	1980	4.0	6,000	35.9	500
Sokli	Rautaruukki Oy	E	OP	F	IGN	—	18.0	1,200	38.0	500
India: Jhamarkotra	Rajasthan State Mines & Minerals, Ltd.	P	OP	F	SED	1970	20.9	1,700	31.7	1,500
Iraq: Akashat	State Organization for Minerals.	P	SL	C	SED	1981	21.0	3,400	31.8	1,700
Israel:										
Arad	Negev Phosphates Ltd.	P	SL	Wa	SED	1970	27.0	900	32.6	500
Beersheba	..do	E	SL	Wa	SED	—	26.8	3,200	32.0	2,000
Nahal Zin	..do	P	SL	Wa	SED	1978	26.8	3,200	32.0	2,000
Oron	..do	P	OP	Wa	SED	1965	23.5	1,000	34.3	500
Jordan:										
El Hasa, El Abiad	Jordan Phosphate Mining Co.	P	SL	Wa	SED	1970	29.8	9,400	33.7	5,700
Esh-Shidiyah	..do	D	SL	F	SED	1987	25.4	3,800	33.4	2,000
Ruseifa	..do	P	SL	Wa	SED	1965	25.0	1,750	32.0	1,000
Mexico:										
San Juan de la Costa	Government of Mexico (ROFOMEX).	P	OP	F	SED	1980	18.0	2,000	31.0	800
Santo Domingo	..do	D	DR	F	SED	1990	4.5	11,200	30.0	1,200
Morocco:										
Ben Guerir	Government of Morocco (OCP).	P	SL	Wa	SED	1981	24.0	5,200	30.2	3,000
Daoui-Recette 4	..do	P	SL	Wa	SED	1971	26.0	17,000	32.1	10,000
Khouribga Underground	..do	P	R&P	S	SED	1965	29.0	2,400	32.3	1,600
Meraa El Arech	..do	P	SL	S	SED	1965	25.0	9,600	32.1	5,000
Meskala District	..do	E	SL	Wa	SED	—	29.7	2,800	31.0	2,000
Recette-10	..do	E	SL	Wa	SED	—	25.0	4,200	32.1	2,500
Sidi Chennane	..do	E	SL	Wa	SED	—	25.0	3,900	32.0	2,000
Sidi Hajjaj	..do	E	SL	Wa	SED	1990	24.0	4,300	31.0	2,000
Youssoufia Black Rock	..do	P	LW	S	SED	1974	27.5	6,300	34.0	3,600
Youssoufia White Rock	..do	P	R&P	S	SED	1980	29.0	6,400	31.0	4,000
Nauru: Nauru	Nauru Phosphate Corp.	P	SL	S	SED	1965	38.4	2,900	39.4	2,000

See footnotes at end of table.

Table 2.—MEC phosphate properties included in this study—Continued

Country and deposit name	Owner	Current status[1]	Mine method[2]	Mill method[3]	Deposit type[4]	Initial year of production[5]	Feed grade, wt pct	Mill capacity,[6] 10^3 t	Product Grade, wt pct	Product Capacity,[6] 10^3 t
Pakistan:										
Kakul-Hazara	Sarhad Development Authority.	D	C&F	S	SED	1986	28.5	60	33.0	40
Lagarban-Hazara	..do	D	C&F	S	SED	1990	27.0	400	34.0	200
Peru: Bayovar	Probayovar S.A.	E	SL	Wa	SED	—	7.8	7,400	30.5	1,500
Saudi Arabia: West Thaniyat	Govt. of S. Arabia, Granges Intl.	E	LW	S	SED	—	22.0	5,000	35.0	2,500
Senegal:										
Pallo (Thies)	Senegal Government, Rhone-Poulenc.	P	SL	S	SED	1965	28.0	720	30.0	600
Taiba-Tobene	Senegal Government, BRGM/IMC.	P	SL	F	SED	1965	26.9	6,400	33.4	1,500
South Africa, Rep. of: Palabora (Foskor)	Foskor (Phosphate Development).	P	OP	F	IGN	1965	6.0	23,900	36.5	3,200
Sri Lanka: Eppawella	Sri Lankan Govt. (SMMDL), Agrico.	E	OP	S	SED	—	32.0	900	32.0	900
Syria:										
Kneifess	Gecopham (General Co. of Phosphate).	P	OP	S	SED	1971	28.0	1,000	32.0	600
Sharkya A	..do	P	OP	S	SED	1974	24.0	1,400	30.0	800
Sharkya B	..do	P	OP	S	SED	1974	26.0	1,300	30.0	800
Togo:										
Dagbat	Togo Government (COTOMIB).	E	OP	Wa	SED	—	30.0	2,700	36.0	1,500
Hahotoe, Kpogame	..do	P	SL	Wa	SED	1965	28.0	6,000	36.0	3,200
Tunisia:										
Djellabia	CIE Des Phosphates De Gafsa (Govt.)	D	SL	S	SED	1992	25.5	2,300	30.2	1,500
Kalaa Khasba	..do	P	R&P	W	SED	1980	27.0	300	30.0	200
Kef Eddour	..do	D	SL	S	SED	1987	25.5	2,150	30.2	1,400
Kef Eschfair	..do	P	SL	Wa	SED	1972	25.5	1,950	30.2	1,300
M'Dilla	..do	P	R&P	Wa	SED	1965	25.5	750	30.2	500
Metlaoui	..do	P	R&P	Wa	SED	1975	25.5	800	30.2	500
Moulares	..do	P	R&P	Wa	SED	1965	25.0	1,300	30.2	800
M'Rata	..do	P	R&P	Wa	SED	1965	25.5	1,300	30.2	900
Oum El Khecheb	..do	D	SL	S	SED	1990	25.5	450	30.2	300
Redeyef	..do	P	R&P	S	SED	1965	25.0	1,000	28.0	700
Sehib	..do	P	LW	Wa	SED	1978	25.5	2,100	30.2	1,400
Sra Ouertane	..do	D	SL	F	SED	1992	15.0	3,800	29.0	1,200
Turkey: Mardin-Mazidag	Etibank	P	CF	F	SED	1985	18.0	300	30.0	100
Uganda: Sukulu Hills	Sukulu Mines, Ltd.	D	OP	F	IGN	1989	11.7	400	42.0	100
United States:										
Florida:										
Acrefoot Johnson	Freeport Phosphate Mining Co.	E	SL	F	SED	—	5.1	16,800	30.0	2,700
Big Four	AMAX Inc.	T	SL	F	SED	1977	W	W	W	W
Boyette and Fishawk	Agrico Chemical Co.	E	SL	F	SED	—	7.4	3,800	33.9	700
Brooker-Dukes	Kerr-McGee Chemical Corp.	E	SL	F	SED	—	6.5	13,870	31.6	2,700
C.F. Hardee Phosphate Complex.	C.F. Industries Inc.	P	SL	F	SED	1978	W	W	W	W
Christina	Mobil Chemical Co.	E	SL	F	SED	—	8.0	2,100	33.4	500
Clear Springs	IMC Corp.	P	SL	F	SED	1948	W	W	W	W
Cooks Hammock #1	Monsanto Co.	E	SL	F	SED	—	4.3	4,200	33.0	500
Cooks Hammock #2	Unidentified major paper company.	E	SL	F	SED	—	4.3	3,600	32.0	500
David C. Turner Heirs	Heirs of D.C. Turner	E	SL	F	SED	—	4.7	3,500	30.2	500
Deep Creek	Occidental Chemical Co.	E	SL	F	SED	—	4.8	15,400	31.4	1,800
Deseret Ranch	Mormon Church	E	SL	F	SED	—	5.5	42,000	28.6	5,500
De-Soto-Manatee Reserve.	AMAX Inc.	E	SL	F	SED	—	6.1	18,100	30.7	3,400
Duette	ESTECH	E	SL	F	SED	—	5.2	15,950	31.0	2,700
Durrance, Waters Tract	U.S.S. Agri-Chemicals	E	SL	F	SED	—	5.5	5,100	30.7	900
Farmland Hardee	Farmland Industries Inc.	E	SL	F	SED	—	7.6	6,800	31.2	1,800
First Mississippi Chemical Tract.	First Mississippi Chemical Corp.	E	SL	F	SED	—	6.5	2,800	33.3	500
Fort Green	Agrico Chemical Co.	P	SL	F	SED	1975	W	W	W	W
Fort Meade #1	Mobil Chemical Co.	P	SL	F	SED	1945	W	W	W	W
Fort Meade #2	Gardinier Inc.	P	SL	F	SED	1967	W	W	W	W
Four Corners	W.R. Grace Co. and IMC Corp.	P	SL	F	SED	1985	W	W	W	W
Fridovich	Agri-Lewis Corp.	E	SL	F	SED	—	7.1	2,500	32.6	500
Hardee	First Mississippi Chemical Corp.	E	SL	F	SED	—	5.2	17,800	30.7	2,700
Hardee West	Various ownerships	E	SL	F	SED	—	4.3	7,800	30.3	900
Haynesworth	Brewster Phosphates	P	SL	F	SED	1965	W	W	W	W

See footnotes at end of table.

Table 2.—MEC phosphate properties included in this study—Continued

Country and deposit name	Owner	Current status[1]	Mine method[2]	Mill method[3]	Deposit type[4]	Initial year of production[5]	Feed grade, wt pct	Mill capacity,[6] 10³ t	Product Grade, wt pct	Product Capacity,[6] 10³ t
United States—Continued										
Florida—Continued										
Hillsborough Co.-Farmland Brewster.	Pruitt, Thompson, Jameson, Simms.	E	SL	F	SED	—	5.7	3,200	33.3	500
Hookers Prairie	W.R. Grace Co.	P	SL	F	SED	1976	W	W	W	W
Hopewell	Noranda	P	SL	F	SED	1985	W	W	W	W
Keys	IMC Corp.	E	SL	F	SED	—	4.0	25,000	29.6	2,300
Kingsford	..do	P	SL	F	SED	1965	W	W	W	W
La Crosse	Kerr-McGee Chemical Corp.	E	SL	F	SED	—	6.2	2,800	31.6	500
Little Payne Creek	U.S.S. Agri-Chem, Gardinier, others.	E	SL	F	SED	—	7.3	9,300	30.9	2,700
Lonesome	Brewster Phosphates	P	SL	F	SED	1976	W	W	W	W
Manatee North	W.R. Grace Co.	E	SL	F	SED	—	5.6	17,700	29.9	2,700
Manatee South	..do	E	SL	F	SED	—	4.2	31,400	30.2	3,600
Manson-Jenkins	U.S.S. Agri-Chemicals	E	SL	F	SED	—	5.1	5,800	30.5	900
Mobil Area	Various ownerships	E	SL	F	SED	—	4.8	7,700	30.9	1,100
N.E. Manatee Swift, Grace.	W.R. Grace Co. and others	E	SL	F	SED	—	4.7	3,500	30.4	500
Nichols	Mobil Chemical Co.	P	SL	F	SED	1970	W	W	W	W
No. Columbia County #2	So. Resin Corp.	E	SL	F	SED	—	3.9	17,400	28.4	1,800
Noralyn, Phosphoria	IMC Corp.	P	SL	F	SED	1948	W	W	W	W
North Lake City	Kerr-McGee Chemical Corp.	E	SL	F	SED	—	6.1	2,600	31.1	500
Northeast Manatee, Texaco.	Various ownerships	E	SL	F	SED	—	3.8	8,800	30.7	900
Osceola National Forest	U.S. Forest Service	E	SL	F	SED	—	4.8	13,000	31.2	1,800
Payne Creek	Agrico Chemical Co.	P	SL	F	SED	1966	W	W	W	W
Pierce, Pebbledale	..do	E	SL	F	SED	—	5.1	3,100	31.4	500
Pine Level	Amax Inc.	E	SL	F	SED	—	6.1	6,750	30.7	1,400
Rockland	U.S.S. Agri-Chemicals and Freeport Phosphate Co.	P	SL	F	SED	1968	W	W	W	W
Rutland-Colvin-Vale	IMC Corp.	E	SL	F	SED	—	3.1	13,100	29.8	1,100
Saddle Creek	Agrico Chemical Co.	P	SL	F	SED	1950	W	W	W	W
Sarasota County No. 1	George Kelce	E	SL	F	SED	—	2.9	16,100	27.8	1,400
SE Hillsborough Reserves.	IMC Corp.	E	SL	F	SED	—	W	W	W	W
Silver City	Estech	P	SL	F	SED	1964	W	W	W	W
South Fort Meade	Mobil Chemical Co. and others.	D	SL	F	SED	1990	W	W	W	W
South Hardee	Gardinier Inc.	E	SL	F	SED	—	4.4	15,100	29.9	2,300
Suwannee River	Occidental Chemical Co.	P	SL	F	SED	1965	W	W	W	W
Swift Creek	..do	P	SL	F	SED	1975	W	W	W	W
Swift, Durrance Area	Various ownerships	E	SL	F	SED	—	5.2	3,500	31.0	500
Texaco Manatee	Texaco Inc.	E	SL	F	SED	—	6.1	8,700	30.7	1,400
Waters Tract	U.S.S. Agri-Chemicals	E	SL	F	SED	—	4.2	6,500	29.6	900
Watson	Estech	P	SL	F	SED	1936	W	W	W	W
Wingate Creek	Beker Industries	P	SL	F	SED	1982	W	W	W	W
Zolfo Springs Area small ownerships.	Mining Development Corp.	E	SL	F	SED	—	7.0	6,000	30.4	1,100
Zolfo, Stauffer	Stauffer Chemical Co.	E	SL	F	SED	—	5.9	6,000	32.0	900
Idaho:										
Diamond Creek	Alumet Corp.	E	OP	C	SED	—	W	W	W	W
Dry Valley	J.R. Simplot Co. and FMC Corp.	D	OP	Wa	SED	1987	W	W	W	W
Enoch Valley	Monsanto Co.	D	OP	Wa	SED	1991	W	W	W	W
Gay Mine	J.R. Simplot Co. and FMC Corp.	P	OP	Wa	SED	1946	W	W	W	W
Henry, N. Henry	Monsanto Co.	P	OP	Wa	SED	1952	W	W	W	W
Lanes Creek	Alumet Corp.	P	OP	C	SED	1978	W	W	W	W
Maybe Canyon, Champ, others.	Conda Partnership	P	OP	C	SED	1966	W	W	W	W
Rasmussen Ridge	..do	D	OP	Wa	SED	1994	W	W	W	W
Smokey Canyon	J.R. Simplot Co.	P	OP	Wa	SED	1920	W	W	W	W
Wolley Valley	Stauffer Chemical Co.	P	OP	Wa	SED	1955	W	W	W	W
Montana: Warm Springs Creek.	Cominco American Inc.	P	R&P	S	SED	1929	W	W	W	W
North Carolina:										
Canvas Creek	North Carolina Phosphate Corp.	D	SL	F	SED	1989	W	W	W	W
Lee Creek	Texasgulf Chemical Corp.	P	SL	F	SED	1966	W	W	W	W
Tennessee:										
Hooker Chemical Properties.	Hooker Chemical Co.	P	SL	Wa	SED	1953	W	W	W	W
Monsanto Properties	Monsanto Co.	P	SL	Wa	SED	1938	W	W	W	W
Stauffer Chemical Co. Property.	Stauffer Chemical Co. and others.	P	SL	Wa	SED	1896	W	W	W	W
Tennessee Valley Authority.	Tennessee Valley Authority	PP	SL	Wa	SED	—	W	W	W	W

See footnotes at end of table.

Table 2.—MEC phosphate properties included in this study—Continued

Country and deposit name	Owner	Current status[1]	Mine method[2]	Mill method[3]	Deposit type[4]	Initial year of production[5]	Feed grade, wt pct	Mill capacity,[6] 10^3 t	Product Grade, wt pct	Product Capacity,[6] 10^3 t
United States—Continued										
Utah:										
Central Wasatch Range #1.	Public land, unleased	E	OP	F	SED	—	24.1	300	28.7	200
Central Wasatch Range #2.	..do	E	OH	F	SED	—	20.1	1,250	28.8	700
Crawford Mountains #1	Stauffer Chemical Co.	E	OP	F	SED	—	23.7	1,000	28.4	800
Crawford Mountains #2	..do	E	OP	F	SED	—	23.7	1,000	28.4	800
Crawford Mountains #3	..do	E	R&P	F	SED	—	19.5	250	26.0	100
Crawford Mountains #4	..do	E	R&P	F	SED	—	26.7	1,250	33.0	800
Crawford Mountains #5	..do	E	R&P	F	SED	—	20.5	250	28.7	100
Flaming Gorge #1	Public land, unleased	E	OP	F	SED	—	23.1	1,000	29.0	700
Flaming Gorge #2	..do	E	R&P	F	SED	—	20.4	1,250	26.0	900
Flaming Gorge #3	..do	E	R&P	F	SED	—	19.4	250	26.0	200
Northern Wasatch Range	..do	E	OP	F	SED	—	26.3	1,000	32.4	700
Vernal Field #1	U.S. Steel	E	OP	F	SED	—	20.6	1,000	26.0	700
Vernal Field #2	..do	E	OP	F	SED	—	20.8	1,000	26.0	700
Vernal Field #3	..do	E	R&P	C	SED	—	19.1	1,250	26.0	800
Vernal Field #4	..do	E	R&P	C	SED	—	16.9	2,500	26.0	1,300
Vernal Field #5	..do	E	R&P	C	SED	—	17.1	1,250	26.0	700
Vernal Mine	Chevron Resources	P	OP	F	SED	1961	W	W	W	W
Wyoming:										
Gros Ventre Range #1	Public land, unleased	E	OP	F	SED	—	25.5	1,000	33.0	700
Gros Ventre Range #2	..do	E	OH	C	SED	—	20.9	5,000	26.4	3,500
Hoback Range #1	..do	E	OP	F	SED	—	21.5	1,000	28.0	700
Hoback Range #2	..do	E	OP	F	SED	—	20.9	1,000	26.0	700
Hoback Range #3	..do	E	OH	F	SED	—	19.6	1,250	26.0	800
S.E. Wind River Range #1.	..do	E	OP	F	SED	—	20.2	1,000	26.0	700
S.E. Wind River Range #2.	..do	E	R&P	C	SED	—	18.1	2,500	26.0	1,400
Salt River Range #1	..do	E	OP	F	SED	—	24.9	1,000	30.2	800
Salt River Range #2	..do	E	OH	F	SED	—	18.3	2,500	26.0	1,300
Salt River Range #3	..do	E	OH	F	SED	—	24.6	1,250	33.0	600
Snake River #1	..do	E	OP	F	SED	—	25.9	1,000	30.4	800
Snake River #2	..do	E	R&P	F	SED	—	22.3	5,000	29.2	2,900
Snake River #3	..do	E	OP	S	SED	—	24.5	1,000	30.7	700
Snake River #4	..do	E	R&P	F	SED	—	20.4	1,250	30.3	500
Snake River #5	..do	E	R&P	F	SED	—	24.3	250	33.0	100
South Ridges #1	..do	E	OP	F	SED	—	23.2	1,000	28.8	700
South Ridges #2	..do	E	OH	F	SED	—	19.6	1,250	29.0	500
South Ridges #3	..do	E	OH	F	SED	—	22.9	5,000	27.2	3,000
Sublette Range #1	..do	E	OP	F	SED	—	23.1	1,000	27.6	800
Sublette Range #2	..do	E	OH	F	SED	—	20.0	1,250	28.1	600
Tunp #1	..do	E	OP	F	SED	—	23.6	1,000	28.7	700
Tunp #2	..do	E	OP	F	SED	—	24.5	1,000	28.7	800
Tunp #3	..do	E	OH	F	SED	—	22.5	250	33.0	100
Tunp #4	..do	E	OH	F	SED	—	19.8	250	26.0	200
Wyoming Range #1	..do	E	OP	F	SED	—	26.4	1,000	32.2	700
Wyoming Range #2	..do	E	R&P	F	SED	—	21.3	5,000	27.8	3,000
Venezuela: Riecito	Petroquimica de Venezuela	PP	OP	F	SED	—	26.1	750	31.8	400
Western Sahara: Bou Craa	Governments of Morocco and Spain.	P	SL	Wa	SED	1973	31.5	5,000	36.0	3,000
Zimbabwe: Dorowa	African Explosive & Chemical Ind.	P	OP	F	IGN	1965	6.4	1,800	35.0	200

W Withheld to avoid disclosing company proprietary data.
[1] As of January 1985: P, producer; D, developing; E, explored; PP, past producer; T, temporarily shut down.
[2] SL, strip level; OP, open pit; R&P, room and pillar; DR, dredge; OH, overhand stope; LW, longwall; C&F, cut and fill.
[3] F, flotation; Wa, wash; C, calcination; S, size.
[4] SED, sedimentary; IGN, igneous; HYD, hydrothermal.
[5] Dash indicates no definite startup date or no production as of 1985. Startup date for any developing deposit is based on company projections.
[6] Capacities are either actual or assumed and represent the capacity in the 4th year of production from 1985 or startup.

U.S. AND WORLD HISTORICAL PRODUCTION

Phosphate rock production in 1985 was estimated to be 159 million mt from 33 countries. Slightly over 70 pct was from MEC's. The United States, Morocco, and the U.S.S.R. accounted for over two-thirds of 1985 production. Figure 2 shows graphically world phosphate rock production in 5-yr intervals from 1950 to 1985. Since 1950, world phosphate rock production has increased sixfold, with the three largest producers (the United States, Morocco, and the U.S.S.R.) approximately maintaining their shares until recent years when other countries have been increasing their production levels. The U.S. production, although continuing to lead the world, has dropped from nearly 54 million mt in 1981 to 37 million mt in 1982. Since then, domestic production has increased gradually and was estimated at 51 million mt in 1985 (*4*).

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

Nearly 90 pct of the phosphate rock product produced in MEC's today is recovered by surface mining methods. The remaining 10 pct is recovered by underground methods, predominantly in Morocco and Tunisia. In the U.S.S.R. and China, approximately 30 and 23 pct, respectively, of the phosphate rock product is recovered by underground mining methods. Table 2 shows mining and milling methods for the deposits and mines included in this study.

The two major surface-mining methods used in the phosphate industry are strip mining and open-pit mining. A third method, dredging, is used in special situations. Strip mining accounts for 90 pct of U.S. and about 60 pct of total world phosphate rock production. MEC production by this method is over 70 pct. Strip mining is predominantly used because of the tabular, bedded, sedimentary nature of most phosphate deposits. Most deposits in the southeastern United States and north Africa use this method.

Over 15 pct of current MEC phosphate-rock production is supplied by the open-pit mining method. Open-pit mining is extensively employed to exploit the massive igneous phosphate carbonatites. The U.S.S.R. and China derive about 70 and 30 pct of their respective production from igneous sources. Dredging is employed at a few deposits throughout the world, particularly at the Wingate Creek operation in Florida (United States) and will be employed at the proposed Santo Domingo operation in Mexico. It is also used to strip overburden at Texasgulf's Lee Creek Mine in North Carolina (United States).

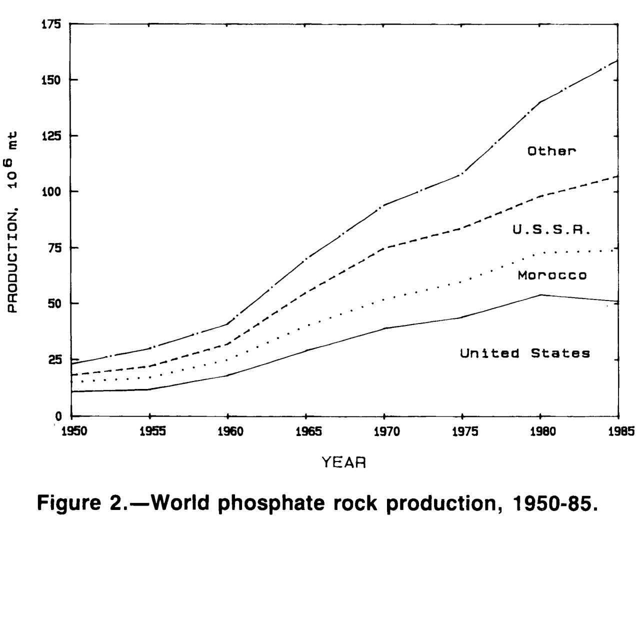

Figure 2.—World phosphate rock production, 1950-85.

The relatively low unit value of phosphate rock makes underground mining methods generally unprofitable. However, steeply dipping phosphate beds, or high stripping ratios, sometimes make the use of underground mining techniques preferable. In such cases, highly mechanized room-and-pillar, longwall caving, and overhand-stoping methods have been used successfully. While only just over 10 pct of present phosphate rock production capacity in MEC's is from underground methods, this study estimates that nearly 20 pct of the capacity of deposits not producing at the time of the study could be from underground mines. The majority of producing underground phosphate mines are located in northern Africa.

PROCESSING

In almost all cases the run-of-mine phosphate material has to be beneficiated. The basic beneficiation methods employed in the phosphate industry are sizing, washing, flotation, calcining, and drying. A phosphate beneficiation plant may use one or more of these methods to produce a marketable product.

The milling method assigned to properties in this study (as shown in table 2) indicates the most significant method used to beneficiate the phosphate material. An example would be a property that screens and washes before sending the phosphate material through a flotation circuit. The milling method for this property would be listed as flotation, even though sizing and washing were used. In the United States, nearly 90 pct of current phosphate rock product capacity is beneficiated through flotation, with calcining and washing accounting for the remainder. Nearly 50 pct of current world production is beneficiated through flotation, with washing, sizing, and calcining making up the rest.

Average feed grade, average product grade, and average mill recovery are shown in table 3 for the various MEC regions. Feed grade is here defined as the recoverable grade of the ore that feeds the mill. The table shows that the southeastern U.S. operations have the highest milling recovery. This is primarily due to flotation being incorporated in their processing. North Africa, on the other hand, has a lower mill recovery, which is due to losses of fine material during washing.

Table 3.—Phosphate mill plant operating parameters,[1] by region

Region	Producing mines: Grade, pct P_2O_5 Feed	Producing mines: Grade, pct P_2O_5 Product	Producing mines: Recovery, pct	Nonproducing deposits: Grade, pct P_2O_5 Feed	Nonproducing deposits: Grade, pct P_2O_5 Product	Nonproducing deposits: Recovery, pct
United States:						
Southeast	10.9	30.4	79.0	5.7	30.5	79.6
West	22.2	31.2	71.3	21.3	27.7	80.9
North Africa	26.0	32.2	69.8	24.5	31.5	67.4
West Africa	27.1	33.3	40.5	W	W	W
Middle East	24.7	31.4	71.4	24.9	32.6	64.3
Australia	W	W	W	17.5	34.0	86.5
South America	9.6	33.8	58.6	9.1	31.8	69.9

W Withheld to avoid disclosing individual deposit data.

[1] Feed grade, product grade, and recovery are weighted averages for all the deposits in each region.

NEW TECHNOLOGY

The following briefly discusses some potential technological advances that could enhance or permit phosphate recovery from phosphate deposits (particularly in the United States). These new processes, if successful, would increase phosphate resource potential, although most likely at higher costs.

The Bureau and various phosphate companies in Florida have in recent years been researching ways to recover phosphate rock from deposits containing high amounts of magnesium oxide (MgO). In phosphate rock, an MgO content of 1 pct or more causes problems in the manufacture of phosphoric acids and is therefore unacceptable to acid producers. If beneficiation technology is developed and proven in this area, as much as 2 billion mt of phosphate rock (at the identified resource level) in Florida alone could become available, but likely at higher costs.

Two important mining technologies are also being researched in relation to improved phosphate recovery. A technique to recover deep phosphate resources, called borehole mining, has been developed and demonstrated, although not on a commercial scale. In addition, mining of offshore phosphate resources has potential, although it has not as yet been developed into a viable economic method. Both of these mining techniques would substantially increase phosphate rock recovery and resource potential, but most likely would have higher costs than present mining systems.

PRODUCTION COSTS

Table 4 and figure 3 illustrate the average production costs for selected surface operations included in this study (expressed in January 1985 U.S. dollars per metric tons of product). These costs are in constant dollar terms and are based on current technology. As technology changes in the future, costs for these operations, particularly those with long lives, will also change.

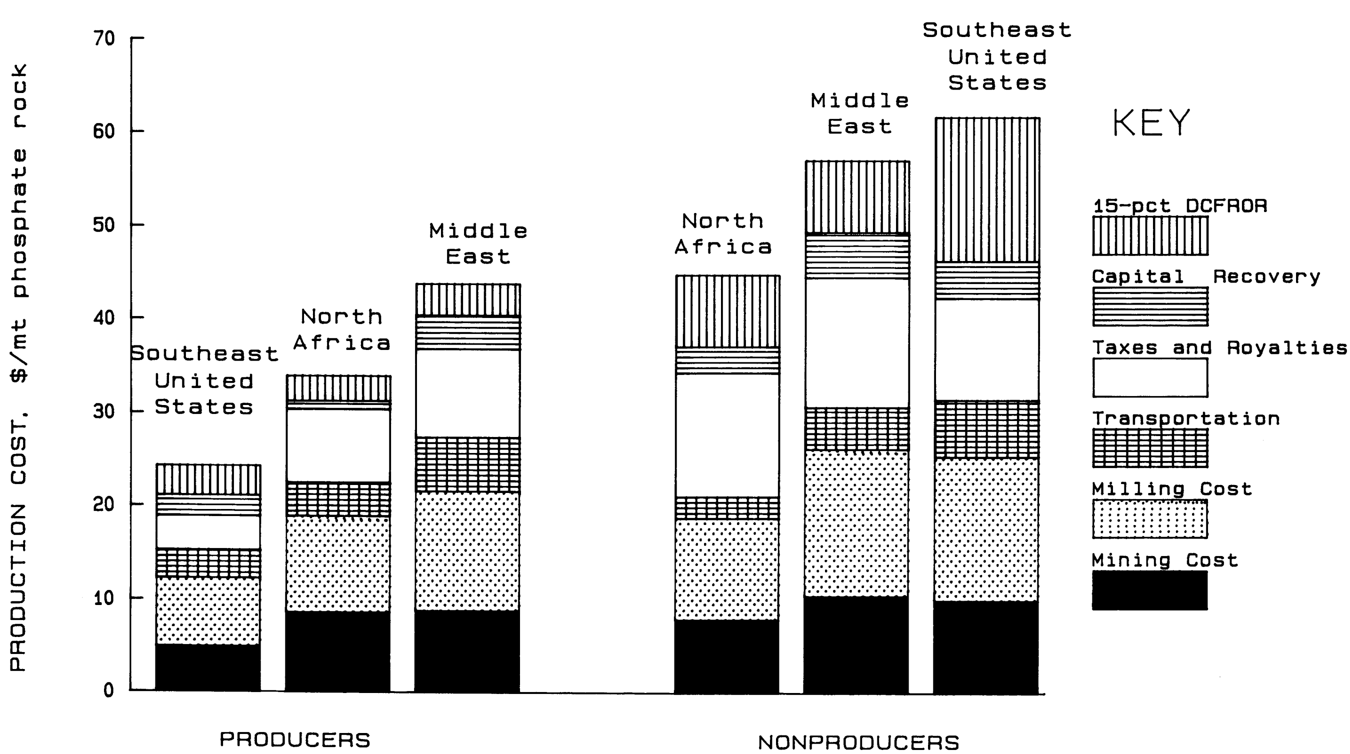

Figure 3.—Phosphate rock production costs for selected regions (January 1985 U.S. dollars).

Mining costs for producing surface mines average \$7.80/mt. Costs for the most significant producing regions outside of the United States (Morocco, Tunisia, Israel, Jordan, Senegal, and Togo) range from \$7.20/mt to \$9.50/mt, indicating a similarity in mining methods and stripping ratios. The mining cost in the southeast United States is somewhat lower than those in other regions, \$4.90/mt, primarily because of lower stripping ratios. Mining costs from deposits not yet developed in the southeast United States are estimated to be significantly greater (more than double that of producers), primarily because of the greater stripping ratios in the "southern extension" deposits (the most significant nonproducing deposits in the United States) as well as increased depth of the ore zones, which results in more material handling and greater lengths of transport.

Milling costs for producing surface mines average \$11/mt. Costs for southeastern U.S. producers are also somewhat less than those of producers in Morocco, Tunisia, Israel, Jordan, Senegal, and Togo (\$7.40/mt as compared with a range from \$10.30/mt to \$12.80/mt). This is primarily because the ore in the southeastern United States mines has a significant pebble fraction that requires little or no beneficiation and, therefore, producers encounter very little cost to recover that portion of the deposit. However, undeveloped deposits in the southern extension in Florida have a much smaller pebble fraction, if any, and therefore

Table 4.—Phosphate rock production costs for producing and nonproducing operations in selected MEC regions

(All costs are in January 1985 U.S. dollars per metric ton phosphate rock on a weighted-average basis)

Region and country	Operating cost: Mine	Operating cost: Mill	Transportation to plant or port[1]	Net operating cost	Recovery of capital[2]	0-pct DCFROR: Taxes and royalties[3]	0-pct DCFROR: Total cost[4]	15-pct DCFROR: Taxes and royalties[5]	15-pct DCFROR: Return on investment[6]	15-pct DCFROR: Total cost[7]
SURFACE OPERATIONS										
North America:										
United States:										
Southeast:[8]										
Producers	4.90	7.40	3.00	15.30	2.20	2.10	19.60	3.60	3.20	24.30
Nonproducers	10.00	15.40	6.10	31.50	4.00	3.10	38.60	10.90	15.40	61.80
West:										
Producers[9]	14.30	10.90	([10])	25.20	1.30	.90	27.40	1.50	1.50	29.50
Nonproducers[11]	20.20	14.70	12.10	47.00	5.10	1.60	53.70	8.70	13.70	74.50
North Africa: Morocco and Western Sahara:										
Producers	8.60	10.30	3.60	22.50	.90	5.00	28.40	7.90	2.70	34.00
Nonproducers	7.90	10.90	2.30	21.10	1.60	5.90	28.60	13.30	8.90	44.90
Tunisia and Algeria:										
Producers	9.50	12.40	7.70	29.60	2.80	7.50	39.90	11.70	3.50	47.60
Nonproducers	4.70	12.90	3.60	21.20	2.60	8.10	31.90	15.30	8.00	47.10
Middle East:										
Israel, Egypt, and Jordan:										
Producers	8.80	12.80	5.80	27.40	3.60	6.90	37.90	9.50	3.40	43.90
Nonproducers	10.50	15.70	4.50	30.70	4.90	8.60	44.20	13.90	7.70	57.20
Syria, and Iraq:										
Producers	19.30	18.60	6.20	44.10	4.40	7.50	56.00	13.70	3.90	66.10
Oceania:										
Christmas Island and Nauru: Producers	6.40	7.30	.00	13.70	2.20	8.20	24.10	9.00	1.50	26.40
Australia: Nonproducers	8.60	14.90	14.10	37.60	8.30	2.20	48.10	7.00	6.60	59.50
South America: Brazil, Peru, and Venezuela:										
Producers	6.00	15.60	6.70	28.30	4.60	4.70	37.60	10.70	9.40	53.00
Nonproducers	11.10	16.80	3.60	31.50	10.00	5.40	46.90	27.00	22.50	91.00
West Africa: Senegal and Togo: Producers	7.20	12.60	2.20	22.00	3.60	5.30	30.90	7.10	3.80	36.50
UNDERGROUND OPERATIONS										
North America:										
United States: Nonproducers[12]	49.10	36.00	14.00	99.10	5.30	2.10	106.50	14.20	28.70	147.30
North Africa:										
Morocco: Producers	18.50	17.30	4.30	40.10	.80	5.20	46.10	7.40	2.00	50.30
Tunisia: Producers	9.60	8.30	6.90	24.80	5.70	7.20	37.70	10.40	3.60	44.50
Middle East:										
Egypt: Producers	18.50	23.50	1.20	43.20	7.10	5.30	55.60	10.50	11.20	72.00

[1] Transportation costs to ports or acid plants that have been assumed as product destination points for this study.
[2] Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure, and reinvestments required over the life of the operation.
[3] Includes property, State, Federal, and severance taxes, and royalties, where applicable, calculated at a 0-pct DCFROR.
[4] Equal to the sum of net operating costs, taxation, and capital recovery determined at a 0-pct DCFROR.
[5] Includes property, State, Federal, and severance taxes, and royalties, where applicable, calculated at a 15-pct DCFROR.
[6] The revenue increase per ton necessary to obtain a 15-pct DCFROR.
[7] Equal to the sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery *plus* the per ton increase necessary to provide a 15-pct DCFROR after taxation.
[8] Includes Florida and North Carolina.
[9] Represents only Idaho.
[10] Transportation costs for Idaho included in mill costs.
[11] Includes Idaho, Utah, and Wyoming.
[12] Includes Montana, Utah, and Wyoming.

the milling costs would be much greater. Another reason the mining and milling costs are less in the southeast United States is the size of the operations. The average size of a producing mine in the southeast United States is over 9 million mt/yr feed to the mill, while Morocco, for example, is under 6 million mt/yr feed (not including the large Daoui Mine). This gives the mines in the southeast United States an economy of scale advantage over many other operations worldwide.

Transportation costs from mine to plant or port are in most cases small except in the western United States, where the rock in Utah and Wyoming is assumed to go to Idaho, and in Australia where the deposits are in the middle of Queensland and the rock must be transported to the coast.

Production costs are also shown for underground mines and deposits, mainly representing the producers in north Africa and the nonproducers in the western United States (Utah and Wyoming). When comparing these costs with surface mine costs, it is apparent, as would be expected, that the underground operations are much more expensive. In the case of the north African underground mines, even though they are more expensive to operate than the surface mines, they are near enough to a market that the mines would still be economical. The underground nonproducers in the western United States (Utah and Wyoming) would have costs much higher than the other phosphate deposits evaluated. This is largely due to the characteristics of the ore, coupled with the higher costs of underground mining. These highly uneconomical deposits are not likely to be developed in the near future.

AVAILABILITY

A few assumptions are inherent in the analysis. For purposes of consistency, in this analysis it was assumed that all rock produced at a mine was transported to a local port for export unless that rock was being used for internal domestic consumption. If internally consumed, the rock was transported to a nearby acid plant or market. Additional costs for further processing of phosphate rock into its many end products were not included in the evaluation.

TOTAL RECOVERABLE

A total of 206 mines and deposits in MEC's were analyzed, 124 domestic plus 82 foreign (see table 2). These mines and deposits account for most of the phosphate rock production and resources in MEC's. As of the begining of 1985, 82 of the mines evaluated were in production[2] (34 in the United States and 48 in foreign countries). A total of 124 nonproducing deposits were also evaluated (90 in the United States and 34 in foreign countries) including deposits that are developing or have definite developmental plans, as well as many deposits that are just explored prospects. Production costs for deposits in CPEC's were not estimated owing to a lack of information.

The tonnage of phosphate rock potentially available from the producing mines and nonproducing deposits analyzed in MEC's is shown in figure 4. Production costs for operating mines are bracketed by lower and upper cost levels. The lower level reflects costs at a 0-pct discounted-cash-flow rate of return (DCFROR) while the upper level reflects total cost including a 15-pct DCFROR. The cost curve for nonproducing deposits includes a 15-pct DCFROR. (These production cost levels are explained in the methodology section of this Bulletin.)

A total 35.1 billion mt of phosphate rock is potentially recoverable from the demonstrated resources of the MEC deposits evaluated. Of the total, 37 pct occurs in mines that were producing in 1985. At the 0-pct DCFROR level, a total of 11.9 billion mt is potentially recoverable from producing mines at a cost of under $50/mt. At a cost of $30/mt (also at a 0-pct DCFROR), which was approximately the average 1985 U.S. (Tampa) market price f.o.b. port (the price f.o.b. Casablanca was approximately $50/mt), total availability from producing mines would be 6.6 billion mt.

North Africa accounts for the largest share of phosphate on the curves. At a cost of $30/mt (0-pct DCFROR), north Africa accounts for 78 pct of the tonnage from producers and the United States accounts for 20 pct. For nonproducers, at a total cost (including a 15-pct DCFROR) of $65/mt, north Africa accounts for 43 pct of the total followed by the United States with 35 pct.

ANNUAL CAPACITY

Potential 1985 production at full capacity from operating U.S. and north African mines studied is shown of figure 5. These regions account for 74 pct of the 127 million mt of phosphate rock capacity potentially available from the MEC's in 1985. The U.S. mines studied had the capacity to produce an estimated 58.7 million mt and north African mines could have produced 34.9 million mt. Each

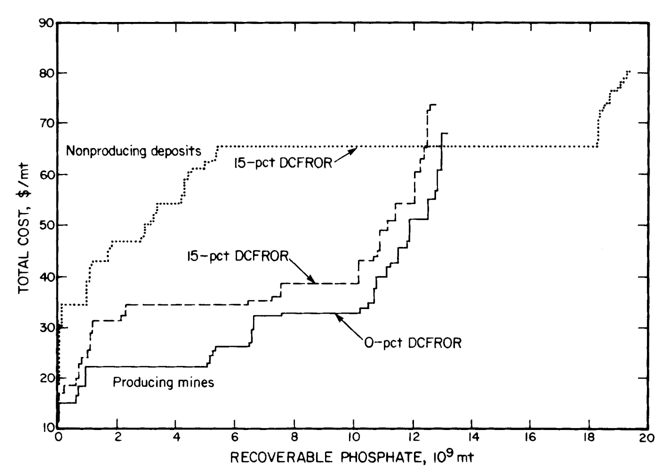

Figure 4.—Potential total phosphate rock available from MEC's (January 1985 U.S. dollars).

[2] Some mines, particularly in Florida, were included as producers even if temporarily shut down as of early 1985.

of the regions is bracketed by a lower operating cost level and an upper cost level that includes a 0-pct DCFROR. At operating costs roughly equivalent to the January 1985 price for phosphate rock in the United States of approximately $30/mt f.o.b. port, an estimated 54 million mt could have been mined from U.S. mines. At the Casablanca price of $50/mt, an estimated 34 million mt could have been mined from north African mines. These tonnages are close to the 1985 estimated production levels of 51 million mt for the United States and 30 million mt for north Africa.

Potential annual 1985-95 production of phosphate rock from the demonstrated resources of producing mines in MEC's, north Africa, the United States, and the Middle East is shown in figure 6 (note that each curve is constructed at a different scale).

The curves reflect the production capacity of existing mines, including planned expansions when known. It was assumed that all operations produce at full (100 pct) capacity over the life of the mine. Because actual production may be at less than capacity levels, the curves would not actually decline as rapidly as shown. In addition, these deposits contain at least 20 billion mt of recoverable phosphate rock at the inferred resource level, some of which may be reclassified in the future to the demonstrated level. Because it was assumed that over the long run all costs of production, including 15-pct DCFROR, must be recovered, potential annual capacities are shown for selected cost ranges that include a 15-pct DCFROR.

The curves illustrate the fact that potential production from the demonstrated resources of producing mines in the United States will likely begin to decline after 1986, while

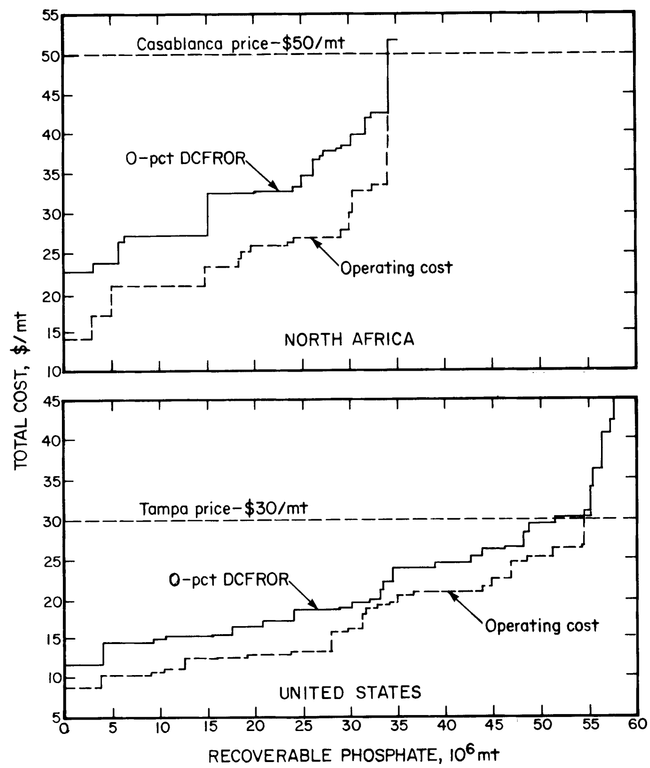

Figure 5.—1985 phosphate rock capacity from producing mines in the United States and north Africa (January 1985 U.S. dollars.)

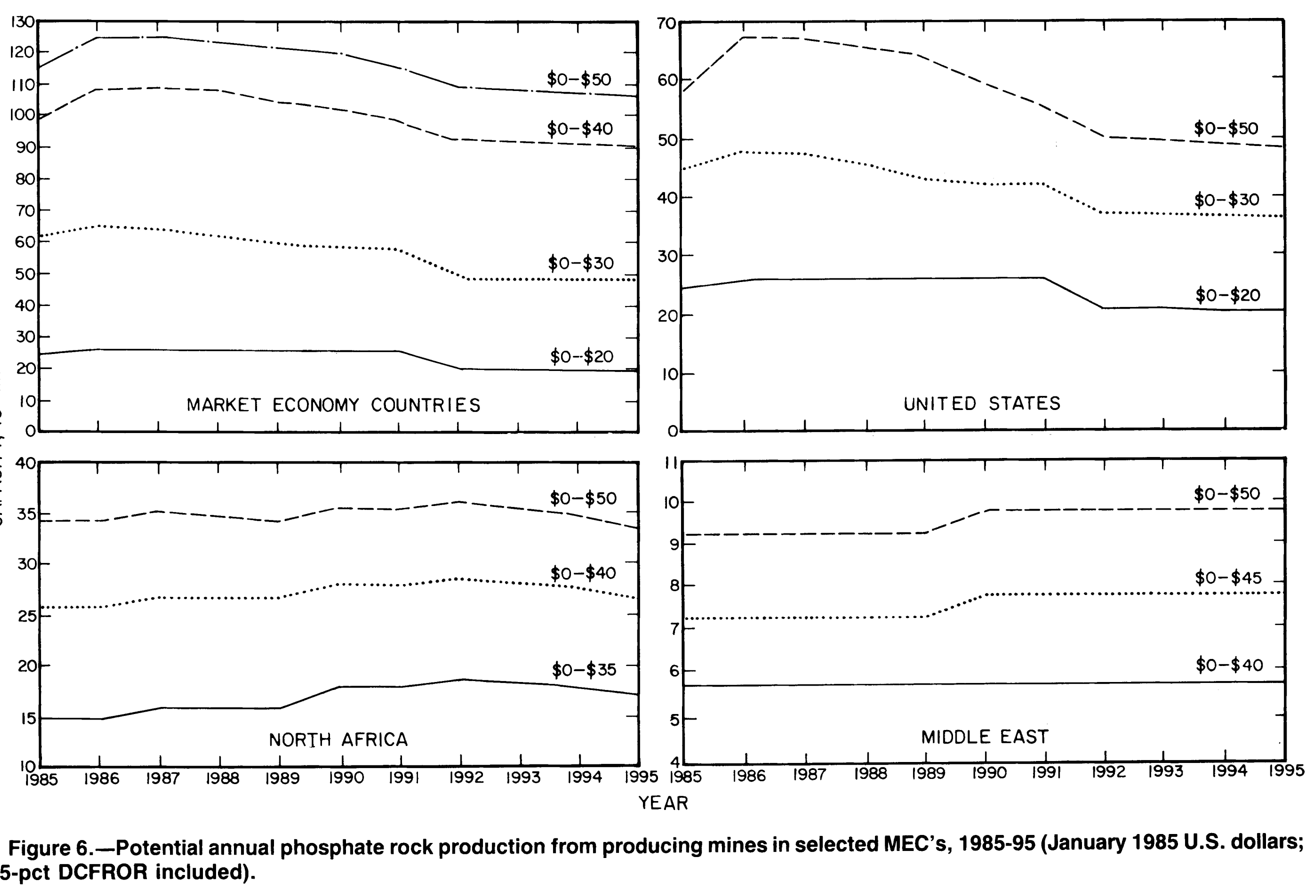

Figure 6.—Potential annual phosphate rock production from producing mines in selected MEC's, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

production from north Africa will decrease only slightly. The U.S. phosphate industry has been producing at less than full capacity in recent years, however, so the decline in potential U.S. production shown on the curve will actually be delayed for several more years and the eventual decline will likely be more gradual than shown. Some of the estimated decline could be counteracted by an expansion of production capacities of the remaining producers, which have large demonstrated resources, although such an expansion would effectively shorten their producing lives. In addition, some of the mines may be able to produce from inferred resources. However, much of the projected decline of U.S. mines would have to be met through imports or through the development of new mines, which in most cases will have higher total costs. Therefore, the United States might have to spend large amounts of capital to maintain production at current levels. North African mines have large demonstrated resources and are capable of producing for many years. As a result, the cost advantage in the world phosphate export industry could shift from Florida to north Africa.

CONCLUSIONS

The United States has traditionally been the largest producer of phosphate rock and phosphate products among MEC's. Based on this study, the costs of mining and processing phosphate rock from producing southeastern U.S. deposits are currently the lowest of the MEC's. However, much of these resources may soon be depleted. Costs for mining phosphate rock from new deposits in this area are estimated to be higher than the costs of those in Morocco and other north African countries, where resources of existing mines are significant and would allow these countries to continue to produce near these cost levels for many years.

However, there are several factors that could greatly enhance the outlook for phosphate availability from the United States, particularly over the long run. In addition to the demonstrated resources evaluated in this study, an estimated 7 billion mt of potentially recoverable phosphate rock exists at the inferred level (over 80 pct in the southeast), and over 24 billion mt exists at the hypothetical resource level (over 60 pct in the southeast). New deposits will likely be discovered (particularly offshore deposits along the eastern seaboard); low-grade material could become economically minable; or technological advances could enable processing high-magnesium oxide material or the mining of deep deposits by the borehole mining technique. Each of these factors could greatly increase the amount of phosphate available in the future. Of immediate interest to the U.S. phosphate industry is more than 2 billion mt of recoverable phosphate rock in Florida, at the identified resource level, which contain high-magnesium oxide material and are presently considered unacceptable by the industry owing to the higher beneficiation costs of producing an acceptable acid plant feed. Given the progress several phosphate companies and the Bureau have made in developing beneficiation technologies to lower the grade of magnesium oxide in the phosphate rock product, this additional 2 billion mt of rock could likely become available in the near future, but at a higher cost.

In summary, although the United States has sufficient demonstrated resources of phosphate rock (plus huge quantities at the inferred and hypothetical resource levels) to satisfy domestic consumption for many years to come, its future ability to continue to compete in the major export markets is uncertain.

REFERENCES

1. Fantel, R.J., T.F. Anstett, G.R. Peterson, K.E. Porter, and D.E. Sullivan. Phosphate Rock Availability—World. A Minerals Availability Program Appraisal. BuMines IC 8989, 1984, 65 pp.

2. Stowasser, W.F. Phosphate Rock. Ch. in Mineral Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 579-594.

3. ______. Phosphate Rock. Ch. in BuMines Minerals Yearbook 1985, v. 1, pp. 745-762.

4. ______. Phosphate Rock. Sec. in BuMines Mineral Commodity Summaries 1986, p. 116-117.

5. Krauss, U.H., H.G. Saam, and H.W. Schmidt. International Strategic Minerals Inventory Summary Report—Phosphate. U.S. Geol. Surv. Circ. 930-C, 1984, 41 pp.

PLATINUM

INTRODUCTION

Platinum possesses a wide variety of physical and chemical properties that make it essential in many industrial applications, including automobile emissions control, petroleum refining, and electronics. It is a strategic commodity for the United States because of the paucity of domestic production and heavy dependence upon imports from one major source—the Republic of South Africa.

The United States and Japan are the major platinum consumers. They accounted for 42 and 48 pct, respectively, of total 1985 market economy country (MEC) platinum consumption of 2.9 million tr oz. In 1985, total U.S. platinum consumption was 1.2 million tr oz with 66 pct for automotive catalytic converters, 10 pct for the electrical industry, and 7 pct for the chemical industry. Total Japanese platinum consumption was 1.3 million tr oz. Jewelry accounted for 49 pct and the automotive and electrical industries ranked a distant second and third, repectively, at 13 and 12 pct of total Japanese platinum consumption.

Platinum, palladium, iridium, osmium, rhodium, and ruthenium are the platinum-group metals (PGM) that commonly occur together in nature. Of the six metals, platinum and palladium have the greatest economic significance and are found in by far the largest quantities. For this reason, only platinum and palladium are evaluated for availability in this chapter. In situ PGM and platinum grades are reported as troy ounces per metric ton. Recoverable platinum is reported in troy ounces of at least 99.9 pct purity.

This chapter is an updated summary of Bureau of Mines Information Circular 8897 "Platinum Availability—Market Economy Countries. A Minerals Availability System Appraisal."(*1*).[1] It incorporates recent (January 1985) information from numerous sources. Additional information on international platinum deposits is available from a recent U.S. Geological Survey publication (*2*).

GEOLOGY AND RESOURCES

GEOLOGY

PGM distribution in the continental crust is difficult to estimate owing to the low concentration of these elements. Mason (*3*) estimated crustal abundances as follows: Pt—5 ppb, Pd—10 ppb, and Ir, Os, Rh, and Ru—1 ppb each. A useful equivalent is 1 ppb equals 0.000032 tr oz/mt.

The major platinum deposits are associated with mafic and ultramafic complexes, such as the Stillwater Complex in the United States and the Bushveld Complex in the Republic of South Africa. Additional geologic information on platinum deposits is available in U.S. Geological Survey Professional Paper 820 (*4*).

Platinum placers consist of unconsolidated alluvial deposits in present or ancient stream valleys, terraces, or beaches. The past producer of the Goodnews Bay mining district in the valley of the Salmon River in Alaska and the placers of Columbia are good examples of alluvial platinum deposits.

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

RESOURCES

Platinum always occurs in nature with one or more members of the PGM's. World resource estimates for platinum (expressed in contained PGM) are illustrated in figure 1.

As shown, total world PGM resources are estimated to be 3.3 billion tr oz and the in situ world reserve base is estimated to be 1.2 billion tr oz of PGM including 970 million tr oz in the Republic of South Africa (5). The Republic of South Africa estimate includes both the Merensky Reef and the Upper Group chromite seam (UG-2) and is calculated to a vertical depth of 600 m. Identified resources for the Merensky Reef and UG-2 (calculated to a vertical depth of 1,200 meters) total 560 and 1,040 million tr oz of PGM, respectively (*6*).

The Great Dyke of Zimbabwe includes four separate igneous complexes containing large quantities of platinum in association with palladium, nickel, and copper. These complexes, the Hartley, Selukwe, Wedza, and Musengezi, contain PGM resources of 100 million tr oz. The PGM resources of Zimbabwe are not sufficiently defined to be considered demonstrated and so were not evaluated for availability.

Table 1 presents evaluated MEC resources in terms of in situ material, in situ PGM and platinum grades, contained PGM and platinum, and recoverable platinum. Total evaluated resources are estimated at 235 million tr oz of contained PGM in four countries, 229 million tr oz of which are from the Merensky Reef of the Bushveld Complex in the Republic of South Africa. Table 2 is a list of the properties included in this study.

In order to estimate the platinum resources contained in PGM resources, the ratio of platinum to palladium must be determined. The ratio not only varies from one complex to the other, but also varies according to the individual layers within the same complex. For example, in the Bushveld Complex of the Republic of South Africa, the Merensky Reef has a platinum to palladium ratio of 2.4:1. Below the Merensky Reef (running roughly parallel to it) lies the UG-2, which has a platinum to palladium ratio of 1.2:1 (*3*).

Evaluated resources of the Republic of South Africa, which are based on 1980 data (*1*), are estimated at 143 million tr oz of platinum contained solely in the Merensky Reef at three producing mines (Rustenburg, Impala, and Western) and one nonproducing deposit, Der Brochen. Production used to be from the Merensky Reef exclusively until 1982 when Western (and recently Impala and Rustenburg) started to mine the UG-2 using a process developed by the Republic of South Africa's National Institute of Metallurgy, which solved the problem of removing chromite prior to smelting. However, because of the lack of cost data concerning mining, milling, and smelting, resources from the UG-2, were not included in this study.

Eighty-nine million troy ounces is potentially recoverable from Zimbabwe according to reference 1; however, recovery of platinum from these resources would be very costly compared with the present market price of platinum. Platinum resources located in the Soviet Union, China, and other centrally planned economy countries (CPEC's) are not analyzed here because the difficulty in collecting quantitative resource information precludes supportable production cost estimates. Some platinum is also recovered as a byproduct from nickel mines in the Sudbury Complex of Canada.

Table 1.—Summary of MEC demonstrated platinum-group metal (PGM) resources evaluated for this study, as of January 1985

Country	In situ material, 10^6 mt	In situ grade, tr oz/mt PGM	In situ grade, tr oz/mt Platinum	Contained, 10^6 tr oz PGM	Contained, 10^6 tr oz Platinum	Recoverable platinum, 10^6 tr oz
Canada,[1] Colombia	[2]NM	[2]NM	[2]NM	3	1	1
South Africa, Rep. of[3]	953	0.24	0.15	229	143	103
United States	[2]NM	[2]NM	[2]NM	3	([4])	([4])
Total	NM	NM	NM	235	144	104

NM Not meaningful. All numbers rounded to nearest whole number.
[1]Does not include Sudbury Complex.
[2]Not meaningful, aggregate figures not calculated owing to a mixture of several different deposit types.
[3]Does not include the UG-2 of the Bushveld Complex.
[4]Less than 1/2 unit or 500,000 tr oz.

Table 2.—MEC platinum properties included in this study

Country and deposit name	Owner	Current status[1]	Mine method[2]	Mill method[3]	Annual mine capacity,[4] 10^3 mt ore
Canada: Lac Des Isles #1.	Boston Bay Mines.	E	OP	Fl	820
Colombia:					
Choco Pacifico.	Mineros Del Choco.	P	DR	Gr	12,000
South Africa, Rep. of:					
Rustenburg	Rustenburg Platinum Holdings Ltd.	P	ST	Fl	17,740
Impala	Union Corp. and others.	P	ST	Fl	14,520
Western Platinum.	Western Platinum Ltd.	P	ST	Fl	2,210
Der Brochen	Geduld Investments Ltd., East Rand Proprietary Ltd.	E	ST	Fl	1,440
United States:					
Salmon River.	R.A. Hanson Mining Co.	PP	DR	Gr	1,300
Stillwater Platinum.	Stillwater Mining Co.	D	C&F	Fl	158,600

[1] As of January 1985: P, producer; PP, past producer; D, developing; E, explored.
[2] C&F, cut and fill; DR, dredging; OP, open pit; ST, stoping.
[3] Fl, flotation; Gr, gravity.
[4] Actual or estimated.

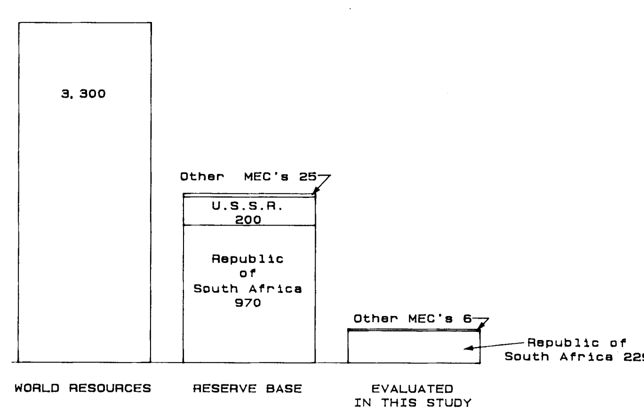

Figure 1.—Estimates of world platinum-group metal resources (million troy ounces).

U.S. AND WORLD HISTORICAL PRODUCTION

World annual production of PGM has increased from 0.6 million tr oz to 7.9 million tr oz between 1950 and 1985. Historical production in 5-yr intervals from 1950 to 1985 is illustrated in figure 2.

The Republic of South Africa and the U.S.S.R. are the two leading producers, having had almost equal shares since 1975. The production of platinum is, however, dominated by the Republic of South Africa because the platinum-palladium ratio in PGM's is about 2.5:1 in the Republic of South Africa and only 0.4:1 in the Soviet Union. In 1985, world platinum production totalled 3.4 million tr oz; the Republic of South Africa accounted for 67 pct or 2.26 million tr oz, and the U.S.S.R. produced 0.95 million tr oz or 28 pct of the total. Canada, which recovers platinum as a byproduct of nickel production from the Sudbury Complex in Ontario, ranked a distant third in production, with 0.15 million tr oz of platinum or approximately 4 pct of the world total. The United States produced only 1,000 tr oz of platinum as a byproduct from copper-gold mines, but byproduct platinum recovery is decreasing steadily because of the poor copper market and the closing of copper mines in recent years. Small amounts of platinum are also produced from deposits in Australia, Colombia, Ethiopia, Finland, Yugoslavia, and Zimbabwe, which produced a combined total of 36,000 tr oz in 1985.

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

Nearly 95 pct of the platinum from MEC's is mined by underground stoping methods. The remaining 5 pct is recovered by dredging and surface methods. The eight deposits evaluated in this study are listed in table 2 with specific data such as owner, status, mining and milling methods, and production capacity.

The large annual capacities at Rustenburg and Impala in the Republic of South Africa are the result of combining workings at several locations. Rustenburg Platinum Holdings Ltd. operates four platinum mines at two separate areas. Three mines, the Amandelbult section near Chromite, the Rustenburg section at Bleskop, and the Union section at Swartkop, are part of the Western Bushveld Complex while the fourth, Atok platinum mines, is located in the northern portion of the Eastern Bushveld Complex. The Impala operation includes four mines: North Bafokeng, South Bafokeng, North Wildebeestfontein, and South Wildebeestfontein, which are all located in the southern part of the Western Bushveld Complex. Rustenburg, Impala, and the Western Platinum Mine are currently recovering platinum from the UG-2. Western Platinum Mine, which is located in the southern half of the Western Bushveld Complex, is the first operation to build a plant exclusively designed for the treatment of UG-2 ore. Recently, about 40 pct of Western Platinum's total production has come from the UG-2.

Mining methods employed in the Republic of South Africa are straightforward because of the regularity of strike and dip of mined units. Dredging operations for gold and platinum at Colombia's Choco Pacifico consist of standard bucket-line dredges with a trommel-screen-jig circuit for metal recovery. Canada's Lac des Iles would employ open-pit methods.

Only two properties in the United States were analyzed; the Salmon River in Alaska and the Stillwater deposit in Montana. The former utilized dredging and the latter would utilize shrinkage or cut-and-fill stoping.

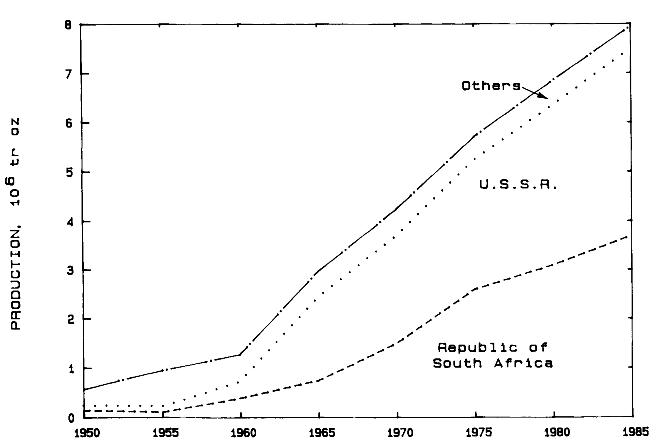

Figure 2.—World platinum-group metal production, 1950-85.

PROCESSING

Except for the relatively insignificant production that results from placer mining, most platinum is beneficiated using flotation methods. In this process, ore is crushed, ground, and floated to recover the metallic minerals. The result concentrate, which in some cases is pelletized, is then sent to an electric furnace where it is partially oxidized to a "green matte." The green matte is transferred to converters where, under an oxygen-rich atmosphere, it is melted and oxidized, resulting in a "white matte." The white matte is sent to a nickel-copper refinery and the sludge resulting from the refining process is sent to a precious metals refinery for further separation.

The precious-metal concentrates (sludge) from the nickel-copper refinery have a PGM content that normally ranges between 25 pct and 75 pct. The first stage of the refining process is the removal of the base metal content by roasting, followed by a leaching process. The individual PGM's are then separated and refined by conventional selective precipitation methods. In the recently developed solvent extraction method, preleaching allows removal of base metals, and a single leaching with chlorine and hydrochloric acid dissolves the PGM's. In a series of continuous operations, metal recoveries from stripping solutions are accomplished by the precipitation of insoluble salts or complexes. The order of PGM recovery is palladium, platinum, osmium (by distillation), ruthenium, iridium, and rhodium. This new solvent extraction method incurs lower capital and operating costs and improves processing time significantly.

NEW TECHNOLOGY

In the Bushveld Complex of the Republic of South Africa, the UG-2 lies below the Merensky Reef. At the Western Platinum operation, the UG-2 ore has higher PGM grades than Merensky ore and the lower nickel and copper contents of the UG-2 ore result in a large reduction in smelting and refining costs. Until recently, UG-2 ore presented milling and smelting problems because of the high content (30-32 pct) of chromic oxide. Metallurgical research at the South African Mineral Technology Institute (Mintek) has provided extensive information for the successful commercial exploitation of the UG-2 at the Western Platinum operation. Procedures were developed to reduce the chromic oxide content of the platiniferous concentrates to levels that were acceptable to a modified smelting technique (7). Western Platinum conducted an extensive testing program prior to installation of the new flotation plant in 1984. A new nickel-copper refinery, located near the PGM smelter, began operation in 1985 and supplies feed for the PGM refinery at Brakpan, thus eliminating the shipping of nickel-copper matte to Norway and thereby speeding up the PGM recovery time cycle. Western Platinum is not currently recovering chromium from the UG-2, and chromium fines are piped to the tailings dam.

PRODUCTION COSTS

The average production costs for the three producing mines in the Republic of South Africa and the two dredging operations, one each in Colombia and United States, are listed in table 3. Costs are in constant 1985 dollars and are based on circa 1980 technology. Costs for operations in other countries are not shown in order to avoid disclosing confidential data. Because platinum and palladium are essentially coproducts, this study uses a price proportioning methodology for economic analysis (see the methodology section of this Bulletin for a detailed description of price proportions).

Mining costs for underground operations in the Republic of South Africa average \$278/tr oz of refined platinum or \$22/mt of ore. It is comparatively low, primarily because of the well-defined nature of the ore body, which allows for simple, conventional mining methods and relatively low cost labor. Dredging operations average \$336/tr oz of refined platinum or \$0.39/mt of ore. The higher mining cost in terms of refined platinum in the dredging operations is due to the extremely low platinum grades that are characteristic of placer deposits. Milling costs average \$41/tr oz for the Republic of South Africa and \$174/tr oz for dredging operations based on costs of \$3.06/mt and \$0.20/mt ore, respectively.

The Republic of South Africa operations recover all members of PGM, thus incurring a \$97/tr oz of other operating cost that includes smelting, refining, and transportation costs for all commodities. The dredging operation in Colombia also recovers gold and the U.S. dredging operation also recovers palladium; they incur an average \$34/tr oz as other operating cost.

The Republic of South Africa has been mining the Merensky Reef for a long time, and capital investment costs associated with Merensky Reef operations are low. Capital investment costs associated with recovery of platinum from the UG-2 at the new facilities of the Western Platinum Mine are not included in table 3. Recovery of capital and return on capital for one placer deposit is high because of investments associated with recently rebuilding the dredge, thus causing the overall capital investment costs to be higher than the Republic of South Africa operations. This higher average capital invesment cost also impacts on the total production cost estimated at the 15-pct-DCFROR level. Recovery on investment for the dredging operations is more than eight times higher than the Republic of South Africa mines.

Table 3.—Platinum production costs for underground and dredging operations

(All costs are in January 1985 U.S. dollars per troy ounce of platinum on a weighted-average basis)

Mining method	Operating cost			Byproduct credit	Net operating cost	Recovery of capital[2]	0-pct DCFROR		15-pct DCFROR		
	Mine	Mill	Other[1]				Taxes and royalties[3]	Total cost[4]	Taxes and royalties[5]	Return on investment[6]	Total cost[7]
Underground: South Africa, Rep. of	278	41	97	288	128	17	30	175	36	9	190
Dredging: Colombia, United States	336	174	34	394	150	103	4	257	18	75	346

[1] Includes cost of smelting, refining and transportation for all commodities.
[2] Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure as of January 1985, and investments required over the life of the operation.
[3] Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at 0-pct DCFROR.
[4] Equal to the sum of net operating costs, taxation, and capital recovery determined at a 0-pct DCFROR.
[5] Includes property, State, Federal and severance taxes and royalties, where applicable, calculated at 15-pct DCFROR.
[6] The revenue increase per troy ounce necessary to obtain a 15-pct DCFROR.
[7] Equal to the sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery *plus* the per ounce increase necessary to provide a 15-pct DCFROR after taxation.

AVAILABILITY

TOTAL RECOVERABLE

A total of eight mines and deposits were analyzed, six foreign and two domestic. These mines and deposits account for most of the platinum production in MEC's as of the beginning of 1985. Apporoximately 104 million tr oz of platinum is potentially available from the mines and deposits evaluated (fig. 3). Four producing mines, three in the Republic of South Africa and one in Colombia, account for 96 million tr oz of platinum, 82 pct of which is available from the Rustenburg Platinum operations. Eighty-six percent of the total available platinum is economic at the January 1985 platinum spot price of $274/tr oz, all from producing mines. Platinum potentially available from nonproducing deposits would require a minimum platinum price of $375/tr oz.

Total platinum availability of the Republic of South Africa in this study is limited to evaluated resources in the Merensky Reef; additional PGM resources with higher platinum and palladium ratios located in the UG-2 could double the Republic of South Africa's platinum resources. The tailings from processing the UG-2 ore are stockpiled pending potential future use in ferrochromium production. Rustenburg has been investigating the feasibility of exploiting UG-2 platinum resources and production could begin in the near future. Mining of the UG-2 would utilize the existing shafts and infrastructures and continue development work to new levels at depth. Consequently, it would be more economical than starting a new mine at a different location. The producers in the Republic of South Africa are expected to maintain their dominance in the platinum market with future production from the UG-2.

ANNUAL CAPACITY

Estimated 1985 platinum consumption for MEC's totaled 2.9 million tr oz with 42 and 44 pct for the United States and Japan, respectively. Annual production at full capacity from the four producing mines would provide for 89 pct of this 1985 consumption level. Additional production in 1985 was from the UG-2, which was not evaluated in this study, and from byproduct platinum production from nickel and copper mines.

Potential annual platinum production through 1995 from four producing mines at two total production cost ranges, including a 15-pct DCFROR, is shown in figure 4. The production levels shown represent the potential annual platinum output at full capacity at $200/tr oz and $500/tr oz cost levels. For example, in 1985 the production potential at a total cost of production of $200/tr oz or less was estimated at 1.35 trillion tr oz of platinum from two producers that should remain in production through 1995. At a total production cost of $500/tr oz or below, 2.59 million tr oz was available in 1985, decreasing significantly in 1993 to 1.49 million tr oz owing to apparent depletion of current demonstrated resources for one deposit.

Maximum annual recoverable platinum of 2.7 million tr oz from all evaluated mines and deposits would be potentially available at a total production cost of $610/tr oz of platinum.

Additional platinum from the UG-2 of Rustenburg Mine would be readily available with modification of current beneficiation facilities at the mine site. Gold Fields of the Republic of South Africa has decided to start development work at its property adjacent to Rustenburg's operation at Amandelbult. The Stillwater (MT) Mining Co. statement to the Senate indicated that the mineralized zone containing PGM extends for 30 miles along the northeast flank of the Beartooth Mountains. Only the small portion of it that had been explored in detail and evaluated for economics was included in this study, thus, additional resources could be developed to expand the current resource estimate. Platinum resources from the Great Dyke in Zimbabwe are currently not economic for development owing to a low platinum grade.

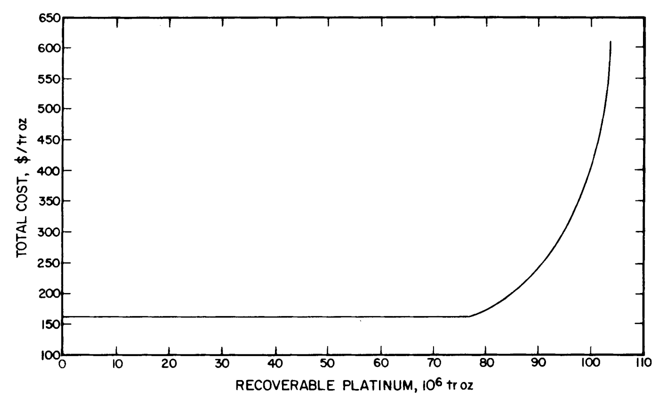

Figure 3.—Total platinum available from MEC's (January 1985 U.S. dollars; 15-pct DCFROR included).

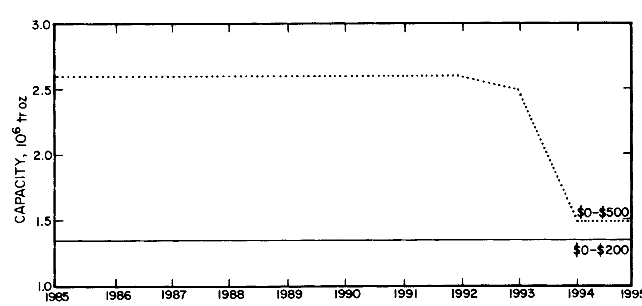

Figure 4.—Potential annual platinum production from producing mines in various MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

CONCLUSIONS

Major known platinum deposits located in four MEC's were chosen for detailed analysis. Approximately 104 million tr oz of recoverable platinum is potentially available at a total production cost range from $150/tr oz to $650/tr oz of platinum including a 15-pct DCFROR. Three producing mines and one nonproducing deposit in the Republic of South Africa account for 99 pct of the total. The remaining 1 pct includes two placer deposits with low platinum ore grades and one surface deposit and one underground deposit with small annual capacities. The Republic of South Africa producing mines, with the advantage of economy of scale, have a weighted-average total production cost of only $175/tr oz at the 0-pct-DCFROR level and $190/tr oz including a 15-pct DCFROR. These producing operations, which are mining the Merensky Reef, accounted for 93 pct of the total potential platinum evaluated in this study.

At a total production cost of $200/tr oz or less, 1.35 million tr oz of platinum could be produced annually from two producers through 1995. This quantity appears adequate to meet current U.S. annual consumption of 1.2 million tr oz. At a platinum production cost of $500/tr oz, 2.59 million tr oz could be produced through 1993 before depletion of current resources significantly affects overall production. Consequently, 89 pct of the 1985 MEC consumption level of 2.9 million tr oz could be met by platinum production at a cost of $500/tr oz. Platinum resources from the UG-2 in the Republic of South Africa have been mined by Western Platinum Ltd. since 1982. The economics of the UG-2 material was not evaluated in this study because of the absence of viable data. Western Platinum has steadily increased production from the UG-2, which suggests that the overall production cost is economic compared with the 1985 platinum spot price of $274/tr oz. Impala and Rustenburg mines are now also producing platinum from the UG-2.

The proposed Stillwater project in the United States plans to mine high-grade platinum resource at 500 st/d. At this planned production level, recoverable platinum from Stillwater would not significantly relieve the U.S. dependence on foreign supply.

There are several factors that could lead to an increase in platinum production. Increased commercial development of the UG-2 of the Bushveld Complex could more than double the amount of platinum available from the Republic of South Africa. Improvements in processing technology could increase the amount of platinum available in producing or currently unexploitable deposits such as the resource in the Great Dyke in Zimbabwe. Expanded recycling efforts in recovery of platinum from automobile catalytic converters could increase the amount of secondary platinum reused in the automotive industry. Discovery of new deposits could also contribute to potential supplies.

REFERENCES

1. Anstett, T.F., D.I. Bleiwas, and C. Sheng-Fogg. Platinum Availability—Market Economy Countries. A Minerals Availability System Appraisal. BuMines IC 8897, 1982, 16 pp.
2. Sutphin, D.M., and N.J. Page. International Strategic Minerals Summary Report—Platinum Group Metals. U.S. Geol. Surv. Circ. 930-E, 1986, 34 pp.
3. Mason, B. Principles of Geochemistry. Wiley, 1958, 310 pp.
4. Page, N.J., A.L. Clark, G.A. Desborough, and R.L. Parker. Platinum-Group Metals. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 379-384.
5. Loebenstein, J.R. Platinum-Group Metals. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 118-119.
6. Robson, B.G. Platinum 1985. Johnson Matthey P.L.C. (London, England), May 1985, 68 pp.
7. Burgman, C.F. UG-2 Reef of the Bushveld Complex. Mintek Rev. No. 1, 1985, 15 pp.

POTASH

INTRODUCTION

Potassium, along with phosphorus and nitrogen, is an essential nutrient for plant growth and is used extensively in fertilizers. There is no substitute for potassium fertilizers. Approximately 95 pct of U.S. consumption is as a fertilizer, other uses include soaps and detergents, glass and ceramics, and chemical dyes and drugs. The principal source of potassium is mined potash.

Canada has the largest quantity of potash resources in the world and is the largest market economy country (MEC) producer. The United States is the largest consumer of potash, and a large share of Canadian potash output is exported to the United States. U.S. mine production was approximately one-fifth of Canadian production during 1985.

The industry uses K_2O, the oxide of potassium, as a standard of comparison for the different forms of potash. The amount of potash in products is measured in terms of the percent K_2O because K_2O best represents the potassium value that is used by plants as a fertilizer. Important potash products include muriate of potash (potassium chloride), potassium sulfate, potassium nitrate, and potassium magnesium sulfate. Potassium chloride accounts for 90 pct of the annual capacity in terms of K_2O equivalents of producing operations included in this study. Potash products are differentiated into grades based on grain size. The common sizes are granular, coarse, standard, and special standard. The quantities discussed in the availability analysis are in terms of metric tons of potash mill product.

Most of the information presented in this chapter is an updated summary of Bureau of Mines Information Circular 9084 "Potash Availability—Market Economy Countries. A Minerals Availability Appraisal (*1*).[1] Additional information on the domestic and foreign potash industry is available from other Bureau publications (*2-4*).

GEOLOGY AND RESOURCES

Potash is the term used for a variety of naturally occurring potassium-bearing minerals and mineral products containing potassium. Sylvite, which is potassium chloride (KCl), is the most important potassium mineral. The product form of potassium chloride is referred to as muriate of potash. Sylvinite, a mixture of sylvite and halite (NaCl), is commercially the most important naturally occurring potassium ore because of its relatively high potassium content and ease of beneficiation. Additional potassium minerals that are mined include langbeinite in the United States, hartsalz in the Federal Republic of Germany and the German Democratic Republic, kainite in Italy, and caliche in Chile. Additional geologic information on these deposits is available from the U.S. Geological Survey (*5*).

GEOLOGY

Potash deposits are usually in sedimentary layers, often interbedded with halite (sodium chloride). This is because the formation of potassium-bearing deposits is usually a result of cyclic precipitation during evaporation of solutions that also contain sodium. Potassium and sodium are common elements found in most rocks. When the rocks weather, ions, including those of potassium and sodium, are released into ground water. The ions in ground water can be absorbed into the soil, or be transported through rivers to the sea.

Potash deposits were usually formed in geologic history when parts of the ocean became landlocked and were subsequently subjected to arid conditions. Evaporation concentrated the brine and caused precipitation of salts, beginning with calcium and sodium salt. When the brine became sufficiently concentrated, the highly soluble potassium salts precipitated. Wet periods would cause the salinity of the sea to decrease, causing the sylvinite precipitation to cease and the halite precipitation to again take place. This cycle could repeat many times, or begin again with more marine brines being added to these segregated seas. When the sea finally dried, a halite body interbedded with sylvinite layers remained.

Today, areas that have this geologic history account for most of the profitably mined resources in the world. In some cases, such as the Dead Sea and the Great Salt Lake of Utah, brines containing recoverable amounts of potassium are also exploited.

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

RESOURCES

As shown in table 1, the MEC deposits included in this analysis contain demonstrated in situ potash resources of 11.7 billion mt K_2O equivalent [centrally planned economy country (CPEC) deposits were not included]; of this total, 82 pct of this is located in Canada, 11 pct in the Dead Sea, less than 4 pct in Western Europe, and 2 pct in the United States. Muriate of potash accounts for 94 pct of the potash potentially recoverable from resources located in MEC's, while 4 pct would be in the form of potassium sulfate. Table 2 is a list of the properties included in this study.

Figure 1 illustrates the resources evaluated in this study, the world reserve base, and world resource estimates of identified resources. Reserve base estimates are larger than the demonstrated resources evaluated in this study because they include data for CPEC's. World identified resources are much larger than the other estimates because they include inferred in addition to demonstrated resources. Inferred resources are associated with many of the deposits included in this study, the largest of which are located in Canada. It has been estimated that Canadian inferred resources minable by conventional methods are 35 billion mt (7).

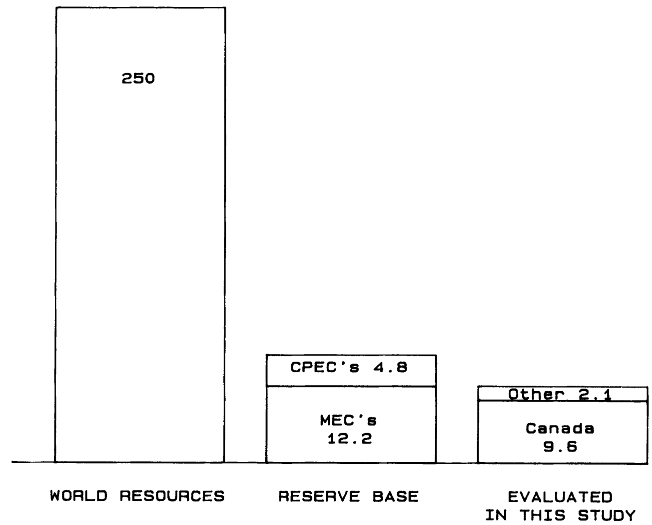

Figure 1.—Estimates of world potash resources (billion metric tons of K_2O).

Table 1.—Summary of MEC demonstrated potash resources evaluated for this study, as of January 1985

Geographic area	Grade, wt pct K_2O	In situ resources, 10^6mt	In situ K_2O, 10^6 mt	Total K_2O,[1] 10^6 mt	Product share, wt pct of K_2O		
					Muriate[2]	Sulfate[3]	Other[4]
United States:							
New Mexico	12.75	831	106	70	80	6	14
Utah	.85	9,057	[5]77	[5]76	3	97	([6])
Total or av[7]	1.85	9,888	183	146	40	53	7
Canada	22.50	42,680	9,604	2,970	100	Unk	Unk
Europe	12.04	3,463	417	239	45	24	31
Dead Sea	.75	163,540	1,226	[8]164	100	Unk	Unk
Other	7.27	3,040	221	66	95	5	Unk
Total or av	[9]5.23	222,611	11,651	3,585	94	4	2

Unk, Unknown.
[1] In recovered products.
[2] 60 pct K_2O content.
[3] 50 pct K_2O content.
[4] Includes potassium magnesium sulfate in New Mexico, manure salt in Utah, and variety of products of the mines in the Federal Republic of Germany for Europe.
[5] Values are close because the analysis assumed that all the K_2O in the north arm of the Great Salt Lake would be recovered. As potash is recovered, more is brought in by streams and when the brine becomes less concentrated, previously precipitated salts on the lake bottom would be redissolved.
[6] Less than 0.5 wt pct.
[7] Does not include the 162 million mt of K_2O equivalent at Searles Lake and the Salton Sea, which could be produced as a byproduct.
[8] K_2O in the product is much lower than in situ K_2O because the analysis is based on a 25-pct recovery. After 25 pct is taken from the Dead Sea, the grade of the remaining brine would be too low to make recovery economic.
[9] Includes both ore and brine, average grades are ore, 20.5 wt pct, brine, 0.75 wt pct.

Table 2.—MEC potash properties included in this study

Country and deposit name	Owner	Current status[1]	Mine method[2]	Mill method[3]	Potash product[4]	Mill feed grade[5]	Mill capacity: Ore[6]	Mill capacity: Muriate product[7]
Brazil:								
Fazendinha	Petrobas Minercao	N	RP	Fl	M	B	A	A
Taquari-Vassouras	..do	N	RP	Fl	M	B	A	A
Canada:								
Allan	Potash Corp. of Saskatchewan, Texasgulf.	P	RP	Fl, Cr	M	C	B	C
Borden	Unknown	N	RP	Fl, Cr	M	C	B	C
Bredenbury	Potash Corp. of Saskatchewan	N	RP	Fl, Cr	M	C	D	D
Burr	Unknown	N	RP	Fl, Cr	M	C	B	C
Cloverhill	Denison Mines	N	ICF	Fl, Cr	M	D	B	C
Colonsay (CCP)	Noranda Mines Ltd	P	RP	Fl, Cr	M	D	B	D
Cory Division	Potash Corp. of Saskatchewan	P	RP	Fl, Cr	M	C	A	B
Dundurn	Unknown	N	RP	Fl, Cr	M	D	B	D
Esterhazy K-1	IMC, Potash Corp. of Saskatchewan.	P	RP	Fl, Cr	M	C	C	D
Esterhazy K-2	..do	P	RP	Fl, Cr	M	C	B	D
Kalium	PPG Industries	P	SOL	Cr	M	B	D	D
Lanigan Division	Potash Corp. of Saskatchewan	P	RP	Fl, Cr	M	C	D	D
Lockwood A	Unknown	N	RP	Fl, Cr	M	D	B	C
Lockwood B	..do	N	RP	Fl, Cr	M	D	B	C
McCauley	IMC	N	RP	Fl, Cr	M	C	B	D
Patience Lake	Potash Co. of America	P	RP	Fl, Cr	M	D	A	C
Quill Lake (Kerr-McGee) A.	Unknown	N	RP	Fl, Cr	M	C	B	C
Quill Lake (Kerr-McGee) B.	..do	N	RP	Fl, Cr	M	C	B	C
Quill Lake (Scurry Rainbow) A.	Potash Corp. of Saskatchewan	N	RP	Fl, Cr	M	C	B	C
Quill Lake (Scurry Rainbow) B.	..do	N	RP	Fl, Cr	M	C	B	C
Rocanville Division	..do	P	RP	Fl, Cr	M	C	B	D
Spy Hill	Unknown	N	RP	Fl, Cr	M	C	B	C
Sussex	Potash Co. of America	N[8]	HCF	Fl, Cr	M	C	A	B
Vade	Cominco Ltd	P	RP	Fl, Cr	M	D	B	C
Whitewood	Unknown	N	RP	Fl, Cr	M	B	B	C
Yorkton	..do	N	RP	Fl, Cr	M	C	B	C
Young	..do	N	RP	Fl, Cr	M	C	B	C
Zelma	..do	N	RP	Fl, Cr	M	C	B	C
Chile: Salar de Atacama	Corfo	N	BR	Fl	M	A	D	A
Congo: Holle	People's Republic of Congo	N[9]	RP	Fl	M	D	A	B
France:								
Amelie	Mines de Potasse d'Alsace	P	LC	Cr	M	B	B	B
Marie Louise	..do	P	RP	Cr	M	B	C	D
Theodore	..do	P	RP	Fl	M	A	A	A
Germany (FRG):								
Bergmannssegen-Hugo, Friedrichshall.[10]	Kali und Salz AG	P	SL	Cr	G	A	A	A
Hattorf	..do	P	RP	Cr	G	A	B	A
Neuhof-Ellers	..do	P	RP	Fl	G	A	B	A
Niedersachsen-Riedel	..do	P	SL	Cr	G	A	A	A
Salzdetfurth	..do	P	SL	Cr	G	A	A	A
Siegfried-Giesen	..do	P[11]	SL	Cr	G	A	A	A
Sigmundshall	..do	P	SL	Fl, Cr	G	B	A	B
Wintershall	..do	P	RP	Cr	G	A	C	A
Israel: Dead Sea Works	Dead Sea Works, Ltd	P	BR	Cr	M	A	D	D
Italy:								
Corvillo	ISPEA	N	RP	Fl	S	A	A	A[12]
Milena	EMS	N	RP	Fl	S	A	A	A[12]
Pasquasia	ISPEA	P	RP	Fl	S	A	A	A[12]
Racalmuto	..do	P	RP	Fl	S	A	A	A[12]
Realmonte	EMSAMS	P	RP	Fl	S	A	A	A[12]
Jordan: Arab Potash	Arab Potash Co	P	BR	Cr	M	A	D	D
Spain:								
Cardona	Union Explosives Rio Tinto	P	SL	Fl	M	A	A	A
Esparza	Potasas de Navarra SA	P	LC	Fl	M	A	A	A
Llobregat	Union Explosives Rio Tinto	P	RP	Fl	M	A	A	A
Suria	Minas de Potasas de Suria	P	RP	Fl	M	A	A	A
United Kingdom: Boulby	Cleveland Potash Ltd	P	RP	Fl, Cr	M	D	A	B
United States:[13]								
New Mexico:								
AMAX Chemical	AMAX Chemical Corp	P	RP	Fl	M	A	B	B
Hecla-Day Potash Lease	Hecla-Day Mining Corp	N	RP	Fl	M, S, KMg	A	B	A
Hobbs Potash Facility	New Mexico Potash Corp	P	RP	Cr	M[14]	A	A	B
Hodges Potash Property	Duval Corp	N	RP	Fl, Cr	M, KMg	A	A	A
IMC	IMC	P	RP	Fl	M, S, KMg	A	B	A
Mississippi Chemical Mine	Mississippi Chemical Co	S	RP	Fl	M	A	A	A
Nash Draw	Western Ag-Minerals	P	RP	L	KMg	A	A	A[15]
National Potash Co	Mississippi Chemical Co	S	RP	Fl, Cr	M	A	A	A
Noranda Prospect	Noranda Exploration Inc	N	RP	Fl	M	B	A	B
Lundberg Industries	Lundberg Industries Ltd	P	RP	Fl, Cr	M, S	B	A	B

See footnotes at end of table

Table 2.—MEC potash properties included in this study—Continued

Country and deposit name	Owner	Current status[1]	Mine method[2]	Mill method[3]	Potash product[4]	Mill feed grade[5]	Mill capacity Ore[6]	Mill capacity Muriate product[7]
United States—Con.								
Utah:								
Bonneville (Wendover) . . .	Kaiser Aluminum and Chemical Co.	P	BR	FI	M[16]	A	C	A
Cane Creek	Texasgulf Inc	P	SOL	FI	M	D	A	A
Little Mountain	GSL Minerals	P	BR	Cr	S	A	D	A[12]

[1] P, producer; N, nonproducer; S, standby.
[2] RP, room and pillar; BR, brine recovery; SOL, solution mining; ICF, inclined cut and fill; HCF, horizontal cut and fill; LC, longwall caving; SL, sublevel.
[3] FI, flotation; Cr, crystallization; L, leach.
[4] M, muriate of potash; S, potassium sulfate; KMg, potassium-magnesium sulfate; G, potash operations in the Federal Republic of Germany produce muriate, sulfate, and numerous other potash products.
[5] Feed grade (weight percent K_2O): A—<15.0; B—15.0-20.0; C—20.1-25.0; D—>25.0
[6] Mill ore capacity (10^3 mt/yr ore): A—<3,000; B—3,001-6,000; C—6,001-9,000; D—>9,000.
[7] Muriate product capacity (10^3 mt/yr muriate product): A—<500; B—501-1,000; C—1,001-1,500; D—>1,500.
[8] Sussex Mine began production in 1983.
[9] Holle produced in the early 1970's.
[10] Friedrichshall Mine closed, reserves included with Bergmannssegen-Hugo.
[11] Siegfried-Giesen shut down in 1984.
[12] Grade and product capacity are for potassium sulfate because the mine produces no muriate.
[13] Searles Lake operation of Kerr-McGee located in California was not costed because potash is not the primary product.
[14] Some glaserite produced but not included in analysis.
[15] Grade and product capacity are for potassium magnesium sulfate because the mine produces no muriate.
[16] Bonneville also produces manure salt.

U.S. AND WORLD HISTORICAL PRODUCTION

Potash production was estimated to be 28.6 million mt contained K_2O from 13 countries during 1985. The U.S.S.R. accounted for almost 34 pct of this production, Canada accounted for over 26 pct, and the United States accounted for less than 5 pct of world production. Figure 2 shows world potash production in 5-yr intervals from 1950 through 1985. It shows the U.S.S.R. and Canada emerging as the dominant potash producers over this period. It should be noted that the quantities in the subsequent availability analysis of this study are in terms of mill product and not contained K_2O.

EXTRACTION AND PROCESSING TECHNOLOGY

Individual deposit or operation information such as mining and beneficiation methods, status, capacities, grades, ownership, and first production year for the 68 MEC mines and deposits that were included in the availability analysis are shown in table 2. Table 3 shows the weighted-average in situ grade, mine recovery, mill recovery, and annual capacity for mines using conventional underground methods, solution mine statistics are not included in this table because they are not comparable and would cause misleading averages.

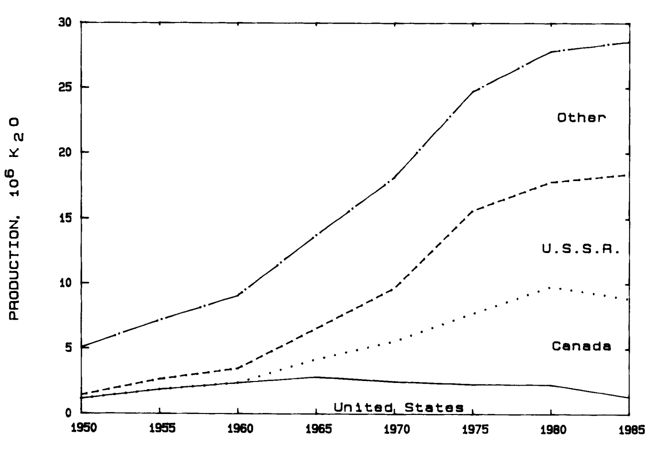

Figure 2.—World potash production, 1950-1985.

MINING

Eighty percent of the potential annual production of muriate of potash from producing mines included in this study would be recovered by conventional underground mining methods, 8 pct would be from underground solution methods, and 12 pct would be recovered from surface brine operations. Producing potassium sulfate operations would derive 91 pct of potential annual production from underground ores recovered using conventional methods, with the remaining 9 pct from brine recovery.

Room-and-pillar is the most comon underground mining method used, although potash recovery using longwall and sublevel techniques is common in Western Europe, and cut-and-fill stoping is the method used in New Brunswick, Canada. Room-and-pillar mining is used when the ore body

Table 3.—In situ grade and mine and mill recovery for potash operations using conventional underground methods, by status and country

Area	In situ grade, pct	Mine recovery, pct	Mill recovery, pct	Annual ore capacity, 10^6 mt
	PRODUCERS			
United States	13.2	87.9	75.3	18
Canada	25.7	36.7	85.5	47
Europe	12.1	63.7	88.9	51
Weighted average or total	21.4	42.3	85.6	116
	NONPRODUCERS			
United States	11.5	88.3	74.5	10
Canada	24.0	35.7	87.7	78
Other[1]	18.2	34.1	87.6	6
Weighted average or total	23.6	35.9	87.7	94

[1] Italy, Brazil, and Congo.

is flat or has a slight dip. In shallow mine depths, such as those in New Mexico, the pillars are recovered. Total mine recovery when pillars are mined is up to 90 pct. However, in deep mines, such as those in Canada, pillars must remain in place for support during the entire life of the mine. Because of this, room-and-pillar mine recoveries in Canada are near 35 pct.

Sublevel stoping is common in steeply dipping deposits that are surrounded with competent country rock. Longwall mining is a highly productive but capital-intensive mining method used in flat-lying deposits that have relatively weak overlying strata.

The Kalium Mine in Canada and the Cane Creek Mine in Utah are the only potash mines using underground solution methods, which involve pumping liquids underground to dissolve the potash and bring it to the surface through wells as a brine. A potash deposit being explored in Michigan would be developed using this method.

The ore body mined at the Kalium Mine, Saskatchewan, occurs at depths greater than 1,000 m, precluding the use of conventional underground mining methods because of rock instability caused by the high pressure of the overlying rock. The Cane Creek Mine, Utah, was originally developed as a conventional underground room-and-pillar mine. Problems were encountered as a result of high rock pressures; a pitched, faulted, and undulating potash bed; weak roof conditions; and gas in the mine. As a result, the mine was converted to a solution operation. Several operations produce potash from naturally occurring brines. They all incorporate solar evaporation ponds in their recovery process.

PROCESSING

Potash ore must be beneficiated to remove halite and clay insolubles before it can be sold as a marketable product. Sylvinite is usually beneficiated by flotation or crystallization, or a combination of both; langbeinite ore is beneficiated only by washing (leaching). Flotation is used in the beneficiation of sylvinite and kainite ores.

The floated or crystallized product is dried to remove as much water as possible to reduce transportation costs and to allow sizing. Sizing via screens is performed after all other beneficiation is completed.

PRODUCTION COSTS

Table 4 and figure 3 illustrate the average production costs per metric ton muriate of potash product for selected underground mines and deposits (all costs are in January 1985 U.S. dollars). These costs are in constant dollar terms and are based on current technology.

Canadian properties have the lowest mine and mill operating costs. This reflects the fact that ore grades in Canada are high relative to other areas. Mine and mill operating costs per metric ton of muriate product for Europe are high because they also include the cost of producing the many byproducts. The revenue generated by byproducts is reflected in the byproduct credit, which is large for Europe, making net operating costs low. Canadian mines in this analysis produce no byproducts and have no byproduct credits.

Transportation cost represents the average cost to transport the muriate product to a port or domestic market. The transportation cost for the operations in the United States represents the cost of shipping by rail from the Carlsbad, NM, area to a typical market in the central Midwest. For other countries, the transportation cost represents the cost to ship to a port or a point of export. Table 4 shows transportation costs to be highest for mines in the United States. The average total transportation cost for mines in the United States to a point in the Midwest is $49/mt. The table shows Canadian transportation costs to port to be about $30/mt. Additional transportation costs would be required to show the total cost to transport potash from Saskatchewan to the market in the United States. Canadian producers appear to have a transportation advantage in the northern part of the United States and domestic producers generally appear to have an advantage in the southwestern States.

Shipments to the central Midwest have shifted. In 1980, domestic producers shipped 94,000 mt to Missouri; however, in 1983 Missouri received no domestic shipments, but received significant shipments from Canada (7). The table shows European operations, which are much closer to their port or market, have lower transportation costs.

Taxation includes property, local, national, and severance taxes, plus royalties, if any. Taxes are higher at a 15-pct discounted-cash-flow rate of return (DCFROR) because taxes are generally related to revenues, and higher revenues are required by definition for a 15-pct DCFROR. Taxes are generally greater for nonproducers in this study, because nonproducers would require a higher taxable income (leading to higher tax payments) in order to cover all costs. At a 15-pct DCFROR, taxes are notably higher for nonproducing operations in Canada because of the impact of the Saskatchewan potash reserve tax. The potash reserve tax is a sliding scale of rates based on operating profit minus a deduction of 5 pct of the capital base. The capital base is an arithmetic sum of all prior capital investments.

The recovery of capital is the sum of the remaining investments and reinvestements estimated over the life of the mine per ton of muriate. It is low for Canada because the high grade of the resource results in more muriate product per ton of capacity. The figure is high for Europe because it is based on the recoverable tonnage of muriate, which is only part of the output of many of these operations.

Total production costs are lowest for European mines partially because their transportation costs are low. Canadian underground producers have costs lower than U.S. underground mines when computed using either a 0- or a 15-pct DCFROR, but nonproducing deposits in Canada, which have lower costs than U.S. operations when computed using a 0-pct DCFROR, have much higher costs when computed at a 15-pct DCFROR as a result of the Saskatchewan reserve tax.

Table 4.—Muriate of potash production costs for underground operations in the United States, Canada, and Europe

(All costs are in January 1985 U.S. dollars per metric ton muriate of potash on a weighted-average basis, weighted by total production)

Country or region	Operating cost: Mine	Operating cost: Mill	Transportation[1]	Byproduct credit	Net operating cost[2]	Recovery of capital[3]	0-pct DCFROR: Taxes and royalties[4]	0-pct DCFROR: Total cost[5]	15-pct DCFROR: Taxes and royalties[6]	15-pct DCFROR: Return on investment[7]	15-pct DCFROR: Total cost[8]
United States:											
Producer	39.50	43.90	49.50	36.30	96.50	6.50	5.90	109.00	8.10	5.20	116.40
Nonproducer	73.00	73.20	49.50	96.00	99.70	22.20	8.90	130.70	26.70	38.50	187.10
Canada:											
Producer	9.70	13.40	32.50	.00	55.60	4.30	6.20	66.10	11.80	5.60	77.30
Nonproducer	11.70	13.00	29.00	.00	53.70	8.40	7.30	69.40	[9]99.00	106.20	267.30
Europe:[10] Producer	61.00	106.50	3.20	[11]155.80	14.90	24.00	10.80	49.60	20.10	12.70	71.70

[1] Transportation cost to selected points of export or domestic markets that have been assumed as the product destination points for this study.
[2] Includes all mine and mill operating costs plus transportation costs minus credit for byproducts.
[3] Sum of remaining undepreciated investments and reinvestments required over the life of the operation.
[4] Includes all property, local, national, and severance taxes, plus any royalty, at a 0-pct DCFROR.
[5] Recovery of all costs of production including capital without discounting.
[6] Includes all property, local and national, and severance taxes, plus any royalty, at a 15-pct DCFROR. Nonproducers would require higher income in order to provide the 15-pct DCFROR; thus, aggregate tax payments are generally higher than for producing operations.
[7] The additional revenue per metric ton required to obtain a 15-pct DCFROR.
[8] Includes a 15-pct DCFROR on all investments over the life of the property.
[9] Taxation of nonproducing potash operations at a 15-pct DCFROR is large as a result of the estimation of the impact of the Saskatchewan reserve tax.
[10] Includes the Federal Republic of Germany, France, Spain, and the United Kingdom.
[11] Byproduct revenues are large owing to the numerous potash products produced in addition to muriate.

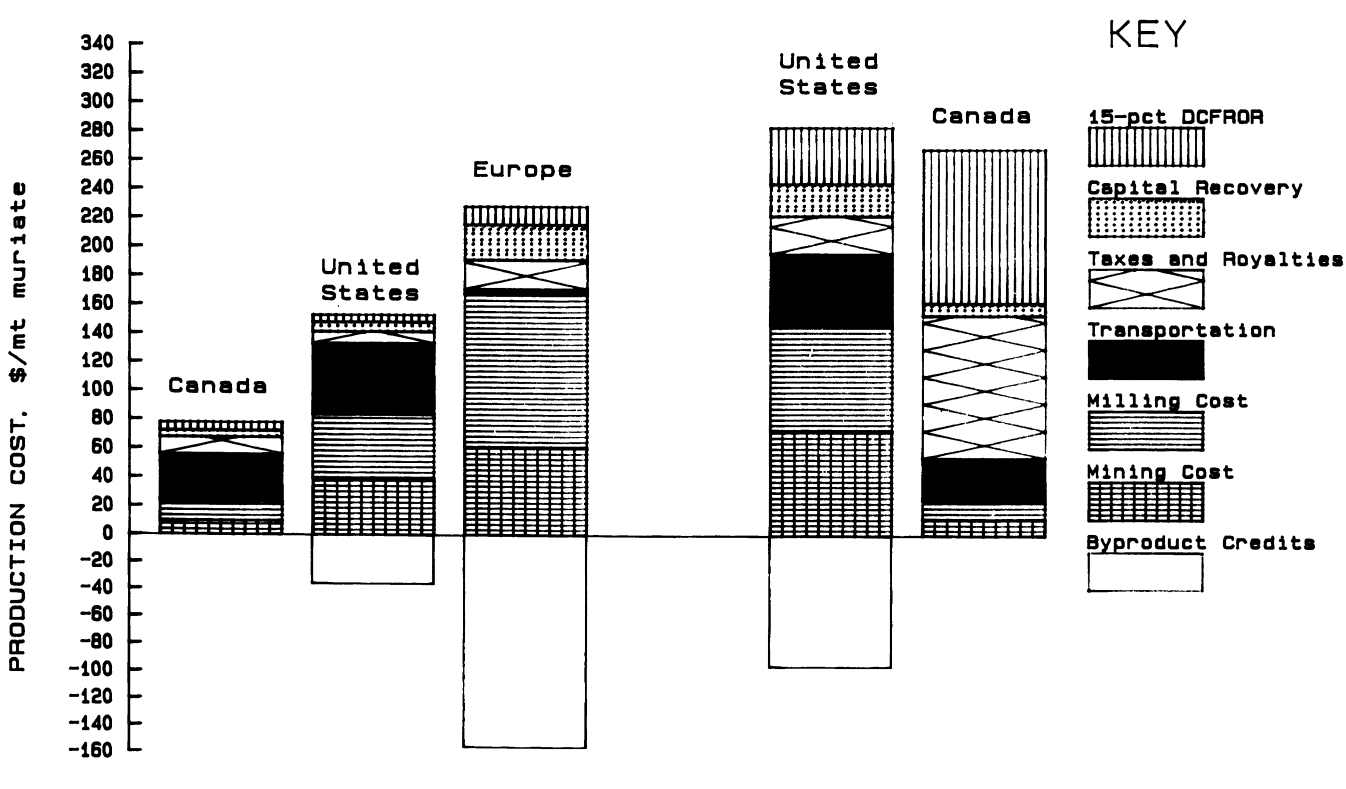

Figure 3.—Muriate of potash production costs for selected MEC's (January 1985 U.S. dollars).

AVAILABILITY

Sixty-eight MEC mines and deposits were analyzed to determine potash availability (13 domestic and 55 foreign); of these, 41 were producers (10 domestic and 31 foreign), although a few of these were temporarily shut down as of early 1985. The 27 nonproducing deposits analyzed, 3 domestic and 24 foreign, include deposits that are developing or have definite developmental plans as well as many deposits that are just explored prospects. Deposits located in CPEC's were not included in this availability analysis owing to a lack of production cost and resource information.

In this analysis it was assumed that all potash product was transported to a point of export or to a domestic market. Muriate of potash, with a minimum of 60 pct K_2O, is the most significant potash product. The availability of potassium sulfate and potassium magnesium sulfate is also discussed.

TOTAL RECOVERABLE

Muriate of Potash

Sixty-one of the properties analyzed produce or are proposed to produce muriate of potash (KCl), 11 domestic and 50 foreign. They represent 90 pct of capacity in terms of K_2O. Thirty-six of these were analyzed as producers, 8 domestic and 28 foreign. Muriate was generally the only product that would potentially be produced at these operations although a small number of operations recover or could recover other potassium and nonpotassium minerals. A total of 5.5 billion mt of muriate is potentially recoverable from the demonstrated resources of the deposits evaluated in this study. Figure 4 shows the total muriate cost-tonnage relationship for all mines and deposits in MEC's included in this analysis with costs less than $400/mt. Total production costs for producing mines are shown on two separate curves. The costs on one curve include a 15-pct DCFROR and the costs on the second producer curve were calculated using a 0-pct DCFROR. The cost curve for nonproducing deposits includes a 15-pct DCFROR.

Fifty-six percent of the potentially recoverable muriate is associated with producing mines. Producing mines with costs less than $114/mt (approximately the January 1985 Carlsbad price, $65/mt, plus transportation), calculated at a 0-pct DCFROR, could potentially recover 3.1 billion mt. Those with costs less than $85/mt could potentially recover 1.6 billion mt.

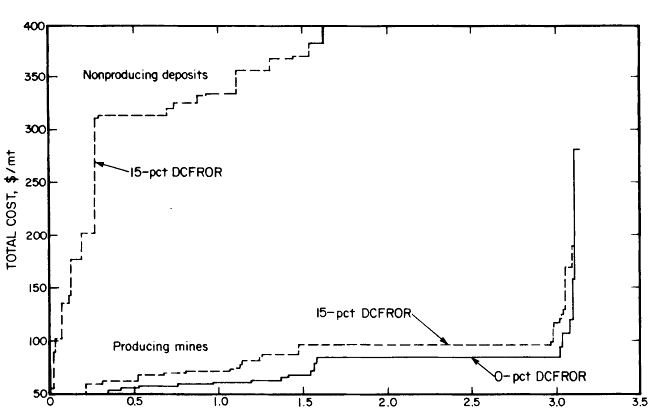

Figure 4.—Potential total potash (muriate) available from MEC's (January 1985 U.S. dollars).

Canada accounts for the largest share of muriate on the curves. At costs less than $114/mt, calculated at a 0-pct DCFROR, Canada accounts for 92 pct of the tonnage from producers, and the United States accounts for 2 pct. For nonproducers, at a total cost (including a 15-pct DCFROR) of less than $180/mt, Canada accounts for 75 pct of the total; the United States accounts for less than 8 pct. Costs for nonproducers in Canada are high as a result of the estimation of the impact of the Saskatchewan reserve tax.

Potassium Sulfate

The potential annual production of potassium sulfate from mines in MEC producing countries, the United States, Italy, and the Federal Republic of Germany, at costs less than $192.90/mt (the January 1985 U.S. price), is 1.8 million mt. More than 80 pct of potential production from currently producing potassium sulfate operations in MEC's would be from Italy and the Federal Republic of Germany; the remainder would be from operations in the United States. The United States, historically, is a net exporter of potassium sulfate. Annual potassium sulfate production in the United States is usually less than 10 pct of the annual production of all potash in terms of K_2O. To supply an expanded demand, annual capacity at existing deposits will have to be expanded.

Potassium Magnesium Sulfate

Four properties in New Mexico produce or could recover potassium magnesium sulfate from langbeinite ores. Three of these operations also recover or could recover muriate of potash. The muriate from these operations was included in the "Muriate of Potash" section.

These four properties could potentially recover 44 million mt of potassium magnesium sulfate with a K_2O grade of near 22 pct, containing 10 million mt K_2O equivalents. Potassium magnesium sulfate potentially recoverable at delivered costs less than $115/mt would potentially be 15 million mt; $115/mt equals the January 1985 market price, $65/mt f.o.b. mill, plus a transportation cost of almost $50/mt.

Miscellaneous Potash Products

Mines located in the Federal Republic of Germany produce other potassium compounds that range in grade from 12 to 50 pct K_2O and sometimes contain magnesium or phosphate in addition to the potash. The revenues from these products were credited to the mines but were not included in separate cost tonnage relationships because of their lack of homogeneity and resulting wide range of market prices. Potash is also produced at nitrate operations in Chile that were not included in this analysis.

ANNUAL CAPACITY

Muriate of Potash

Potential 1985 muriate production at full capacity from operating mines in the United States and Canada is shown in figure 5. The United States and Canada account for 64 pct of the 26.6 million mt of muriate of potash capacity potentially available from producing mines in MEC's during 1985. Currently producing Canadian mines had the capacity to produce an estimated 13.9 million mt product and mines in the United States could have produced 3.1 million mt. Figure 5 has two curves for each region, a 0-pct DCFROR curve and a curve representing only operating costs. At operating costs of \$85/mt (roughly equivalent to the January 1985 Canadian price plus transportation to a Midwestern market), all of the Canadian capacity could have been utilized. At operating costs of \$114/mt (roughly equivalent to the January 1985 U.S. price plus transportation to a Midwestern market), 2.7 million mt could have been produced from U.S. mines. Actual production from Canadian mines during 1985 was 7.5 million mt K_2O (12.3 million mt muriate) while production from U.S. mines was 1.3 million mt K_2O (2.1 million mt muriate equivalents). Actual production is close to the study's capacity estimates at 0-pct DCFROR.

Potential annual production of muriate to 1995 from the demonstrated resources of producing mines is shown in figure 6 by geographic area. Figure 6 has curves for mines in MEC's, in Canada, in the United States, and Europe, including the area near the Dead Sea (note that each curve is constructed at a different scale). The curves reflect the production capacity of existing mines, including planned expansions when known. It is assumed that all operations produce at full (100 pct) capacity over the life of the mine. Because actual production may be at less than full capacity levels, a decline in a curve may not be as rapid as shown. Also, the discovery of additional demonstrated resources would extend these curves. The selected cost ranges on these

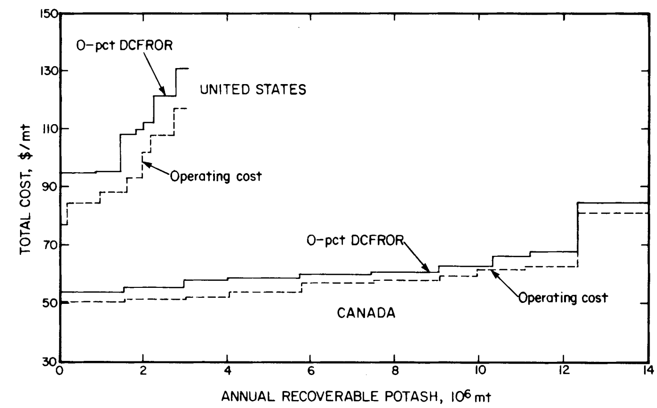

Figure 5.—1985 muriate capacity from producing mines in the United States and Canada (January 1985 U.S. dollars).

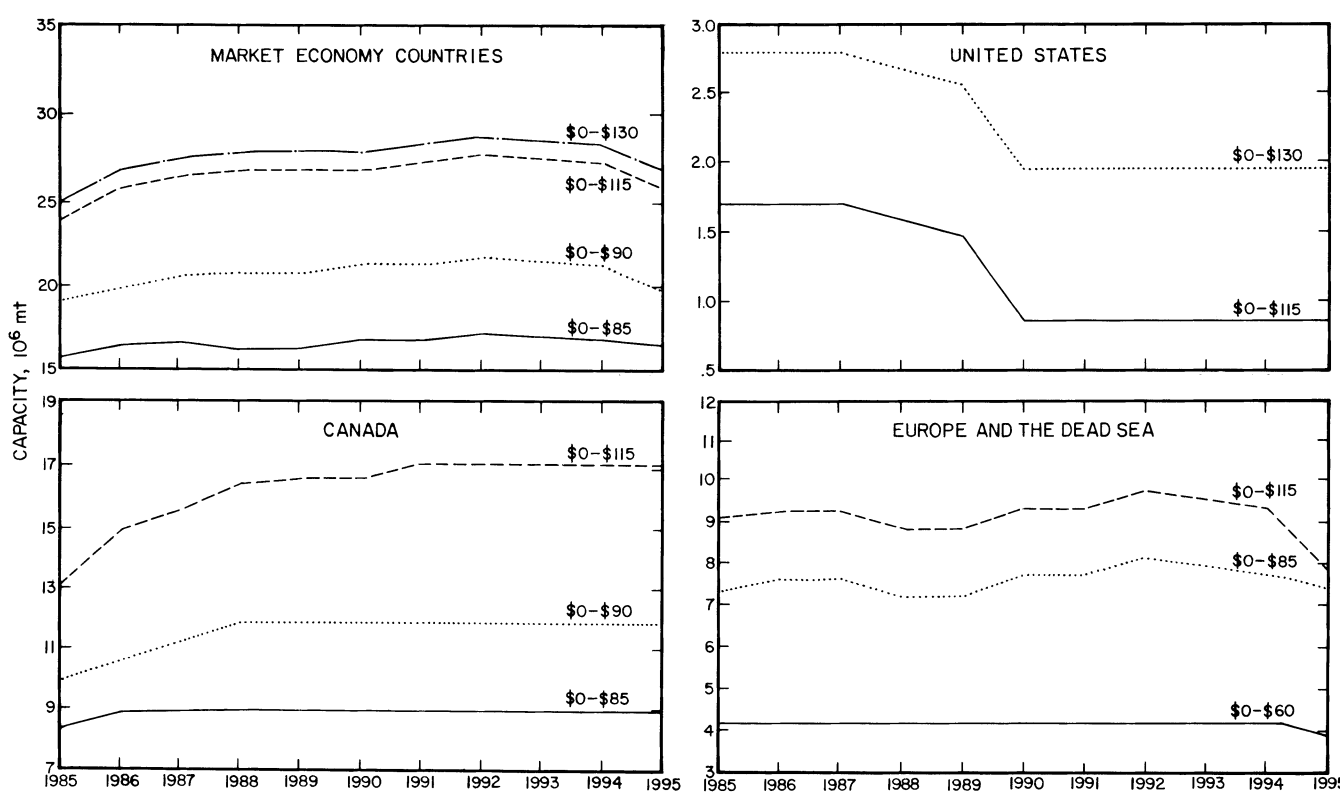

Figure 6.—Potential annual muriate production from producing mines in various MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

curves include a 15-pct DCFROR because it was assumed that over the long run all costs of production, including a 15-pct DCFROR, must be recovered.

These curves do not indicate a decline in potential production from currently estimated demonstrated resources of producing mines in Canada. However they indicate that potential production from currently producing U.S. mines may decline after 1986 and level off from 1990 through 1995. Potential production from producing mines in Europe and the Dead Sea area do not begin to decline until 1994.

In recent years, Canada has exported nearly two-thirds of its muriate production to the United States, and imports into the United States from Canada have represented two-thirds of domestic consumption. This analysis indicates that unless new, low-cost resources are discovered in the United States, the ratio of domestic production to domestic consumption will continue to decline. This decline should not be a problem for consumers in the United States because Canadian resources have the potential to supply all of projected demand in the United States.

Potassium Sulfate

The annual capacity of potassium sulfate from MEC producers is about equal to current production levels. At costs less than $192.90/mt, 1.8 million mt could be recovered from mines in the United States, Italy, and the Federal Republic of Germany. Potential capacities are not sufficient to meet increased world demand for potassium sulfate in the future. To supply increased demand, annual capacity at existing deposits will have to be expanded.

Potassium Magnesium Sulfate

Recent production of potassium magnesium sulfate and other minor potassium salts was 756,000 mt in 1984; this is near the tonnage potentially recoverable annually from producing mines in 1984 and through 2000. However, the costs associated with this tonnage are above the January 1985 market price of $65/mt.

CONCLUSIONS

Deposits included in this analysis contained demonstrated in situ potash resources of 11.7 billion mt K_2O equivalent; 82 pct of this is located in Canada, 11 pct in the Dead Sea, 4 pct in Western Europe, and 2 pct in the United States. Muriate of potash accounts for 94 pct of the potash potentially recoverable from resources located in MEC's, while 4 pct would be in the form of potassium sulfate.

World potash production was estimated to be 28.6 million mt contained K_2O from 13 countries during 1985. The U.S.S.R. accounted for almost 34 pct of this, Canada accounted for over 26 pct, and the United States accounted for less than 5 pct of world production.

Canadian producers have costs lower than U.S. underground mines when computed using either a 0- or a 15-pct DCFROR, but nonproducing deposits in Canada, which have lower costs than U.S. operations when computed using a 0-pct DCFROR, have much higher costs when at 15-pct DCFROR, as a result of the Saskatchewan reserve tax.

U.S. production and the percent of U.S. consumption supplied by this domestic production of muriate has been declining in recent years. Imports from Canada make up two-thirds of the potash consumed in the United States. This analysis indicates that potash mining in the United States will continue to decline over the long run. This decine will have a significant impact on potash producers, but it is not necessarily a problem for consumers in the United States because Canadian resources are readily accessible and have the potential to meet projected increases in demand.

Canada is the largest MEC potash producer. Canada also has large resources of potash. Canadian resources associated with producing mines are capable of meeting U.S. demand well into the future.

REFERENCES

1. Sullivan, D.E., and N. Michael. Availability of Potash—Market Economy Countries. A Minerals Availability Appraisal. BuMines IC 9084, 1986, 32 pp.
2. Searls, J.P. Potash. Ch. in Mineral Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 617-633.
3. ______. Potash. Ch. in BuMines Minerals Yearbook 1985, v. 1, pp. 775-786.
4. ______. Potash. Ch. in BuMines Mineral Commodity Summaries 1986, pp. 120-121.
5. Smith, G.I., C.L. Jones, W.C. Culbertson, G.E. Erickson, and J.R. Oyni. Evaporites and Brines. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 197-216.
6. Energy, Mines, and Resources, Canada. Potash, a Proposed Strategy. Miner. Bull. 194, 1982, p. 29.
7. Cox, J.L. The History and Future of Potash Production in the U.S. Paper in Proceedings, Raw Minerals Conference of the International Fertilizer Industry Association (Palm Springs, CA, Sept. 1984). Int. Fertilizer Ind. Assoc., Paris, France, 1984, v. I, pp. 34-51.

RARE-EARTH AND THORIUM OXIDES

INTRODUCTION

The rare-earth elements, or lanthanides, are 15 chemically similar elements with atomic numbers 57 through 71. They have been classified into two general groups: the light or cerium subgroup, comprising the first seven elements (atomic numbers 57-63), and the heavy or yttrium subgroup, comprising the elements with atomic numbers 64 through 71, as well as yttrium.

Important industrial applications of the rare earths include petroleum cracking catalysts, metallurgical (including iron and steel additives, alloys, and mischmetal), ceramics and glass (including polishing compounds and glass additives), and miscellaneous (including phosphors, electronics, nuclear energy, lighting, and research).

Because of the byproduct nature of a large percentage of world rare-earth production, and the fact that the rare-earth-bearing mineral monazite is produced as an integral part of the recovery process of the titanium minerals (rutile and ilmenite), the most practical investigative approach to assess rare-earth availability is in terms of its availability as a function of overall profitability of the properties evaluated. Consequently, availability results are presented as a function of a measure of profitability of each property, as indicated by its discounted-cash-flow rate of return (DCFROR). Availability is in terms of metric tons of rare-earth oxide (REO) shown at various DCFROR's. Thorium is an important constitutent in the mineral monazite and a brief discussion on thorium is included in the resource section of this chapter.

A large percentage of rare earths produced in the world is sold in concentrate form at or near the mine-mill site, and published market prices are generally available for mill concentrates. Because of the highly competitive nature of the industry, postmill processes and related costs are highly confidential and were not available for this evaluation. Consequently, this evaluation includes all costs associated with producing a marketable REO-bearing mill concentrate. Availability results show the amount of recoverable REO contained in the marketable concentrate.

Because nearly all properties do or would produce the REO-bearing mineral, usually monazite, as a relatively minor byproduct, it would be meaningless to present operating costs in terms of recovered REO. Therefore, mining and milling costs are presented in terms of dollars per metric ton of ore.

Most of the information presented in this chapter is an updated summary of Bureau of Mines Information Circular 9111 "Availability of Rare-Earth, Yttrium, and Related Thorium Oxides—Market Economy Countries. A Minerals Availability Appraisal" (*1.*)[1] Additional information on the domestic and foreign rare earths industry is available from other Bureau of Mines publications (*2-8*).

GEOLOGY AND RESOURCES

GEOLOGY

Monazite, bastnasite, and xenotime are the most important REO-bearing minerals. Heavy mineral sands occurring in modern placer deposits are the major source of monazite; the mineral normally is produced only as a byproduct of processing rutile, ilmenite, and zircon minerals. The placers are formed by the natural processes of weathering, transportation, and concentration at a site of accumulation of heavy minerals whose origin is a primary source rock. Beach deposits are the most significant commercial placers. The largest accumulations exist where a coastline is indented and the beach is gently sloping. Important deposits of this type occur in Australia, India, Brazil, the Republic of South Africa, and the United States. Similar deposits occur in southeast Asia, where small amounts of xenotime are recovered as a byproduct of tin mining.

An important source of rare earths is carbonatite deposits, igneous assemblages of primarily carbonate minerals occurring as intrusions associated with undersaturated alkali igneous complexes formed along major rift zones. The most significant market economy country (MEC) commercial carbonatite complex is at Mountain Pass, CA, which supplies much of the world's bastnasite and has recently accounted for nearly half of the world's annual REO production. Other carbonatite complexes, such as Palabora in the Republic of South Africa, are known to contain large amounts of REO-bearing minerals, but none has produced commercially significant quantities on a sustained basis. Additional information on the geology of rare-earth and thorium oxide deposits is available from the U.S. Geological Survey (*9-10*).

RESOURCES

Figure 1 compares the amount of demonstrated resources of contained REO in properties evaluated for this study with the reserve base. Important countries shown include the United States (which has the largest contained resources of any MEC), India, and Australia.

Table 1 shows resource information, by country, for the evaluated properties, including the total amount of

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

recoverable REO and thorium at the demonstrated level. Table 2 is a list of the properties evaluated for this study. The United States accounts for 59 pct of total recoverable REO in evaluated properties; producers, including the important Mountain Pass, CA, property, account for 46 pct of the total. Australia, the second leading annual MEC producer, accounts for 9 pct of the total, while India and Sri Lanka account for 24 pct; 21 pct of the total amount of recoverable REO is from Indian and Sri Lankan producers.

Thorium occurs in the important REO-bearing-mineral monazite. However, there currently is very little demand for thorium, and its presence in monazite is widely considered to be a nuisance because of the precautionary measures required while handling and processing thorium-bearing material. Should thorium demand increase in the future, monazite will be an important source. Based on average ThO_2 content of typical monazite from the various countries included in this evaluation, the total amount of ThO_2 contained in concentrate from the various countries is shown in table 1.

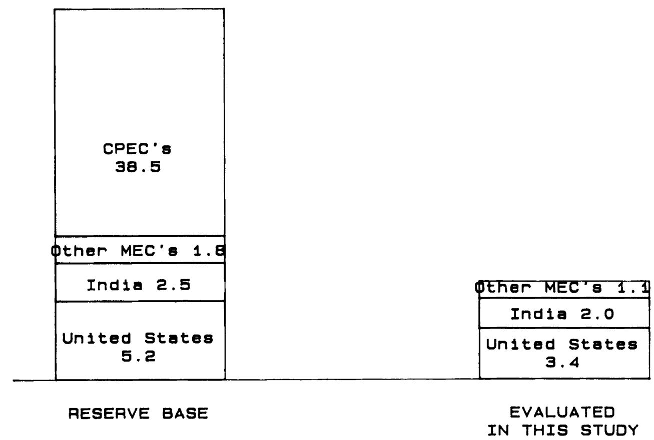

Figure 1.—Estimates of world rare-earth oxide resources (million metric tons of rare-earth oxide content).

Table 1.—Summary of MEC demonstrated REO and thorium oxide resources evaluated for this study, as of January 1985

Country and status	Number of properties	Ore treated, 10^6 mt	Feed grade, pct REO	Recoverable REO, 10^3 mt	Recoverable THO_2, mt
Australia:					
Producers	8	1,153	0.04	246	24,600
Nonproducers	5	1,218	.01	44	4,400
Total or av	13	2,371	.03	290	29,000
Brazil:					
Producers	2	3	.43	10	1,000
Nonproducers	3	2	.34	5	500
Total or av	5	5	.39	15	1,500
Canada: Nonproducers	3	241	.01	14	0
India and Sri Lanka:					
Producers	5	549	.37	695	69,500
Nonproducers	1	86	.19	115	11,500
Total or av	6	635	.35	810	81,000
Malawi and South Africa, Rep. of: Nonproducers	2	626	.07	214	21,400
United States:					
Producers	2	105	2.09	1,536	227
Nonproducers	7	876	.13	436	43,000
Total or av	9	981	.34	1,972	43,827
Grand total or av	38	4,859	.14	3,315	176,727

Table 2.—MEC REO properties included in this study

Country and property name	Owner	Current status[1]	Deposit type[2]	Mining method	Products[3]
Australia:					
Allied Eneabba	Allied Eneabba Ltd.	P	Pl	Strip level	R, I, L, Z, RE
Cable Sands	Cable Sands Pty. Ltd.	P	Pl	Dredge	I, R, Z, RE
Capel	Associated Minerals Consolidated Ltd. (AMC).	P	Pl	..do	I, R, L, SR, RE
Cataby	Metals Exploration, Alliance	Exp	Pl	..do	R, I, Z, RE
Cooloola	State of Queensland, Australian Government.	PP	Pl	..do	R, I, Z, RE
Eneabba	Associated Minerals Consolidated Ltd. (AMC).	P	Pl	Strip level	I, R, L, SR, RE
Fraser Island	Murphyores, Dillingham	PP	Pl	Dredge	R, I, Z, RE
Jurien Bay, Cooljarloo	Western Mining Corp.	PP	Pl	Strip level	R, I, L, Z, RE
Munmorah	Associated Minerals Consolidated Ltd. (AMC).	PP	Pl	Dredge	R, I, Z, RE
North Capel	State of Western Australia	P	Pl	Strip level	I, R, L, Z, RE
North Stradbroke (AMC)	Associated Minerals Consolidated Ltd. (AMC).	P	Pl	Dredge	R, I, Z, RE
North Stradbroke (CRL)	Consolidated Rutile Ltd.	P	Pl	..do	R, I, Z, RE
Yoganup Extended	Westralian Sands Ltd.	P	Pl	Open pit	I, R, L, Z, RE
Brazil:					
Alcobaca[4]	Brazilian Government	Exp	Pl	..do	RE, I, Z, R
Anchieta[4]	..do	P	Pl	..do	RE, I, Z, R
Aracruz[4]	..do	Exp	Pl	..do	RE, I, Z, R
Buena[4]	..do	P	Pl	..do	RE, I, Z, R
Serra[4]	..do	Exp	Pl	..do	RE, I, Z, R
Canada, Elliot Lake:					
Denison	Denison Mines Ltd.	P[5]	Hr	Room and pillar	U, RE
Quirke-Panel	Rio Algom Ltd.	P[5]	Hr	..do	U, RE
Stanleigh	..do	P[5]	Hr	..do	U, RE
India:					
Chavara (IRE)	India Rare Earths Ltd. (IRE)	P	Pl	Strip level	R, I, L, Z, RE
Chavara (KMML)	Kerala Minerals and Metals Ltd. (KMML)	P	Pl	Dredge	R, I, L, Z, RE
Manavalakurichi	India Rare Earths Ltd. (IRE)	P	Pl	Strip level	I, R, SR, Z, RE
Orissa-Chatrapur	..do	Dev	Pl	Dredge	R, I, L, Z, RE
Ranchi-Purulia[4]	Indian Government	Exp	PL	Strip level	RE, I, Z, R, S
Malawi: Kangankunde[4]	Lonhro Ltd.	Exp	Hr	Open pit	RE
South Africa, Rep. of:					
Richards Bay	Quebec Iron and Titanium, Ltd., Union Corp., Development Corp. of South Africa, Ltd.	P[5]	Pl	Dredge	S, R, Fe, Z, RE
Sri Lanka: Pulmoddai	Sri Lankan Government	P	Pl	Strip level	I, R, Z, RE
United States:					
Bear Valley	Bear Valley Industries	PP	Pl	Dredge	I, RE, G, C
Big Creek[4]	Several	PP	Pl	..do	RE, I, G, Z
Brunswick, Altamaha	Union Camp Corp.	Exp	Pl	..do	R, M, Z, RE
Gold Fork-Little Valley	Several	Exp	Pl	..do	I, Z, RE, Au, G
Green Cove Springs	Associated Minerals Consolidated Ltd. (AMC).	P	Pl	..do	R, I, L, Z, RE
Mountain Pass[4]	Molycorp Inc.	P	Hr	Open pit	RE
Oak Grove	Ethyl Corp.	Exp	Pl	Dredge	I, R, Z, RE
Powderhorn	Buttes Gas and Oil Co.	Exp	Hr	Open Pit	P, RE
Silica Mine	Tennessee Silica Sand	P[5]	Pl	..do	SS, I, R, L, Z, RE

[1] P, producer; PP, past producer; Exp, explored prospect; Dev, developing property.
[2] Pl, placer; Hr, hard rock.
[3] The first product listed was assumed to be the primary product for this study. Au, gold; C, columbium; Fe, magnetite; G, garnet; I, ilmenite concentrate; L, leucoxene concentrate; M, mixed ilmenite-leucoxene concentrate; P, perovskite concentrate; R, rutile concentrate; RE, REO concentrate; S, titanium slag; SR, synthetic rutile concentrate; SS, silica sand; Z, zircon concentrate; U, uranium.
[4] Do or could produce REO as the primary product.
[5] Producers that do not presently recover REO.

U.S. AND WORLD HISTORICAL PRODUCTION

Table 3 provides a summary of available world production data from Bureau sources in 5-yr increments between 1950 and 1985. World mine production was estimated to total 38,800 mt in 1985; it is difficult to estimate total world production for previous years, because figures for many important producers were unreported, not available, or withheld. For example, Indian production is reported as 2,245 mt in 1958, but Indian data are not available for any other year between 1948 and 1962, inclusive. Similarly, Brazil produced 1,621 mt in 1954, but data are not available for any other year from 1950 through 1957. Of the years shown, data for China are available only for 1985. U.S. production figures were withheld, to avoid disclosing proprietary data, for several years.

Table 3.—World REO production, 1950-85, metric tons REO

Country	1950	1955	1960	1965	1970	1975	1980	1985
Australia	728	118	221	1,406	2,664	2,706	8,624	8,250[e]
Brazil	NA	NA	628	358	1,386	871	1,362	1,100[e]
India	NA	NA	NA	1,525	2,181	1,797	1,634	2,200[e]
Malaysia	NA	152	26	423	995	1,972	1,180	2,000[e]
United States	418	664	W	W	W	[1]14,940	[1]15,986	[1]13,428
Other MEC	NA	5,504	472	705	229	639	454	340[e]
China	NA	NA	NA	NA	NA	NA	NA	10,000[e]
Other CPEC	NA	NA	NA	NA	NA	NA	2,451	1,500[e]

[e] Estimated. NA Not available. W Withheld.

[1] Comprises only the rare earths derived from bastnasite, as reported in Unocal Corp. (previously Union Oil of California) annual reports.

NOTE—Production data for 1950-1975 reported as monazite concentrate. In order to report figures in terms of REO content, a conversion factor of 0.60 was used.

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

Properties evaluated, including ownership, status, mining method, and products are shown on table 2.

Because commercial amounts of REO-bearing minerals occur in placers, veins, and igneous intrusive complexes, mining methods include placer (predominantly dredging), open pit, and underground. For heavy mineral operations recovering monazite as a byproduct, such as those in Australia and India, dredging is the most common mining method employed. Floating cutterhead dredges are the most commmon type of machinery used if the mineral sands are loose and occur at depths of less than 20 m. Preliminary concentrating occurs on the dredge, or on barges alongside, using Reichert cones, spirals, jigs, tables, and similar equipment.

Where floating dredges are not practicable (e.g., deposits that are several kilometers inland and water is in short supply), or the size, shape, and lithology of the ore body are impractical for dredges, other mining methods are employed. Draglines and front-end loaders, and trucks are used in most cases.

The only producing, open pit hard-rock operation evaluated for this study is Mountain Pass, CA, where bastnasite ore is produced from a tabular intrusive carbonatite ore body. Kangankunde, an undeveloped hard-rock property in Malawi, would probably also utilize open pit mining methods.

The only underground operations evaluated for this study were the Elliot Lake district uranium mines in Ontario, Canada. There, room-and-pillar methods are used to extract uranium ore from quartz pebble conglomerates. Monazite occurs as a secondary mineral in the uranium ore.

PROCESSING

Initial concentration of placer sands occurs in wet mills, with final concentration and mineral separation in dry mills. In some cases, prior to wet concentration, ore goes through a feed preparation stage (to wash clay particles) that can include a number of separate processes depending on the amount of clay and the throughput rate. A wet-screening stage is used to prepare suitably sized feed for the wet gravity concentrator.

Wet mills can be land based or floating. In a typical operation, ore from a dredge or slurry sump is pumped at 25 to 30 pct solids to the wet mill, where ore is fed into one or more stages of Humphreys spirals and/or Reichert cone concentrators that produce a preliminary heavy mineral concentrate. Rough, heavy mineral concentrate from the wet mill is transported, usually by truck or barge, to the dry mill for further processing.

Dry mills use various stages of magnetic, electrostatic, and gravity separation techniques to produce ilmenite, rutile, leucoxene, zircon, and monazite concentrates. The specific flowsheet of a dry mill depends on the type of ore and the heavy mineral assemblage to be recovered. In general, a dry mill separation process consists of high-tension electrostatic separators to separate conducting (including monazite) from nonconducting (zircon) minerals. The conductors are further separated using both dry, induced-roll, high-intensity mangetic separators and very high intensity, crossbelt magnetic separators into the ilmenite, rutile, and monazite concentrates.

The Mountain Pass operation utilizes flotation to process the bastnasite ore. Kangankunde, the other hard-rock property evaluated, would also use flotation methods. A unique beneficiation process was used at Denison Mines Elliot Lake operation in Canada, where a Y_2O_3-REO concentrate was produced as a byproduct from the barren uranium solution. The process included leaching, solvent extraction, and precipitation to produce a 63.5 pct concentrate, of which Y_2O_3 is the major component, averaging 40 pct in the concentrate.

PRODUCTION COSTS

Mine and mill operating costs for evaluated properties are shown in table 4. Costs are weight averaged on the basis of annual ore capacity, in terms of dollars per metric ton of ore. Although availability results are presented in terms of recovered REO in product (concentrate), it would not be useful to present costs in these terms, because nearly all properties produce REO as a byproduct of titanium mining.

Table 4.—REO mining and milling costs for selected MEC's

(All costs are in January 1985 U.S. dollars per metric ton phosphate rock on a weighted-average basis)

Country and status	Mine	Mill	Total
Australia:			
Producers	0.65	0.34	0.99
Nonproducers	.63	.41	1.04
Brazil:			
Producers	1.42	1.44	2.86
Nonproducers	1.55	1.50	3.05
India and Sri Lanka:			
Producers	.55	1.31	1.86
Nonproducers	2.09	4.44	6.53
Malawi and South Africa, Rep. of: Nonproducers	.45	.47	.92
United States:			
Producers	W	W	W
Nonproducers	.81	.92	1.73

W Withheld to avoid disclosing company proprietary data.

Mining and milling costs are not shown for the Elliot Lake, Canada, properties because they would use special leach, solvent extraction, and precipitation techniques to recover REO from a barren uranium leach solution. Only the marginal costs associated with treating the barren uranium solution were included in this evaluation.

The weighted-average mine operating cost for the eight Australian producers is \$0.65/mt. Because the costs are weight averaged according to annual capacity, they are heavily influenced by the low unit costs at the North Stradbroke operation of Consolidated Rutile Ltd., which is expected to produce nearly 40 million mt ore annually by the late 1980's. The average mill cost is \$0.34/mt for all Australian producers.

The five Australian nonproducers that could recover byproduct monazite have a weighted-average mining cost of \$0.63/mt, and a milling cost of \$0.41/mt ore. These figures are strongly influenced by the costs at Cooloola, which is a past producer that terminated production when the area became part of the Cooloola National Park. All but Jurien Bay, Cooljarloo, would be dredge operations.

The four Indian producers and one Sri Lankan producer together have a weighted-average mining cost of \$0.55/mt, and a milling cost of \$1.31/mt. The two Brazilian producers, Anchieta and Buena, both of which are strip level operations with relatively small capacities (40,000 to 80,000 mt/yr), have weighted-average mining and milling costs of \$1.42/mt and \$1.44/mt ore, respectively. The Brazilian nonproducers could be brought into production at comparable operating costs (\$1.55/mt mine, \$1.50/mt mill); however, the capital expense necessary to build the mill and develop the deposits, and especially the high transportation costs associated with shipping the concentrates to port or process plant in Sao Paulo (as much as 1,200 km distant) render the Brazilian properties relatively expensive in terms of the total cost of production. The same is true for the undeveloped Indian property, Ranchi-Purulia.

Only one large-scale rare-earth property, Mountain Pass, is operating in the United States; consequently, operating costs are not disclosed. Because Mountain Pass is an open pit operation in a hard-rock deposit, the operating costs per metric ton of ore are high relative to heavy minerals sand mining operations; however, the deposit has a very high grade (12 pct bastnasite with 6 to 7 pct REO content) compared with heavy minerals sand deposits, so that the operating costs in terms of recovered product are relatively low.

The weighted-average costs for the five U.S. nonproducers that are heavy minerals sand deposits (Big Creek, Bear Valley, Gold Fork-Little Valley, Brunswick-Altamaha, and Oak Grove) are \$0.81/mt for mining, \$0.92/mt for milling. These costs are comparable to those for other world properties. However, costs of transporting concentrates to existing processing plants for titanium and rare-earths would be sufficiently high, especially in the case of the Idaho deposits, to place them at a significant disadvantage relative to several other world properties. Transportation costs for the Idaho properties would be four to eight times higher than those for the western Australia properties. Additionally, the Big Creek and the Gold Fork-Little Valley (Idaho) deposits have multiple landowners, and the actual costs associated with developing a unitized operation are not known.

AVAILABILITY

Of the 38 properties evaluated for this study, 9 do or would produce REO as the primary product. Five of the nine properties (i.e., the Brazilian properties) would be considered to be titanium properties from a revenues standpoint. However, several deposits are mined primarily to obtain the thorium contained in monazite for proposed use in nuclear power generation.

Only Mountain Pass derives its total revenues from REO; therefore, the profitability of the property is singularly dependent on the REO market. Because of the by-product status of monazite, most properties included in this evaluation depend largely on the titanium market for their economic health. Australian producers on average derive 3 pct of their total revenues from monazite. Similarly, Indian and Sri Lankan producers on average derive 7 pct of total revenues from monazite.

TOTAL RECOVERABLE

Figure 2 shows total potential availability by property status (producer and nonproducer), categorized by DCFROR range. Only 21 pct (about 705,000 mt) of the total recoverable 3.315 million mt of REO in all properties evaluated is contained in properties that could not realize a positive DCFROR at the January 1985 market prices for commodities that are or could be produced from those properties. About 75 pct (2.487 million mt) of total recoverable REO in all properties evaluated is contained in producing properties. A logical supposition is that the majority of these properties are operating because they can consistently produce at a profit. A notable exception is the Brazilian operations, which are owned by Nuclemon, a government entity that has motives other than immediate profit or benefit (i.e., the need for thorium produced as a result of processing or refining monazite).

About 126,000 mt, or less than 4 pct of the total recoverable REO in all properties analyzed, is contained in nonproducing properties that could realize a positive DCFROR. Included here are the Silica Mine, TN, and Richards Bay, Republic of South Africa, which are producing operations that do not currently recover monazite, but for which the marginal costs of recovering monazite were included.

ANNUAL CAPACITY

Figure 3 shows the amount of REO potentially available on an annual basis from producers at various DCFROR ranges. Because the general approach for this study was to evaluate the properties at their production capacity over the life of each property, the annual curves present the total potential availability for each year shown.

Annual availability from all producing properties including a DCFROR greater than 0 pct varies from a low of 46,500 mt in 1985, to a peak of 51,500 mt in 1988 through 1992, and declines to 49,600 mt by the year 1995. The increase between 1986 and 1988 is because of planned expansions that were included in the evaluation.

Of the total amount potentially available each year, only 6,600 mt is contained in properties that would not receive at least a 20-pct DCFROR at the January 1985 market prices for REO concentrate. Total estimated MEC production in 1985 was 27,300 mt of REO; capacity at producing properties is more than adequate to sustain 1985 production levels for at least the next 10 yr.

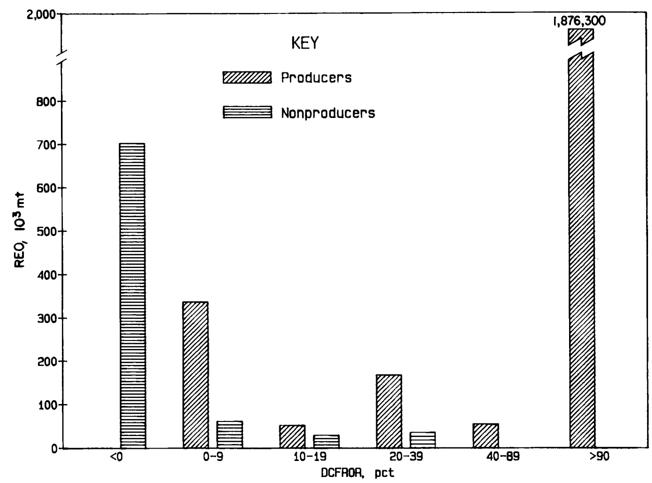

Figure 2.—Total recoverable rare-earth oxide by production status and DCFROR.

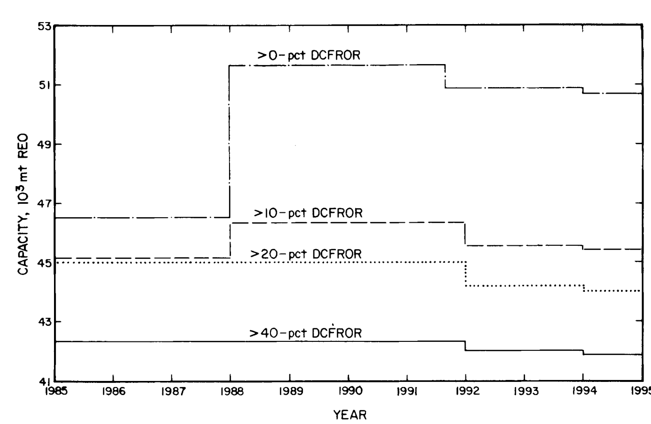

Figure 3.—Potential annual rare-earth oxide production from producing mines in MEC's at selected DCFROR ranges, 1985-95 (January 1985 U.S. dollars).

CONCLUSIONS

Resource information for REO properties worldwide is difficult to obtain, because rare earths are used in highly advanced technological applications, and historical demand has been relatively small compared with other mineral commodities. Furthermore, demand has been fulfilled by a relatively small number of properties, most of which (e.g., Australian operations) produce small amounts of REO minerals that make a relatively insignificant (byproduct) contribution to the revenues of titanium operations. Mountain Pass is the only operation that is important as a principal producer of rare earths. It did not become a prominent producer until the mid-1960's, when demand for rare earths began to increase at a substantial rate. The industry in many ways can be considered to be in its infancy, and demand is expected to increase as more uses are found for rare earths and as new applications are developed.

There are sufficient resources of rare earths in producing deposits to sustain present production levels at least through the end of the century, and probably well beyond. However, because of the byproduct status of monazite from several important producing properties, the supply of certain of the rare-earth elements depends largely on the titanium market. Should the titanium market become unfavorable to heavy mineral sands producers, the supply of many of the heavy subgroup of rare earths could be a matter of some concern.

The availability and supply of the light subgroup of rare earths, which are relatively abundant in the mineral bastnasite, are assured because of the Mountain Pass, CA, operation, which has sufficient resources to last for many years. Given favorable future demand and prices, Mountain Pass should continue to be a viable operation for the foreseeable future.

REFERENCES

1. Anstett, T.F. Availability of Rare-Earth, Yttrium, and Related Thorium Oxides—Market Economy Countries. A Minerals Availability Appraisal. BuMines IC 9111, 1986, 19 pp.

2. Hedrick, J.B. Rare-Earth Elements and Yttrium. Ch. in Mineral Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 647-664.

3. ______. Thorium. Ch. in Mineral Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 835-846.

4. ______. Rare-Earth Minerals and Metals. Ch. in BuMines Mineral Yearbook 1985, v. 1, pp. 791-803.

5. ______. Thorium. Ch. in BuMines Minerals Yearbook 1985, v. 1, pp. 955-962.

6. ______. Rare-Earth Metals. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 126-127.

7. ______. Thorium. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 164-165.

8. ______. Yttrium. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 176-177.

9. Adams, J.W., and M.H. Staatz. Rare-Earth Elements. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 547-556.

10. Staatz, M.H., and J.C. Olson. Nuclear Fuels. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 468-476.

SILVER

INTRODUCTION

Silver is a metal critical to the production of many manufactured products owing to its high electrical conductivity, resistance to oxidation, and strength over a wide range of temperatures. In addition, the light-sensitive properties of some silver compounds are essential to the manufacture of photographic film, while other silver compounds find important uses in medicine. Silver is also a precious metal which, like gold, is hoarded as a hedge against inflation.

This chapter addresses the potential availability of demonstrated silver resources from 447 mines and deposits in 45 market economy countries (MEC's) comprising five major economic categories of deposits: (1) predominantly silver deposits, (2) predominantly zinc deposits, (3) predominantly lead deposits, (4) predominantly copper deposits, and (5) predominantly gold deposits.

Any definition of a predominantly silver deposit employed for the purpose of longrun economic analysis is subject to some ambiguity and exception. This chapter applies two-tier grade and revenue criterion to define a predominantly silver deposit: (1) deposits containing less than 3.5 pct combined payable lead and zinc in association with recoverable silver, and (2) those deposits of the first group where silver revenues provide greater than 60 pct of the total revenue requirements under both $5/tr oz and $10/tr oz silver price scenario. Sixty-six mines and deposits meet the first criterion and 36 meet both criteria.

Following the most prevalent world industry practice, silver grades are reported as grams per metric ton, and total contained and recoverable silver from a mine or deposit are reported in troy ounces (1 tr oz equals 31.1035 g). For predominantly silver mines and deposits, the economic analyses address the total cost of extracting silver in the form of at least 99.6-pct Ag product that is produced either by the refining of silver and gold dore bullions or the smelting and refining of copper, lead, zinc, or silver concentrates. For byproduct silver mines and deposits, silver availability is expressed as a function of the total cost of the primary commodity.

Most of the information presented in this chapter is an updated summary and extension of Bureau of Mines Information Circular 9090 "Primary Silver Availability—Market Economy Countries. A Minerals Availability Appraisal" (*1*).[1]

GEOLOGY AND RESOURCES

GEOLOGY

The reader is referred to the copper, lead and zinc, and gold chapters for discussions of the general geology of the deposits of those commodities. This section summarizes the general geologic aspects of what are considered to be predominantly silver properties.

In the 1973 U.S. Geological Survey (USGS) study (*2*), the silver chapter describes nine different types of deposits where, according to the USGS, silver was a major constituent. These nine categories are listed as follows:

1. Epithermal veins, lodes, and pipes.
2. Epithermal disseminated and breccia deposits.
3. Epithermal silver and manganese deposits.
4. Epithermal silver-lead-zinc replacement deposits.
5. Epithermal silver-copper-barite deposits.
6. Mesothermal silver-lead-zinc-copper deposits.
7. Mesothermal cobalt-silver, cobalt-uraninite-silver-zeolite deposits.
8. Sandstone silver deposits.
9. Sea-floor muds and hot-springs deposits.

In the context of discussion in reference 2, the term epithermal is defined as (1) formed within a few thousand feet of the surface, under slight load (pressure) and generally in brittle rocks where open fractures and cavities are abundant, (2) formed by deposition of silver-bearing minerals from hot ascending solutions in areas of volcanic activity, and (3) few deposits are known to extend to depths of more than 610 m (2,000 ft) below the surface, with most confined to less than 305 m (1,000 ft) below the outcrop.

Mesothermal deposits, by contrast, are formed at moderate depths (600-3,600 m by one definition) under moderate pressures, and the deposition of silver-bearing minerals occurs from hydrothermal solutions.

Of the 36 predominantly silver properties that met the two-tier criterion, 29 can be classified as epithermal deposits. As many as 23 of the 29 can be said to be intimately associated with volcanic activity and only one has an indicated extension to a depth greater than 1,000 m below the surface. The other seven properties can all be classified as mesothermal deposits. Six of the seven can be put into the mesothermal silver-lead-zinc-copper deposit category and the seventh is a mesothermal cobalt-silver deposit.

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

By definition, the vein-type deposits in the mesothermal classification occur at great depths and, not surprisingly, have the highest mining cost structure of the predominantly silver deposits. The mines and deposits in the epithermal classification show a much wider range in structural occurrence. For example, of all 29, 17 have either vein or lode (ore shoot) occurrence, there is 1 pipelike occurrence, 3 stockwork or lens-type occurrences, 4 breccia occurrences, 2 combination breccia and fracture vein occurrences, 1 disseminated occurrence, and 1 silver-lead-zinc replacement occurrence. Of the 10 properties consisting of pipelike, stockwork or lens-type, breccia or combination breccia-fracture vein mineralization (all or which are more economic to mine than the vein-type occurrences), 6 are located in Mexico, 2 are located in Peru, 1 is located in Morocco, and only 1 is located in the United States.

There are no examples of the epithermal silver-copper-barite deposits, the sandstone silver deposits, or the seafloor muds and hot spring deposits in the 36 "true" predominantly silver deposits analyzed.

RESOURCES

Figure 1 shows the estimated world reserve base for silver (*3*) and the estimated total recoverable silver potentially available from the 447 MEC mines and deposits analyzed in this study. The reserve base estimate is based on contained in situ silver while the recoverable values reflect expected losses due to mining, beneficiation, smelting, and refining. The recoverable silver estimate for the MEC mines and deposits evaluated in this study is 6.6 billion tr oz. A detailed breakdown of these resources, by predominant commodity and country, is given in table 1. Table 2 is a list of the predominantly silver properties evaluated for this study. As shown, most silver is available as a byproduct of copper and zinc mines. The United States, Mexico, Canada, Australia, and Peru are the five largest countries in terms of total available silver. Of these five, only Australia is deficient in the predominantly silver category.

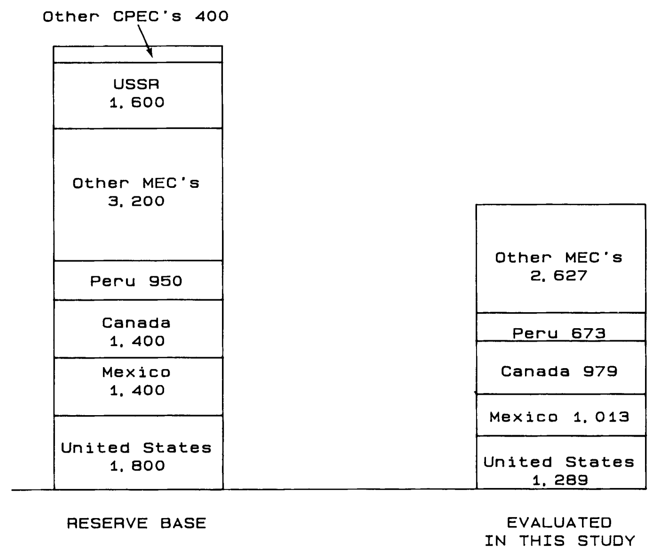

Figure 1.—Estimates of world silver resources (billion troy ounces).

Table 1.—Summary of MEC recoverable silver demonstrated resources, by predominant commodity, evaluated for this study, as of January 1985, thousand troy ounces

Country	Silver Resources	Silver No. of deposits	Lead Resources	Lead No. of deposits	Zinc Resources	Zinc No. of deposits	Copper Resources	Copper No. of deposits	Gold Resources	Gold No. of deposits	Total Resources	Total No. of deposits
United States	479,268	22	71,494	9	300,367	13	426,152	34	11,257	16	1,288,538	94
Mexico	464,421	14	27,203	2	353,273	17	167,751	4	—	—	1,012,648	37
Canada	61,594	8	—	—	637,380	25	274,621	21	5,782	17	979,377	71
Australia	—	—	36,672	1	709,000	12	172,088	6	1,239	7	918,999	26
Peru	192,361	11	—	—	190,303	16	290,527	9	—	—	673,191	36
Chile	—	—	—	—	—	—	432,933	9	16,478	1	449,411	10
South Africa, Rep. of	44,704	1	66,765	1	37,663	1	1,832	1	64,700	41	215,664	45
Portugal	—	—	—	—	107,321	1	16,362	1	—	—	123,683	2
India	—	—	—	—	84,676	4	31,720	1	—	—	116,396	5
Argentina	—	—	—	—	17,017	1	70,421	2	—	—	87,438	3
Panama	—	—	—	—	—	—	83,541	1	—	—	83,541	1
Papua New Guinea	—	—	—	—	—	—	70,139	3	—	—	70,139	3
Japan	—	—	—	—	38,275	8	27,879	1	—	—	66,154	9
Spain	36,382	2	946	1	24,179	2	4,287	2	—	—	65,794	7
Sweden	295	1	6,697	1	29,630	2	28,785	2	—	—	65,407	6
Philippines	—	—	—	—	—	—	46,777	16	1,682	3	48,459	19
Yugoslavia	—	—	—	—	—	—	44,959	3	—	—	44,959	3
Morocco	19,387	3	17,039	5	6,310	1	689	1	—	—	43,425	10
Greece	—	—	—	—	37,981	2	—	—	—	—	37,981	2
France	9,498	1	—	—	14,085	4	—	—	—	—	23,583	5
Others[1]	16,081	3	10,423	1	64,639	13	62,842	14	12,714	22	166,699	53
Total	1,323,991	66	237,239	21	2,652,099	122	2,254,305	131	113,852	107	6,581,486	447

— Indicates no recoverable silver, mines, or deposits.

[1] Includes Italy and Finland with primary silver mines; Honduras, Namibia, Federal Republic of Germany, Italy, Zaire, Bolivia, Ireland, Burma, Greenland, Algeria, and Zambia with silver in predominantly lead-zinc deposits; Namibia, Turkey, Finland, Indonesia, Brazil, Norway, Malaysia, Fiji, Iran, Oman, Pakistan, and Zimbabwe with silver in predominantly copper deposits; and Dominican Republic, Bolivia, Brazil, Taiwan, Zimbabwe, and Colombia with silver in predominantly gold deposits.

Table 2.—MEC predominantly silver properties included in this study

Location and property name	Owner	Mine type[1]	Current status[2]
Canada:			
Abcourt-Barvue	Abcourt Silver Mines-Noranda	U	N
Agnico-Eagle (Silver Division)[3]	Agnico-Eagle Mines Ltd	U	P
Beaverdell[3]	Teck Corp	U	P
Detour Project	Selco Mining Corp. Ltd	S	P
Goldstream	Noranda Mines Ltd	C	N
Silverfields (Teck)[3]	Teck Corp	U	P
Terra[3]	Various owners	U	P
United Keno Hill[3]	United Keno Hill Mines Ltd.—Falconbridge Ltd	C	P
Finland:			
Pyhasalmi	Outokumpu Oy	U	P
Vihanti	..do	U	P
France: L'Argentiere	Penarroya	U	N
Italy: Funtana Raminosa	SAMIM	U	P
Mexico:			
Angangueo[3]	Impulsora Minera de Angangueo	U	P
Avino[3]	Cia. Minera Mexicana de Avino	C	P
Huautla[3]	Rosario Mexico S.A. de C.V	U	P
La Colorada[3]	Minera Victoria Eugenia	U	P
La Negra	Industrias Penoles S.A. de C.V	U	P
Lampazos[3]	Minera Lampazos S.A. de C.V	U	P
Las Torres[3]	Cia Minera Las Torres	U	P
Parral Unit[3]	Industrial Minera Mexico S.A	U	P
Real de Angeles	Minera Real de Angeles	S	P
Real del Monte y Pachuca[3]	Cia Real del Monte y Pachuca	U	P
San Geronimo	Industrias Luismin	U	P
Santa Maria de La Paz	Minera Santa Maria de La Paz	U	P
Tayoltita	Industrias Luismin	U	P
Tocayos	Minera Victoria Eugenia	U	P
Morocco:			
Bougaffer	SODECAT (100 pct BRGM)	C	P
Imiter[3]	Societe Miniere Imiter	U	P
Zgounder[3]	BRGM-ARMICO	U	P
Peru:			
Alpamarca[3]	Sindicato Minero Rio Pallanga	S	P
Arcata[3]	Minas de Arcata S.A	U	P
Berenguela	Minero Peru	S	N
Caylloma[3]	Compania Minera de Caylloma	C	P
Julcani[3]	Cia. de Minas Buenaventura S.A	C	P
Morococha	Centromin	U	P
Orcopampa[3]	Cia. de Minas Buenaventura S.A	U	P
Quiruvilca	Corporacion Minera Nor-Peru	U	P
San Genaro[3]	Cia. Minera Castrovirreyna S.A	U	P
Sayapullo	Cia. Minera Sayapullo S.A	U	P
Uchucchucua[3]	Cia. de Minas Buenaventura S.A	U	P
South Africa, Republic of: Black Mountain	Phelps Dodge-Goldfields of South Africa	U	N
Spain:			
Aznalcollar	Sociedad Andaluza de Piritas	S	P
Cartagena	Penarroya	S	P
Sweden: Vassbo-Guttusjo	Boliden Metall AB	U	P
United States:			
Alaska: Golden Zone	Enserch Exploration, Inc	U	N
Arizona: Silver-Eureka District[3]	Orbey Minerals-New Jersey Zinc	S	N
Colorado:			
Bulldog Mine[3]	Homestake Mining Co	U	P
Revenue-Virginius[3]	Revenue-Virginius Mines Co	U	N
Idaho:			
Clayton Silver Mine[3]	Clayton Silver Mines	U	P
Crescent Mine[3]	Bunker Hill Ltd	U	P
Delamar	Mapco Minerals-Superior Oil	S	P
Galena[3]	Callahan-Hecla Mining Co	U	P
Lucky Friday Mine[3]	Hecla Mining Co	U	P
Sunshine[3]	Sunshine Mining Co	U	P
Maine: Big Hill	Scintilore Explorations Ltd	S	N
Montana:			
Black Pine[3]	Inspiration Consolidated Copper Co	U	P
Butte District Zinc	Anaconda Copper Co. (ARCO)	U	N
Flat Head[3]	Coca Mines	S	N
Troy	ASARCO	U	P
Nevada:			
Candelaria[3]	Nerco Minerals-Coca Mines	S	P
Sixteen-to-One[3]	Sunshine Mining Co.-Mid Continent	U	P
Taylor Silver Mine	Silver King Mines-Agnew Enterprises	S	P
Ward Mountain	Gulf Oil-Silver King Mines	U	N
New Mexico: St. Cloud Mine[3]	St. Cloud Mining Co.	U	P
Utah:			
Escalante[3]	Ranchers Exploration and Development	U	P
Trixie Mine	Sunshine Mining Co	U	N

[1] U, underground; S, surface; C, combined.
[2] P, producer; N, nonproducer.
[3] 1 of 33 producers and 3 nonproducers in which silver revenue equals at least 60 pct of total revenue at both $5/tr oz and $10/tr oz.

U.S. AND WORLD HISTORICAL PRODUCTION

Figure 2 shows annual mine production of silver in 5-yr intervals from 1950 through 1985 for the five largest MEC producers and the two largest centrally planned economy country (CPEC) producers. In 1950, the five MEC's (Mexico, Peru, the United States, Canada, and Australia) accounted for 68.3 pct of total world production and the two CPEC's (the U.S.S.R. and Poland) accounted for an additional 11.9 pct. In 1970, immediately following U.S. steps of the late 1960's to demonetize silver, the same five MEC's accounted for 65.8 pct of total world production, and the U.S.S.R. and Poland had increased their percentage share to 13.9 pct. As of 1985, more than a decade and a half after demonetization, these five MEC's have seen their share of total world production slip to 59.1 pct while the U.S.S.R. and Poland have increased their share to 17.5 pct.

Since 1970, total annual world mine production has increased from 301 million tr oz to 394 million tr oz, a 30.9 pct increase, with 75.9 pct of the increase coming from five of the seven countries shown in figure 2. Production in both the United States and Canada has moderately decreased since 1979. The largest increases to MEC annual output since 1970 have occurred in Mexico, 20.3 million tr oz; Peru, 17.0 million tr oz; and Australia, 6.1 million tr oz, while the U.S.S.R. has increased its annual output by 9.1 million tr oz and Poland by 18.1 million tr oz.

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

Table 2 lists the 66 properties classified as predominantly silver. Table 3 summarizes the breakdown of the total mining capacities (on an ore tonnage basis) for the 14 Mexican, 10 Peruvian, and 14 U.S. predominantly silver producers included in the analysis. Four items of importance should be noted in further descriptions.

1. Over 54 pct of the ore capacity at the predominantly silver producers in Mexico comes from only two surface mining operations. Equivalent values are 40 pct from three surface mines in the United States and only 23 pct from one full and one partial surface mining operation in Peru. By way of comparison, the two Mexican surface mines are responsible for an estimated 1985 output of 9,125 million tr oz of silver, the three U.S. surface mines for an estimated 3,865 million tr oz in 1985, and the two Peruvian operations should have produced about 800,000 tr oz of silver from surface mining operations in 1985.
2. The majority of the U.S. underground ore capacity (64.4 pct) is represented by the large-scale Troy silver-copper mine in western Montana, with 15.1 pct represented by the four Coeur d'Alene mines in Idaho and 10.5 pct in six other small-scale underground mines.
3. Only about one-third of the Mexican underground capacity represents the use of high-cost, cut-and-fill mining methods while nearly all of the Peruvian underground ore capacity uses the cut-and-fill method.
4. With the exclusion of the large-scale Troy mine from the comparision, the Mexican underground mines have an average ore capacity of about 520,000 mt/yr, the Peruvian mines about 279,000 mt/yr, and the four Coeur d'Alene and six small-scale U.S. mines average only about 151,000 mt/yr of ore.

Table 3.—Annual ore capacities, by mining method, for predominantly silver producers in Mexico, Peru, and the United States

Mining method	Mexico		Peru		United States	
	10^3 mt/yr	pct	10^3 mt/yr	pct	10^3 mt/yr	pct
Underground	3,325	45.6	2,155	77.4	4,226	59.6
Surface	3,960	54.4	630	22.6	2,862	40.4
Total	7,285	100.0	2,785	100.0	7,088	100.0

PROCESSING

Ore from the four Coeur d'Alene District producers is concentrated using flotation methods. Recoveries of silver are high (greater than 90 pct); however, only one of the four producers recovers significant quantities of byproducts. Of the other 10 U.S. producers, 5 use conventional flotation techniques to produce either bulk or selective concentrates and 5 use cyanide leaching (one heap leach and four agitation leach) to extract the precious metals from the ore, with subsequent Merrill-Crowe deoxygenation-zinc dust precipitation and smelting of the zinc dust precipitate with fluxes to produce silver-gold dore bullions.

Of the 24 Mexican and Peruvian producers, 22 were using bulk and/or selective flotation to treat their ores. One Mexican operation used only cyanidation (for producing coproduct gold and silver), and another Mexican operation was sending approximately one-third of its ore to a selective lead-zinc flotation unit with the other two-thirds going to a cyanidation mill. The 13 individual flotation units in Mexico were producing a total of 23 different concentrates, compared with the 20 concentrates produced by the 10 mills in Peru.

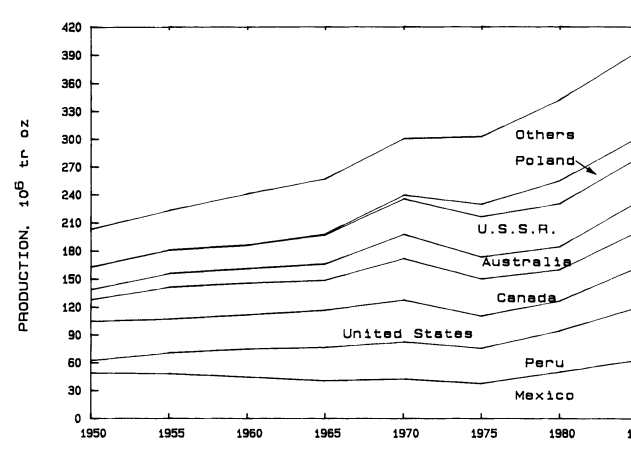

Figure 2.—World silver production, 1950-85.

In terms of overall silver recoveries in beneficiation, the estimates for the 14 mills in Mexico ranged from 70.0 to 95.3 pct, compared with 74.2 to 94.6 pct in Peru.

Of the other 15 MEC predominantly silver producers, only 3 were indicated to be using cyanidation and then only as a complement to flotation. One was indicated to produce a fair amount of its total silver production in the form of dore bullion.

Of the eight U.S. nonproducers included in the study, three are proposed to require a combination of cyanidation-flotation, and five are proposed to utilize flotation exclusively. The three cyanidation-flotation mills would have capacities ranging from 265,000 to 490,000 mt/yr, and the five flotation mills are proposed to have a wide range of ore capacities, from 54,300 to 1,797 million mt/yr.

All four of the nonproducing foreign deposits would utilize flotation methods for ore treatment.

The silver contained in the concentrates produced at the predominantly silver producers must be sent to an appropriate smelter or refinery for the further extraction of silver into marketable bullion forms. For the analyzed properties, the various concentrates produced represent a variety of bulk concentrates, lead concentrates, zinc concentrates, and so-called silver concentrates.

PRODUCTION COSTS

Table 4 and figure 3 present production cost estimates per troy ounce of recoverable silver for the 53 predominantly silver producing operations analyzed. The costs are shown on a weighted-average country basis for all major cost categories.

The table clearly indicates that the main cost advantages at the 14 Mexican producers are in the mine operating cost and the byproduct credit categories, with the result that net operating cost (before the recovery of capital and payment of taxation obligations) is $2.37/tr oz less than the 14 U.S. producers and $4.17/tr oz less than the 10 Peruvian producers. The total production cost advantage of the Mexican producers is $1.01/tr oz less than the U.S. producers and $4.51/tr oz less than the Peruvian producers at the 0-pct discounted-cash-flow rate of return (DCFROR) level, and $1.08 and $5.85/tr oz, respectively, at the 15-pct DCFROR level.

For perspective, it must be remembered that the estimated 1985 production capacities by country of the operations included in table 4 represent 63 pct of total 1985 U.S. mine production of silver, 48 pct of Mexican production, and only 26 pct of Peruvian production. Remaining silver production for these countries is as a byproduct of zinc, copper, lead, or gold production. The January 1985 U.S. dollar cost advantage, shown in table 4 for Mexico vis-a-vis the United States, has narrowed somewhat from the January 1984 estimates (see reference 1), which reflects the combined effects of Mexican inflation and changes to the Mexican peso exchange rate relative to the U.S. dollar.

Of particular interest in figure 3 is the very high amount of byproduct and coproduct credits for the six Canadian and the nine "other" producers. This is an indication that this group of properties includes a number of borderline byproduct and coproduct silver properties. To further distill the list, a second-tier criterion based on the percentage of total revenues that is provided by silver revenues was applied. The properties were reevaluated to determine which receive more than 60 pct of total revenues from silver production under both $5/tr oz and $10/tr oz silver price scenarios (with fixed 1985 prices for the other payable commodities).

The result was identification, as indicated in table 2, of 33 producing and 3 nonproducing properties, from the original list of 53 producing and 13 nonproducing properties. These 36 properties can probably be considered to represent the true predominantly silver properties. Table 5 and figure 4 show the reevaluated weighted-average production costs for the 33 producing operations (11 in the United States, 8 in Mexico, 7 in Peru, 5 in Canada, and 2 in Morocco) that are predominantly silver mines based upon the revenue criterion.

Comparison of table 5 with table 4 shows that the major result of this reclassification is that the very high byproduct credits for Canada have been nearly eliminated, and byproduct credits for the United States, Canada, and Peru declined 45.2, 37.7, and 53.1 pct, respectively. However, the overall net operating costs for the United States, Mexico, and Peru did not change significantly (plus $0.18/tr oz, minus $0.63/tr oz, and minus $0.50/tr oz, respectively). The same was true for the total production costs at the 0-pct DCFROR level (minus $0.09/tr oz, minus $0.70/tr oz, and minus $0.88/tr oz, respectively) and at the 15-pct DCFROR level (minus $0.25/tr oz, minus $0.98/tr oz, and minus $0.99/tr oz, respectively).

It should be noted that these 33 producers, combined, contain only 701.7 million tr oz of recoverable silver with 80.3 pct in only 11 mines. The U.S. operations contain 40.8 pct of the total, the Mexican operations 34.8 pct, and the Peruvian operations 18.7 pct. This demonstrates that true predominantly silver producers account for only a small percentage of total recoverable silver (10.5 pct) and represent a lesser degree of influence on the economics of silver production and the silver market of the MEC's relative to byproduct silver production from predominantly zinc or copper mines.

Table 4.—Silver production costs for 53 predominantly silver producing mines in selected MEC's

(All costs are in January 1985 U.S. dollars per troy ounce of recoverable silver on a weighted-average basis)

Country	Operating cost		Other operating cost[1]	Byproduct credit	Net operating cost	Recovery of capital[2]	0-pct DCFROR		15-pct DCFROR		
	Mine	Mill					Taxes and royalties[3]	Total cost[4]	Taxes and royalties[5]	Return on investment[6]	Total cost[7]
United States . .	4.79	2.09	1.30	1.99	6.19	1.03	0.15	7.37	0.36	0.77	8.35
Mexico	3.88	1.59	2.04	3.69	3.82	1.60	.94	6.36	1.35	.50	7.27
Peru	5.86	1.55	2.71	2.13	7.99	2.22	.66	10.87	1.73	1.18	13.12
Canada	6.26	4.05	9.83	16.70	3.44	3.93	.82	8.19	5.08	5.41	19.04
Others[8]	9.29	12.46	13.91	28.42	7.24	4.01	.17	11.42	.84	2.35	14.44

[1] Includes cost of smelting, refining, and transportation of flotation concentrates or silver dore bullion.
[2] Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure as of January 1985, and investments required over the life of the operation.
[3] Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 0-pct DCFROR.
[4] Equal to the sum of net operating costs, taxation, and capital recovery determined at a 0-pct DCFROR.
[5] Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 15-pct DCFROR.
[6] The revenue increase per troy ounce necessary to obtain a 15-pct DCFROR.
[7] Equal to the sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery *plus* the per troy ounce increase (shown on figure 3) necessary to provide a 15-pct DCFROR after taxation.
[8] Includes 9 mines or deposits in Finland (2), Italy (1), Morocco (3), Spain (2), and Sweden (1).

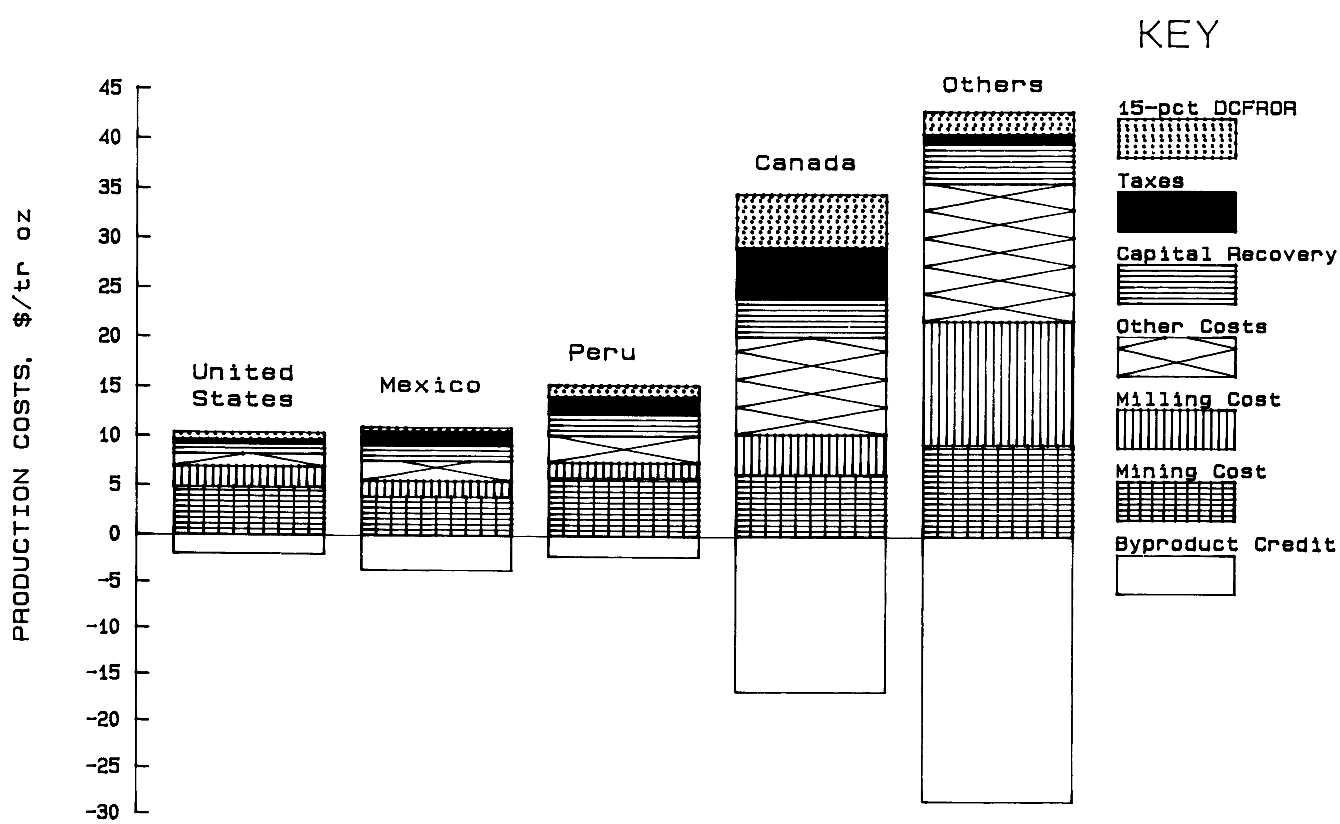

Figure 3.—Silver production costs for 53 producing predominantly silver mines in selected MEC's (January 1985 U.S. dollars).

Table 5.—Silver production costs for 33 predominantly silver producing mines in selected MEC's

(All costs are in January 1985 U.S. dollars per troy ounce of recoverable silver on a weighted-average basis)

Country	Operating cost		Other operating cost[1]	Byproduct credit	Net operating cost	Recovery of capital[2]	0-pct DCFROR		15-pct DCFROR		
	Mine	Mill					Taxes and royalties[3]	Total cost[4]	Taxes and royalties[5]	Return on investment[6]	Total cost[7]
United States ..	4.97	1.38	1.11	1.09	6.37	0.84	0.07	7.28	0.27	0.57	8.05
Mexico	3.02	1.36	1.11	2.30	3.19	1.63	.84	5.66	1.15	.32	6.29
Peru	5.05	1.28	2.16	1.00	7.49	1.91	.58	9.98	1.61	2.45	12.13
Canada	6.41	2.03	2.52	.24	10.72	3.35	.20	14.27	1.27	1.25	16.59
Morocco	.97	.62	.74	.04	2.29	.80	.31	3.40	.73	.45	4.27

[1] Includes cost of smelting, refining, and transportation of flotation concentrates or silver dore bullion.

[2] Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure as of January 1985, and investments required over the life of the operation.

[3] Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 0-pct DCFROR.

[4] Equal to the sum of net operating costs, taxation, and capital recovery determined at a 0-pct DCFROR.

[5] Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 15-pct DCFROR.

[6] The revenue increase per troy ounce necessary to obtain a 15-pct DCFROR.

[7] Equal to the sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery *plus* the per troy ounce increase (shown on figure 3) necessary to provide a 15-pct DCFROR after taxation.

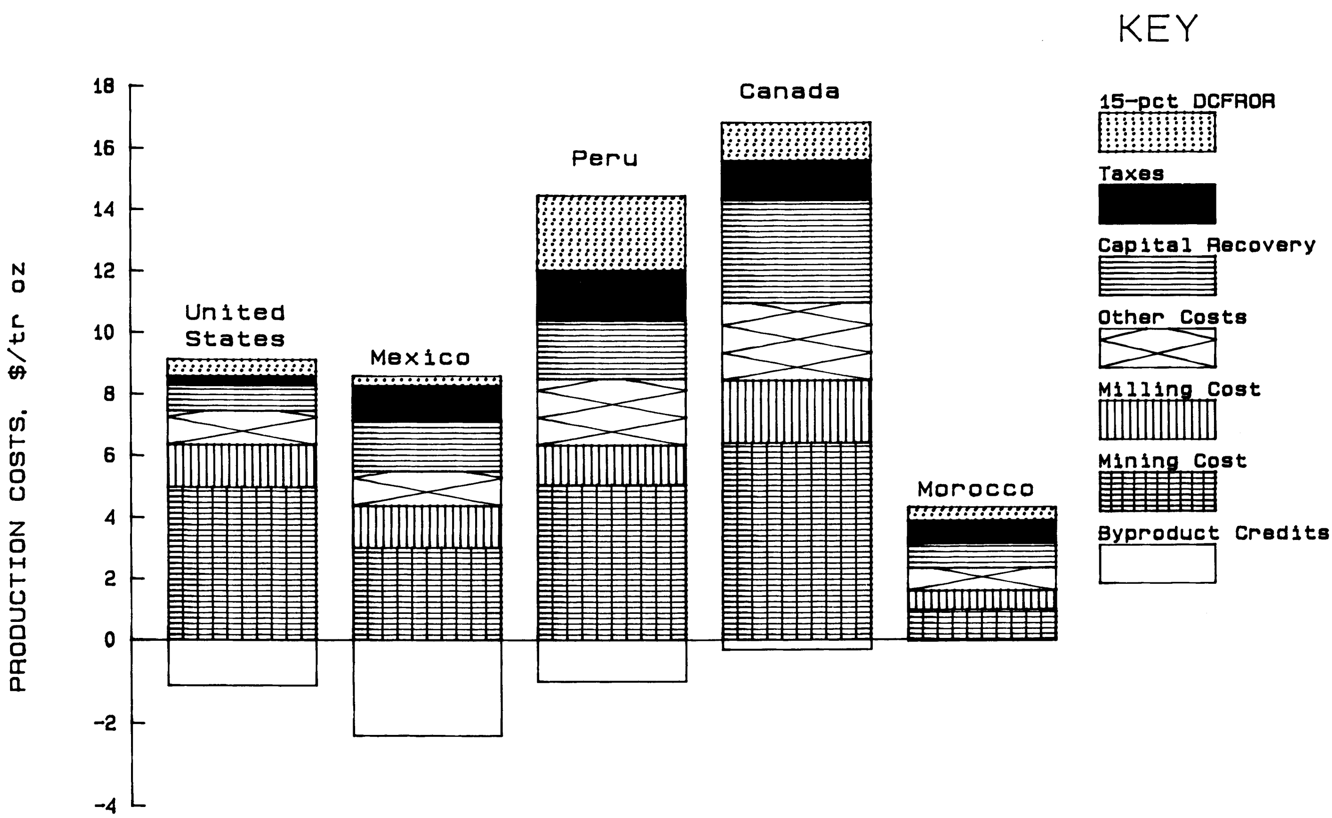

Figure 4.—Silver production costs for 33 producing predominantly silver mines in selected MEC's (January 1985 U.S. dollars).

AVAILABILITY

TOTAL RECOVERABLE

Table 6 summarizes the estimates of recoverable silver from the demonstrated resources of the 447 evaluated mines or deposits. The largest quantity of silver, 2.65 billion tr oz (40 pct), is available from the 122 mines and deposits classified as predominantly zinc deposits; 2.25 billion tr oz (34 pct), is available from the 131 mines and deposits classified as predominantly copper deposits; 0.24 billion tr oz (4 pct) is available from the 21 mines and deposits classified as predominantly lead deposits; and 0.11 billion tr oz (2 pct) is available from 107 predominantly gold mines and deposits. The 66 mines and deposits classified as predominantly silver deposits, which were defined as those with combined payable lead plus zinc grades of less than 3.5 pct, accounted for a total of only 1.32 billion tr oz (20 pct) of the total MEC recoverable silver resource analyzed.

Table 6.—Recoverable silver in demonstrated resources, thousand troy ounces

	January 1984		January 1985	
Commodity	Number of deposits	Recoverable silver	Number of deposits	Recoverable silver
Zinc	121	2,625,493	122	2,652,099
Copper ...	120	1,586,526	131	2,254,305
Silver	66	1,408,878	66	1,323,991
Lead	21	244,951	21	237,239
Gold	108	120,170	107	113,852
Total ...	436	5,986,018	447	6,581,486

Table 6 is also a comparison of the January 1985 values of recoverable silver and number of deposits shown in table 1 with the comparable values as of January 1984 that were reported and analyzed in reference 1. As shown, the number of deposits has increased from 436 to 447 (with the number of MEC's included increasing from 41 to 45), while the estimate of total recoverable silver has increased from 5.99 billion tr oz to 6.58 billion tr oz. Nearly all of this difference is due to a comprehensive reevaluation over the last year of the silver grades of domestic and foreign copper deposits, most of which were originally evaluated during the 1978-82 period. The reevaluation resulted in the addition of 23 new mines and deposits to the original list of those predominantly copper deposits containing recoverable silver and the deletion of 12 mines and deposits from the original list because of either exhaustion of reserves, too small a quantity of recoverable copper, or the combining of properties. The 23 new predominantly copper mines and deposits that were added contain an estimated 0.632 billion tr oz of recoverable silver as of January 1985.

Predominantly Silver

Figure 5 shows cumulative silver available from 53 currently producing operations at increasing total costs of production determined at the 0- and 15-pct DCFROR levels. A total of 1.1 billion tr oz of silver is potentially recoverable from all 53 operations. The 14 U.S. operations account for 364 million tr oz (33 pct) of the total.

At a 0-pct DCFROR, approximately 329 million tr oz of silver (29 pct of the total) is recoverable from 16 operations at a longrun break-even total cost level of \$6/tr oz or less; Mexico accounts for 6 of the operations and 45 pct of the total silver available in this cost range, and the U.S. accounts for 2 operations and 31 pct of the total. At a cost level of \$10/tr oz or less, approximately 900 million tr oz of silver (81 pct of the total) is potentially recoverable. Thirty-six operations have total cost estimates below \$10/tr oz; Mexico accounts for 10 of the operations and 43 pct of the total silver in this cost range, the United States accounts for 11 operations and 37 pct, and Peru accounts for 7 operations and 13 pct of the total silver. Only two Canadian operations, representing only 3 pct of the silver, have longrun total cost estimates of \$10/tr oz or less.

At a 15-pct DCFROR, the amount of silver available at total cost levels of \$6/tr oz and \$10/tr oz decreases, respectively, to 233 million tr oz (21 pct) and 780 million tr oz (70 pct) of the total. The United States and Mexico continue to account for a majority of silver available at or below these total cost levels. Mexico accounts for approximately 50 pct of all silver available at both the \$6/tr oz and \$10/tr oz levels.

If the 15-pct DCFROR criterion is assumed to be the necessary minimum level of return on investment for long-term silver production from predominantly silver mines, then 80 pct of the total silver estimate, representing 45 operations, is not economically extractable at a constant dollar price of \$6/tr oz. Even at the break-even level, 37 operations would be unable to cover their longrun costs at a constant dollar price of \$6/tr oz. The obvious implication is that, as of early 1985, a majority of current (predominantly silver) producers were suffering a long-term profitability squeeze. One result occurring from this profitablity squeeze is the increasing trend toward silver production from polymetallic deposits and the development of new polymetallic deposits where the economics of the operation do not depend so heavily on just the market price of silver.

Byproduct Silver

The long-term availability of byproduct silver is a function of the price and cost structure of the primary commodity with which it is associated. Copper and zinc mines are by far the most important sources of byproduct silver and account for roughly 75 pct of total silver availability as well.

In the case of byproduct silver from copper mines and deposits, approximately 1.4 billion tr oz (60 pct) of the total

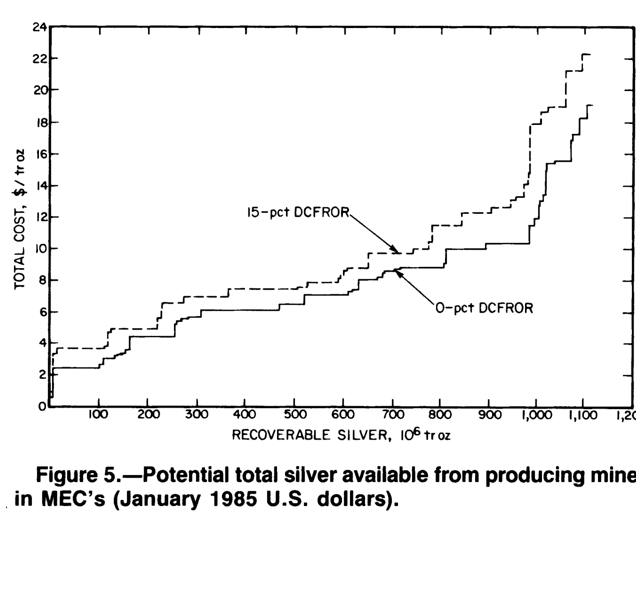

Figure 5.—Potential total silver available from producing mines in MEC's (January 1985 U.S. dollars).

is available from currently producing mines and those mines classified as temporarily shut down. Of this total, approximately 0.4 billion tr oz (30 pct) is available from mines with long-term total production costs (including a 15-pct DCFROR) below $0.63/lb copper (the January 1985 market price) and 1.1 billion tr oz (82 pct) is available at $1/lb copper.

In the case of byproduct silver from zinc, approximately 1.6 billion tr oz (60 pct) of the total is available from mines classified as producers for this study. Of this total, approximately 0.72 billion tr oz (45 pct) is available from mines with long-term total production costs (including a 15-pct DCFROR) below $0.40/lb zinc (the January 1985 market price).

In total, there is approximately 3.0 billion tr oz of byproduct silver available from producing copper and zinc mines. These mines represent roughly 45 pct of all available silver and more than 2.5 times as much silver as is available from mines classified as predominantly silver producers.

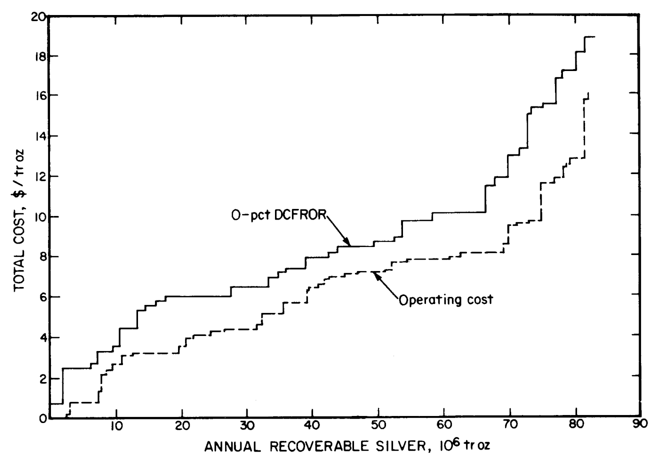

Figure 6.—1985 silver capacity from 53 predominantly silver producing mines in MEC's (January 1985 U.S. dollars).

ANNUAL CAPACITY

In 1985, it was estimated that 84.3 million tr oz of silver was potentially available from the 53 predominantly silver producing operations. This total represented roughly 21 pct of total world production (including byproducts) or 26 pct of total MEC production. The majority of annual MEC silver production remains a byproduct of copper and zinc mines and is dependent upon the price and production of copper and zinc.

Figure 6 presents 1985 capacity production levels from the producing predominantly silver operations. The graphic displays two curves that relate operating cost and total cost at the break-even level (0-pct DCFROR) to potential capacity production.

For 1985, it is estimated that 21 of the 53 operations could cover their longrun operating costs at a cost level of $6/tr oz; at $10/tr oz, 44 of the 53 operations could cover these costs. Operating costs for some of these predominantly silver producers, as already demonstrated, are significantly influenced by byproduct revenues for copper, lead, zinc or gold.

CONCLUSIONS

A total of 447 mines and deposits in 45 MEC's are estimated to contain 6.58 billion tr oz of recoverable silver as of January 1985. Approximately 40 pct of this total is available as a byproduct of predominantly zinc mines and deposits, 34 pct as a byproduct of predominantly copper mines and deposits, and predominantly lead and gold mines and deposits combined account for only 6 pct of the total demonstrated silver resource.

A two-tier criterion was applied to define a predominantly silver deposit. By the first tier (less than 3.5 pct combined payable lead plus zinc) 66 properties (53 producing mines and 13 nonproducing deposits) were included. These mines and deposits are estimated to contain 1.32 billion tr oz of recoverable silver (20 pct of total) at the demonstrated level. By the second tier (greater than 60 pct silver revenues at both $5/tr oz and $10/tr oz) 36 properties (33 producing mines and 3 nonproducing deposits) were included. The 33 producers included in this classification contain only 701.7 million tr oz (10.5 pct) of the total available silver. Predominantly silver mines account for a minority of total available silver and are judged to have much less of an influence on the economics of silver production and the silver market in the MEC's than byproduct silver production and total availability from predominantly zinc or copper mines.

Production cost analysis (of 53 producing predominantly silver operations) indicates that at a constant January 1985 U.S. dollar silver price of $6/tr oz, 37 producing mines are unable to cover their long-term total production costs at the 0-pct DCFROR. At a 15-pct DCFROR, 45 producing operations are unable to cover their long-term total production costs at $6/tr oz silver. The implication of this analysis is that many of these predominantly silver producers are suffering a long-term profitability squeeze and will require higher silver prices to remain in production over the long term. As a result of relatively low silver prices, the trend toward the development of polymetallic mines, that are not primarily (or totally) dependent upon the price of a single commodity, is expected to continue.

REFERENCES

1. Boyle, E.H., Jr. G.R. Peterson, and P.R. Thomas. Primary Silver Availability—Market Economy Countries. A Minerals Availability Appraisal. BuMines IC 9090, 1986, 36 pp.

2. Heyl, A.V., W.E. Hall, A.E. Weissenborn, H.K. Stager, W.P. Puffett, and B.L. Reed. Silver. Ch. in United States Mineral Resources: U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 581-603.

3. Reese, R.G. Silver. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 142-143.

SULFUR (ELEMENTAL) AND PYRITE CONCENTRATE

INTRODUCTION

Sulfur differs from most major mineral commodities in that it is used as a processing and manufacturing reagent and is therefore considered an important industrial mineral. Elemental sulfur and other sulfur compounds (pyrites) usually are converted to intermediate chemicals, such as sulfuric acid (H_2SO_4) and sulfur dioxide (SO_2) prior to use. Sulfuric acid is the most important of these intermediate chemicals. More than 86 pct of the sulfur consumed annually in the world is either converted to H_2SO_4 or produced directly in that form. Consumption of sulfur as H_2SO_4 has been regarded as an index of a nation's industrial development. Little, if any, of the sulfur content is retained in the final product. Many agricultural and industrial products such as phosphatic fertilizers and titanium dioxide (TiO_2) pigments require these intermediate sulfur chemicals for their manufacturing and processing (*1*, pp. 783, 787).[1]

Sulfur producers have been actively researching and promoting new and/or relatively undeveloped markets for sulfur. These new markets include the direct application of elemental sulfur to the ground as a soil conditioner, plant nutrient, pesticide, and fungicide. Other new markets are sulfur-extended asphalts, sulfur concretes, and other building and construction materials. Research into these uses has been pursued in regions with large domestic sulfur resources, notably Canada, France, the Middle East, and the United States (*2*, pp. 31-34).

Two products, elemental sulfur and pyrite concentrate, are analyzed in this chapter. Elemental sulfur (97-99.8 pct S) is found in native sulfur deposits. Pyrite concentrates (20-50 pct S) are recovered from sulfide ore (pyrite) deposits for the contained sulfur. Some coproduct pyrite is included in this chapter. For purposes of consistency in this analysis, it is assumed that all elemental sulfur and pyrite concentrates are transported to the nearest port, acid plant, or smelter-refinery. Additional costs for the conversion of elemental sulfur and pyrite concentrate to H_2SO_4 or other sulfur compounds are not included in this evaluation.

Secondary sulfur sources are not included in the analysis. They include recovered elemental sulfur (a byproduct from the refining of petroleum and sour gas resources), all waste product pyrite concentrates, and smelter-refinery SO_2 offgases converted to H_2SO_4. Recovery of sulfur from these secondary sources is nondiscretionary and cannot be adjusted to sulfur demand, since the "productivity" of secondary sources is based on market requirements for low-sulfur and sulfur-free products, or removal of sulfur and its compounds for environmental reasons.

Most of the information presented in this chapter is an updated summary of Bureau of Mines Information Circular 9106 "Availability of Elemental Sulfur and Pyrite Concentrate—Market Economy Countries. A Minerals Availability Program Appraisal"(*3*).

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

GEOLOGY AND RESOURCES

GEOLOGY

Sulfur is widely distributed in nature. It is the 13th most abundant element and constitutes 0.6 pct of the Earth's crust. Sulfur is found in a wide variety of geological settings in elemental form, as sulfate and sulfide minerals, and as organic compounds in fossil fuels. Sulfur deposits can be divided into six types: elemental or native sulfur, petroleum and tar sands, sour gas, coal and oil shale, sulfate (gypsum), and metal sulfide (pyrites). Elemental or native sulfur deposits, metal sulfide (pyrite) deposits, and sour gas deposits, are the most important and supply most of the world's sulfur (*1*, p. 784; *4*, pp. 607, 613-617).

Native Sulfur Deposits

Native sulfur deposits are associated with anhydrite caprock overlying salt domes, bedded anhydrite and gypsum evaporite formations, and unconsolidated and weathered volcanic rock formations. In salt domes and evaporite beds, sulfur is considered to have been formed by hydrocarbon reduction of the anhydrite or gypsum, assisted by bacterial attack. Examples of salt dome deposits occur along the gulf coasts of Louisiana and Texas in the United States, and of Vera Cruz, Mexico (*5*). Bedded evaporite deposits of minable native sulfur occur in west Texas and in the Mosul region of northern Iraq (*6-7*). The majority of volcanic sulfur deposits occur in the Circumpacific Volcanic Belt; however, some volcanic sulfur deposits have been found in Turkey (*8*).

Pyrite Deposits

Ferrous and nonferrous metal sulfide deposits are also sources of sulfur. Ferrous metal sulfide deposits are the most important; these generally occur as massive high-grade pyritic ore bodies ranging from 17 to 50 pct S and are primarily developed for their sulfur content. The recovered pyrite concentrates are converted to H_2SO_4 or, less commonly, elemental sulfur. These deposits may also contain copper, lead, zinc, gold, and silver, which can be recovered as coproducts. Coproduct credits (payments) determine the economic feasibility of many pyrite operations. Such deposits occur widely throughout the world. Some of the richest pyrite deposits are found in the Iberian Pyrite Belt of southern Portugal and southwestern Spain (*9*, pp. 63-65; *10*); other massive pyrite ore deposits occur in Cyprus (*11*), and Japan. Sulfide deposits where pyrite is an important coproduct are found in Finland (*12*), Norway (*13*, p. 745; *14*, p. 311; *15*, p. 14), Sweden (*16-17*), and Turkey (*8*).

RESOURCES

World sulfur resources are immense. Estimates vary widely owing to the different deposit types included. As shown on figure 1, world sulfur resources are estimated to be 5 billion mt. The reserve base, which includes in situ demonstrated resources that were economic or marginally economic and some that are subeconomic, is estimated to be 2.7 billion mt (*18*). The lack of available sulfur-related data for many of these deposits limits the sulfur resources analyzed in this report to 25 pct of the reserve base estimate.

The demonstrated resources analyzed in this report total 684 million mt (267 million mt of elemental sulfur and 417 million mt as pyrites). Major market economy countries (MEC's) included are Mexico and the United States for elemental sulfur production, and Italy, Portugal, and Spain for production of pyrite concentrates. A breakdown of these resources, by country, is provided in table 1; table 2 lists the 34 MEC elemental sulfur and pyrite concentrate operations that were evaluated. Centrally planned economy countries (CPEC's) were not evaluated because of a lack of cost information.

Table 1.—Summary of MEC demonstrated elemental sulfur and pyrite concentrate resources evaluated for this study, as of January 1985, million metric tons

Country[1]	In situ	Recoverable[2]
	ELEMENTAL SULFUR	
Iraq	135.1	W
Mexico	62.5	46.4
Turkey	1.4	W
United States	68.0	67.9
Total	267.0	195.5
	PYRITE	
Cyprus	W	W
Italy	36.5	29.2
Japan	W	W
Finland, Norway, and Sweden	59.1	23.1
Portugal	163.3	114.1
Spain	113.1	96.9
Turkey	W	W
Total	417.0	289.1

W Withheld to avoid disclosing company proprietary data; included in total.
[1] Data from other countries not available.
[2] Estimated tonnage includes losses from mine and mill or processing recovery.

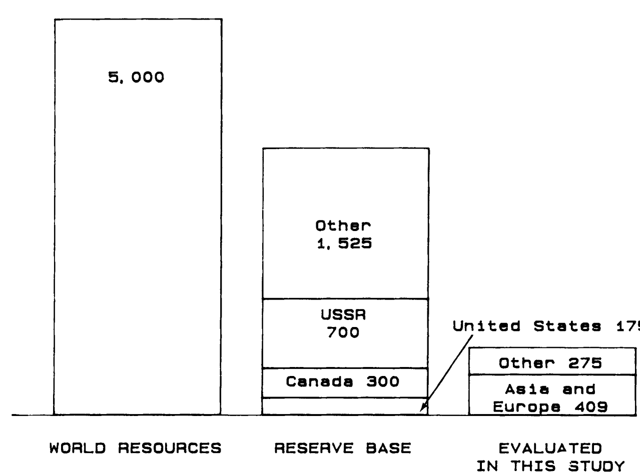

Figure 1.—Estimates of world sulfur resources (million metric tons).

Table 2.—MEC elemental sulfur and pyrite concentrate properties included in this study

Property	Current status[1]	Mining method	Benefication method	Sulfur product[2]
Cyprus:				
Kambia (Kampia)	P	Open pit	Flotation	Pyr
Mathiatis	P	..do	..do	Pyr
Sha (Shia)	P	..do	..do	Pyr
Finland: Pyhasalmi	P	Sublevel caving	..do	Pyr
Iraq: Mishraq	P	Frasch	Filtration	S
Italy:				
Campiano	P	Room and pillar	Sizing	Pyr
Fen ce Capanne	P	..do	Flotation	Pyr
Niccioleta	P	Sublevel stoping	Sizing	Pyr
Japan: Yanahara	P	..do	..do	Pyr
Mexico:				
Coachapa	P	Frasch	None performed	S
Jaltipan	P	..do	Filtration	S
Texistepec	P	..do	..do	S
Norway:				
Grong Gruber	P	Open stoping	Flotation	Pyr
Sulitjelma	P	Room and pillar	..do	Pyr
Tverrfjellet	P	Sublevel stoping	..do	Pyr
Portugal				
Aljustrel	P	Horizontal cut and fill	..do	Pyr
Lousal	P	Open stoping	..do	Pyr
Spain				
Herrerias	P	Room and pillar	..do	Pyr
La Zarza-Calanas	P	Fill stoping	Sizing	Pyr
Tharsis	P	Open pit	..do	Pyr
Sweden:				
Kristineberg	P	Horizontal cut and fill	Flotation	Pyr
Langsele-Udden	P	..do	..do	Pyr
Turkey:				
Asikoy	D	Open pit-underground	..do	Pyr
Cayeli	NP	Inclined cut and fill	Flotation	Pyr
Keciborlu	P	Open pit-horizontal cut and fill	Flotation and direct melting	S
United States:				
Louisiana:				
Caillou Island[3]	PP	Frasch	None performed	S
Caminada	PP	..do	..do	S
Garden Island Bay	P	..do	..do	S
Grand Isle	P	..do	..do	S
Texas:				
Boling Dome	P	..do	..do	
Comanche Creek	PP	..do	..do	S
Culberson	P	..do	..do	S
Long Point	PP	..do	..do	S
Phillips Ranch	NP	..do	Filtration	S

[1] P, producer; PP, past producer; NP, Nonproducer; D, developing.
[2] Pyr, pyrite concentrate; S, elemental sulfur.
[3] Closed May 1984.

U.S. AND WORLD HISTORICAL PRODUCTION

Elemental sulfur from native sulfur deposits and the desulfurization of sour gas, petroleum, and tar sands accounts for 65 pct of the world's total sulfur production. The remaining sulfur is produced as H_2SO_4 from pyrites, metallurgical operations, and coal gasification. Currently, no single country is the dominant producer or supplier of sulfur and/or sulfur compounds to world markets. Approximately 50 pct of the world's total sulfur production comes from countries in which the industry is either partially or entirely government owned and/or operated. In terms of total sulfur output, the CPEC's produced an estimated 37 pct of the world's sulfur in 1985; the U.S.S.R. accounted for nearly 57 pct of this production. Leading MEC producers, in terms of total sulfur output, in 1985 were the United States, 11.4 million mt; Canada, 6.8 million mt; Japan, 2.7 million mt; and Mexico, 2.0 million mt (*18*, p. 155).

Native elemental sulfur production in 5-yr intervals from 1955 to 1985 is shown in figure 2. Production has increased rapidly since 1955. Most of the increase occurred in Mexico, Poland, the United States, and the U.S.S.R. Highest levels of annual production have averaged about 17.4 million mt between 1974 and 1976 and again at 17.2 million mt in 1980. Production has since declined, particularly in the United States, where 1985 production of 5.0 million mt was 882,000 mt below 1955's production. As a comparison to native elemental sulfur production, the production of recovered elemental sulfur from petroleum and sour gas (also in 5-yr intervals from 1955 to 1985) is shown

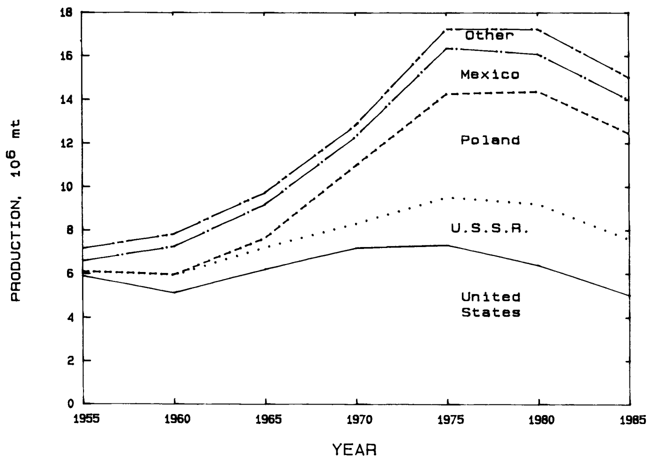

Figure 2.—World native elemental sulfur production, 1955-85.

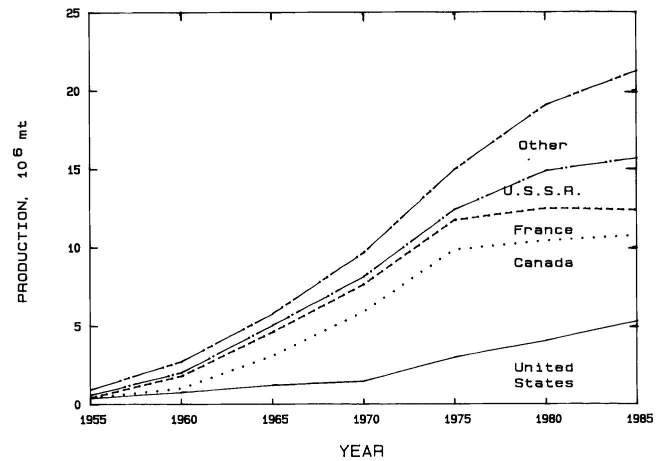

Figure 3.—World recovered elemental sulfur production, 1955-85

in figure 3. Since 1955, production has steadily increased, although recently it seems to be leveling off, except in the United States, where it is still increasing.

Pyrite concentrate production has remained relatively stable, averaging about 9.7 million mt annually, with some minor fluctuations, over the past 30 yr. Production in 5-yr intervals from 1955 to 1985, is shown in figure 4. The European and Scandinavian countries and the U.S.S.R. account for most of the pyrite concentrate production, with slightly over one-half of the total. Until the mid-1970's, Asia, mainly Japan, was a major contributor to world pyrite concentrate production. Currently the U.S.S.R. and countries included in the "other" category produce nearly three-quarters of the pyrite concentrate. Lowest levels of annual pyrite production were reached in the 1960's at about 9.0 million mt, increasing to 10.3 million mt by 1970. Highest levels were reached in the 1970's and early 1980's. During those years, production fluctuated between 10.3 million mt and 9.3 million mt, averaging about 9.9 million mt annually.

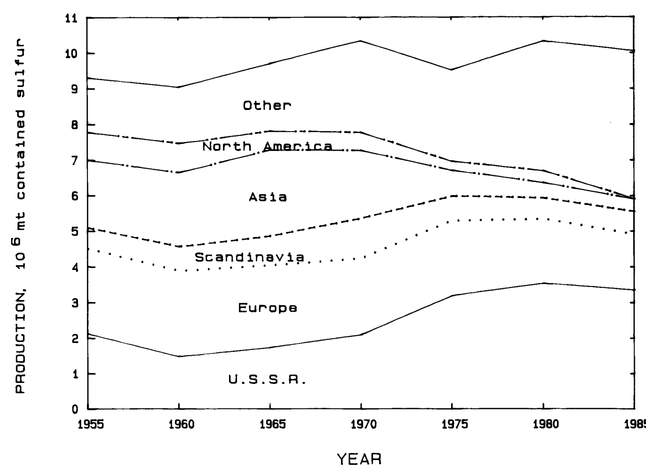

Figure 4.—World pyrite concentrate production, 1955-85.

EXTRACTION AND PROCESSING TECHNOLOGY

Frasch mining and processing technology was developed by Herman Frasch in 1894 (*19*, p. 40) and involves the injection of superheated (163° C) water through wells drilled into formations containing native sulfur mineralization. In some mining areas (Iraq and Poland) where the formation is not porous enough to promote sulfur and water migration, blasting near the bottom of the well is used to fracture the formation. Heat from the water is transferred to the formation, thus melting the sulfur. The sulfur, being heavier than water, accumulates in a pool at the bottom of each well. Compressed air is injected into each well to raise the molten sulfur to the surface where it is extracted by bleeder wells located along the flanks of the formation. Once on the surface, the liquid sulfur (97-99.8 pct S) may require filtration, generally through a mixture of H_2SO_4 and diatomaceous earth, to remove organic impurities; it is then pumped to surface storage facilities.

The Frasch process is used exclusively to extract sulfur from salt dome formations along the gulf coast of Louisiana and Texas in the United States, and of Vera Cruz, Mexico. This process is also used in bedded evaporite formations such as those in west Texas, the Mosul region of northern Iraq, and in formations in Poland, and the U.S.S.R.

Elemental sulfur deposits not amenable to the Frasch process use surface and/or underground mining methods. Beneficiation methods vary widely. High- to medium-grade ores require little or no beneficiation. Low-grade ores are treated by processes including direct melting, distillation, agglomeration, solvent extraction, and flotation, each recovering elemental sulfur. Sulfur ores of the Keciborlu operation, near Isparta in western Turkey, are mined by open pit and horizontal cut-and-fill methods; direct melting and flotation beneficiation processes are used to recover elemental sulfur.

Surface and underground mining methods are also used on pyrite deposits. High-grade pyrite ores require only minor beneficiation consisting of crushing, grinding, screening, and washing. Examples of these types of operations are found along the Iberian Pyrite Belt in Portugal and Spain, the Troodos Massif in Cyprus, and massive pyrite deposits in Japan.

Lower grade pyrite ores are upgraded by flotation methods. Examples of these types of operations are found in Italy, Finland, Norway, and Sweden.

The domestic elemental sulfur market price, exterminal, Tampa Bay, FL, increased four times in 1984 (*20*, p.10), starting at $130.40/mt and finishing the year at $155/mt, where it remained throughout 1985. The average international elemental sulfur market price for 1984 ranged from a low of $124.50/mt from February to early July, then increased to $132.50/mt for the remainder of the year. In 1985 the price increased to $142.75/mt; by the end of the year the price had increased to nearly $150/mt (*21*, p. 4).

The 1984 market price for pyrite concentrate is $42.99/mt (45-50 pct S). No changes to this market price were reported in 1985.

PRODUCTION COSTS

Table 3 and figures 5 and 6 illustrate the average operating and production costs for selected operations included in this evaluation (expressed in January 1985 U.S. dollars per metric ton of recovered product). These costs are in constant dollars and are based on current technology. As technology changes in the future, costs for these operations, particularly those with longer lives, will also change. Costs do not include the conversion of elemental sulfur or pyrite concentrate to H_2SO_4 or other sulfur compounds.

Net operating costs for eight U.S. Frasch operations averaged nearly $85/mt of elemental sulfur. Because the other U.S. Frasch operation is a nonproducer, its costs were not included in the table. Mining costs for these eight operations accounted for about 75 pct of the net operating cost.

Table 3.—Elemental sulfur and pyrite concentrate production costs for producing operations in selected MEC's

(All costs are in January 1985 dollars per metric ton of recovered product on a weighted-average basis)

Region and country	Operating cost		Transportation[1]	Coproduct credits	Net operating cost	Recovery of capital[2]	0 pct-DCFROR		15 pct-DCFROR		
	Mine	Mill					Taxes and royalties[3]	Total cost[4]	Taxes and royalties[5]	Return on investment	Total cost[6]
					ELEMENTAL SULFUR						
Mexico: Average	30.23	5.20	2.97	0	38.40	4.20	1.61	44.21	13.65	10.16	66.41
United States:											
Producers	56.75	0	24.26	0	81.01	1.73	0.27	83.01	0.91	1.44	85.09
Temporarily closed.	89.23	0	11.45	0	100.69	5.50	1.26	107.45	7.15	8.01	121.34
Average	63.36	0	21.65	0	85.01	2.50	0.47	89.99	2.18	2.77	92.47
Other:											
Iraq	W	W	W	W	W	W	W	W	W	W	W
Turkey	W	W	W	W	W	W	W	W	W	W	W
Average	13.53	2.64	8.24	0	24.41	0.75	0.19	25.35	2.06	1.36	28.58
					PYRITE CONCENTRATE						
Europe:											
Italy	W	W	W	W	W	W	W	W	W	W	W
Portugal	W	W	W	W	W	W	W	W	W	W	W
Spain	5.77	1.10	1.70	0	8.56	1.69	0.26	10.51	1.20	1.81	13.26
Average	6.81	1.57	1.67	2.23	7.82	2.45	1.07	11.34	2.98	2.76	16.01
Scandinavia:											
Finland	W	W	W	W	W	W	W	W	W	W	W
Norway	20.43	13.44	6.72	63.73	−23.14	25.73	4.12	6.71	16.88	13.22	32.69
Sweden	W	W	W	W	W	W	W	W	W	W	W
Average	27.89	15.80	7.21	73.02	−22.12	34.26	3.58	15.72	12.49	8.87	33.50
Other:											
Cyprus	26.11	12.51	1.46	0	40.08	4.94	.25	45.27	2.07	3.59	50.69
Japan	W	W	W	W	W	W	W	W	W	W	W
Turkey	W	W	W	W	W	W	W	W	W	W	W
Average	13.06	22.92	4.06	16.29	23.75	12.60	2.11	38.46	6.78	6.76	49.89

W Withheld to avoid disclosing company proprietary data.

[1] Transportation costs to ports, acid plants, or smelters that have been assumed as product destination points for this study.

[2] Includes cost of recovering remaining undepreciated investments and reinvestments over the life of the operation.

[3] Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 0-pct DCFROR.

[4] Equal to the sum of net operating costs, taxation, and recovery of capital calculated at a 0-pct DCFROR.

[5] Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 15-pct DCFROR.

[6] Equal to the sum of net operating cost, recovery of capital, taxation, and return on investments calculated at a 15-pct DCFROR.

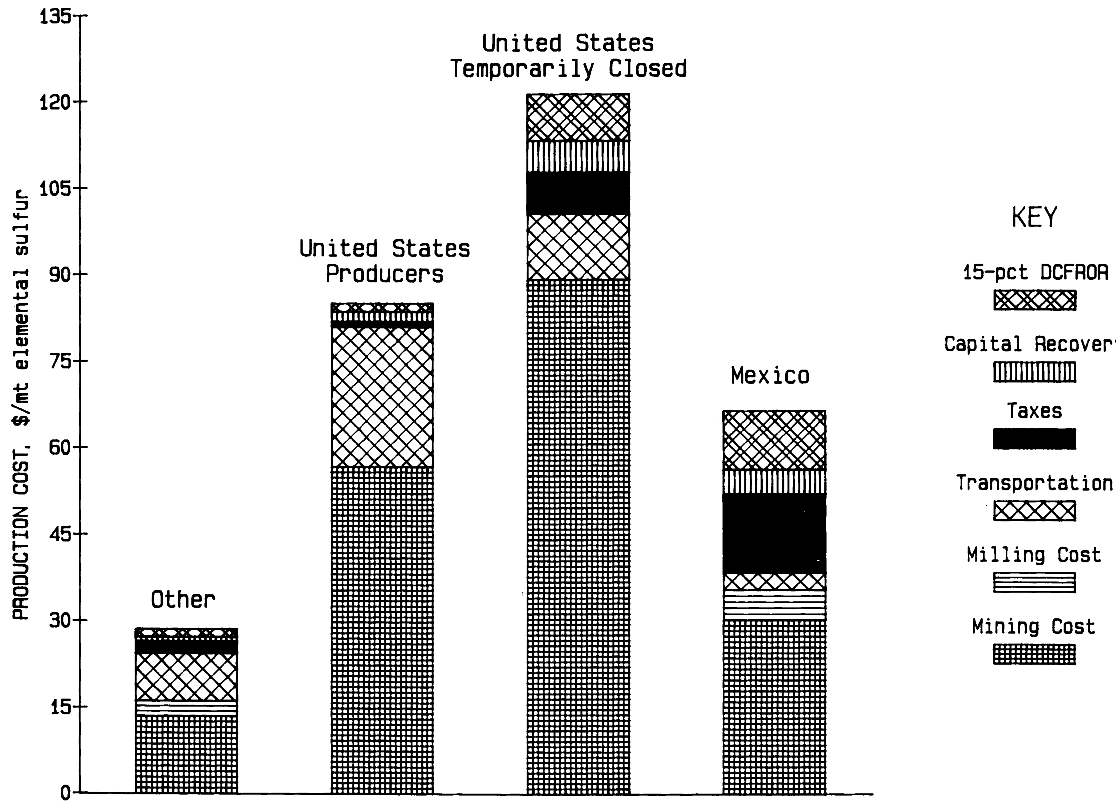

Figure 5.—Elemental sulfur production costs for selected MEC's (January 1985 U.S. dollars).

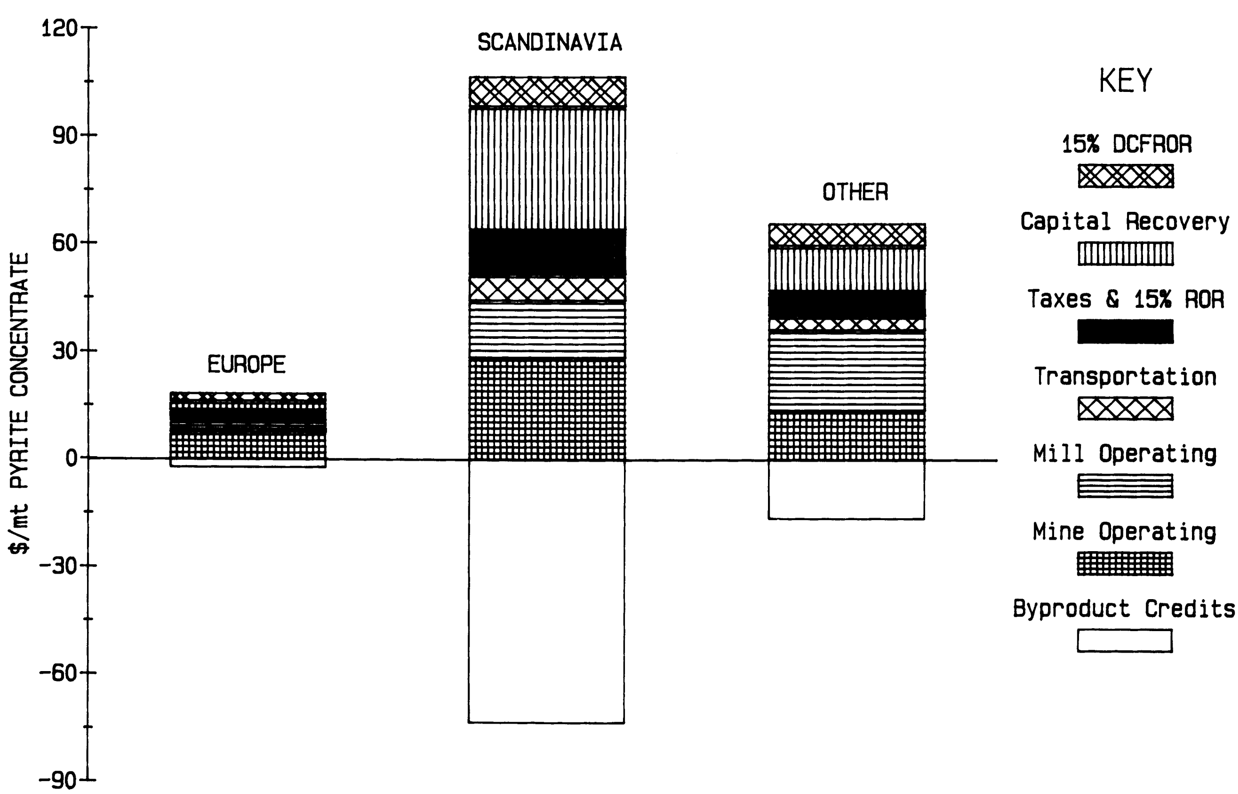

Figure 6.—Pyrite concentrate production costs for selected regions (January 1985 U.S. dollars).

Transportation costs by ocean barge and rail account for the remaining 25 pct. Four producing U.S. operations have a net operating cost average of about $81/mt, with mining costs accounting for 71 pct. The four U.S. temporarily closed (past producing) operations have a net operating cost of nearly $101/mt of elemental sulfur, with mining costs accounting for 87 pct of the net operating cost. No beneficiation is required at these eight U.S. Frasch operations.

Mexico's Frasch operation's net operating costs average $38/mt sulfur. Mining costs accounted for about 79 pct of the net operating cost. Filtration costs at these operations average $5.20/mt of elemental sulfur, or about 13 pct of the net operating cost. River barge transportation costs accounts for the remaining 8 pct of the net operating cost.

Frasch sulfur operations can lower operating costs by recycling well bleed water and generating electricity from waste boiler heat. Foreign operations, particularly those in Mexico and Iraq, have a cost advantage owing to lower energy, labor, and material costs.

Net operating costs for pyrite concentrate operations in Italy, Spain, and Portugal range from $5/mt to $8.56/mt, and are the lowest of all the pyrite concentrate properties evaluated. Most of these operations use underground mining methods, averaging about $7/mt. Beneficiation (sizing) and transportation (truck and rail) costs average $1.57/mt and $1.67/mt, respectively. The average coproduct credit is low, at $2/mt. However, for the three Italian properties this credit averages $18/mt.

Coproduct revenues at pyrite concentrate operations in Finland, Norway, and Sweden are sufficiently high to cover total operating costs. These operations use underground mining methods and flotation for beneficiation at costs averaging nearly $28/mt and $16/mt, respectively. Transportation costs, mostly truck and rail, average about $7/mt.

Highest net operating costs are found at the three Cypriot operations, averaging about $40/mt, resulting from high mining and beneficiation costs and coproduct credits.

AVAILABILITY

Fourteen elemental sulfur and twenty pyrite concentrate properties were included in this analysis (see table 1). Elemental sulfur and pyrite resources located in the U.S.S.R., China, and other CPEC's were not included in this study, owing to lack of resource and production cost data. Costs for the conversion of elemental sulfur or pyrite concentrate to H_2SO_4 or other sulfur compounds are not included in this analysis.

ELEMENTAL SULFUR

Total Recoverable

A total of nearly 196 million mt of elemental sulfur is potentially recoverable from the demonstrated resources of the 14 native sulfur operations evaluated (fig. 7). Production costs for the nine producing operations are bracketed by lower and upper cost levels. The lower level reflects costs at a 0-pct discounted-cash-flow rate of return (DCFROR) while the upper level reflects total cost including a 15-pct DCFROR. The cost curve for five nonproducing operations, four past producers and one nonproducer, includes a 15-pct DCFROR.

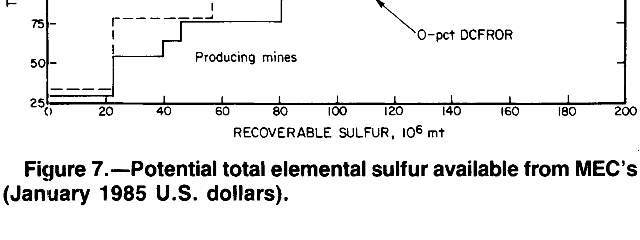

Figure 7.—Potential total elemental sulfur available from MEC's (January 1985 U.S. dollars).

At a 0-pct DCFROR, nearly all of the potentially recoverable elemental sulfur is available below a cost of production of $103/mt. Approximately 193 million mt is available below a cost of production equal to the estimated January 1985 international market price for elemental sulfur ($143/mt). Operations that were producing as of January 1985 contribute a potential of 180 million mt. Most of this tonnage is available below a cost of production of $143/mt.

Iraq and Turkey account for about 81 million mt, the majority of recoverable elemental sulfur. This tonnage is available below an average production cost (refer to table 3) of $26/mt (0-pct DCFROR). Mexico accounts for about 46 million mt of elemental sulfur; all of this production is available at an average production cost $44/mt. The United States accounts for about 68 million mt of elemental sulfur, 77 pct of which is available from four producing operations at an average total production cost of $83/mt. Twenty-three percent (16 million mt) is potentially available from four temporarily closed and one nonproducing operation. Most of this tonnage is available below an average total production cost of $108/mt.

Annual Capacity

Estimated potential production for 1985 (fig. 8) was 5.9 million mt elemental sulfur from nine producing operations. The curve is bracketed by a lower operating cost level and an upper cost level at a 0-pct DCFROR. Producers in Mexico and the United States account for about 92 pct of the total annual tonnage; all of their production is available at a cost of production of less than $108/mt. The remainder, about 450,000 mt, comes from producers in Iraq and Turkey.

Total potential annual availability of elemental sulfur from 1985 to 1995 is shown in figure 9. It is assumed that over the long run, all costs of production, including a 15-pct

DCFROR, will be recovered. Therefore, potential annual capacities are shown for selected cost ranges that include a 15-pct DCFROR. The curves reflect the production capacity of existing mines, including planned expansions, if known. It is assumed that all operations will produce at full (100 pct) capacity over the life of the mine. Because actual production may be at less than capacity, the curves may not decline as rapidly as shown.

The curves show that potential production from the demonstrated resources of producing operations begins to decline in 1988, with a further decline occurring in 1994. This decrease in potential elemental sulfur production is caused by the depletion of January 1985 estimated resources, particularly in the Mexican and U.S. Frasch sulfur operations. Some of the estimated decline could be counteracted by the expansion of production capacities from remaining producers, such as Iraq, which have large demonstrated resources, although such an expansion would effectively shorten their production lives. In addition, the four past producers and one nonproducer located in the United States and new Frasch operations in Mexico could be brought into production. The four U.S. producers have a potential annual production of 3.6 million mt sulfur over the next 10 yr. Mexico's three producers have a potential annual production of 1.4 million mt, which is expected to remain constant over the next 10 yr. Estimated annual production at the operations in Iraq and Turkey will be less than 400,000 mt/yr, owing to engineering and production problems. At these low production levels, their resources could last well into the next century. The five nonproducing operations in the United States could add an additional annual tonnage of nearly 1.0 million mt.

PYRITE CONCENTRATE

Total Recoverable

A total tonnage of 289 million mt of pyrite concentrate containing nearly 133 million mt of sulfur, is potentially recoverable from the demonstrated resources of the 20 operations evaluated (fig. 10). Production costs for producing operations are bracketed by lower and upper cost levels. The lower level reflects costs at a 0-pct DCFROR while the upper level reflects total costs including a 15-pct DCFROR. Two nonproducing properties were evaluated; one is currently under development and was scheduled to open in mid-1985 and the other is a nonproducer. Both are included with the producers to avoid disclosing proprietary data.

At a 0-pct DCFROR, all of the potentially recoverable pyrite concentrate is available at a cost of production below $97/mt. At a cost of production below $43/mt, which relates to the estimated January 1985 market price for a 45-pct-S pyrite concentrate, approximately 278 million mt of pyrite concentrate (128 million mt sulfur), is potentially recoverable.

The European countries account for nearly 86 pct (240 million mt) of the total recoverable pyrite concentrate. This tonnage is available below a production cost (table 3) of $12/mt. The Scandinavian countries account for 23 million mt (8 pct of the total); this tonnage is available below a production cost of $16/mt. Cyprus, Japan, and Turkey account for about 26 million mt of pyrite concentrate, 9 pct of the total; this tonnage is available below a production cost of $39/mt.

Annual Capacity

Estimated potential production for 1985 from the properties evaluated (fig. 11) was nearly 5.6 million mt of pyrite concentrate. The curve is bracketed by a lower operating

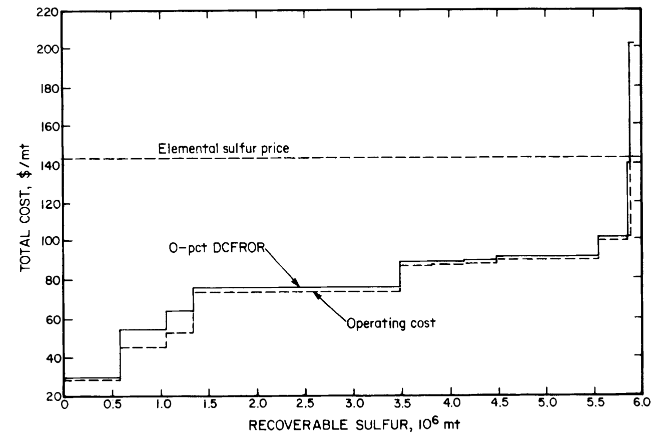

Figure 8.—1985 elemental sulfur capacity from producing mines in MEC's (January 1985 U.S. dollars).

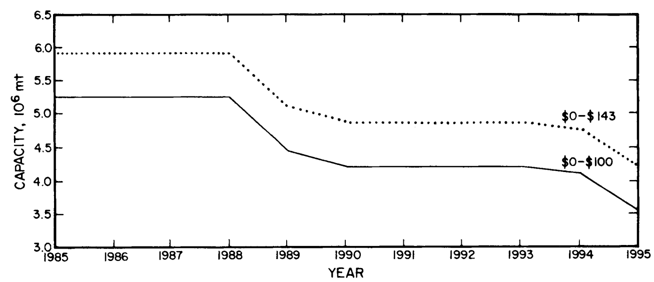

Figure 9.—Potential annual elemental sulfur production from producing mines in MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

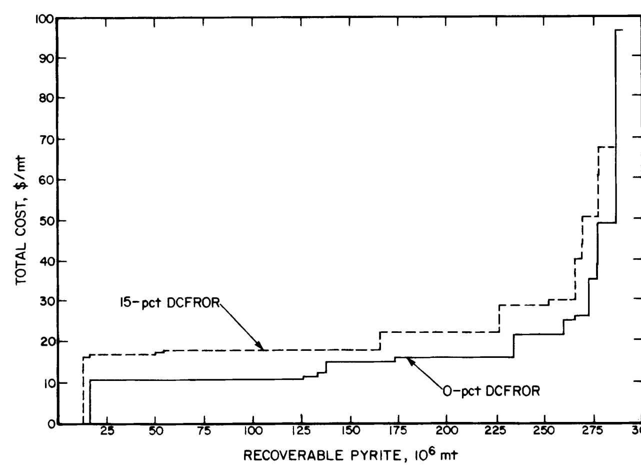

Figure 10.—Potential total pyrite concentrate available from MEC's (January 1985 U.S. dollars).

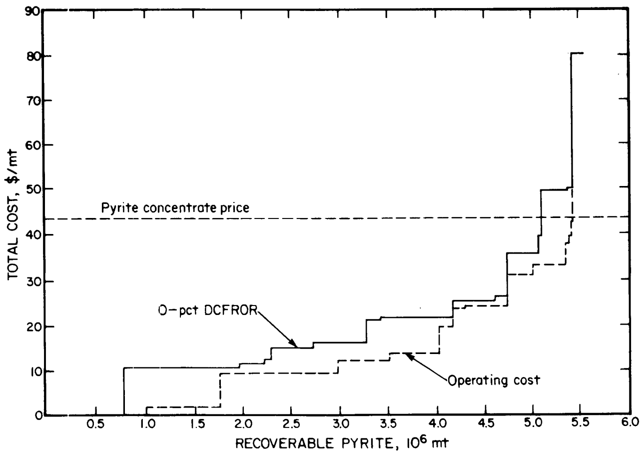

Figure 11.—1985 pyrite concentrate capacity from producing mines in MEC's (January 1985 U.S. dollars).

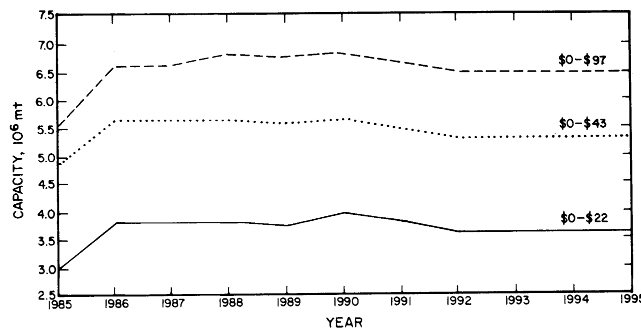

Figure 12.—Potential annual pyrite concentrate production from producing mines in MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

cost level and an upper cost level at a 0-pct DCFROR. Properties in Italy, Portugal, and Spain account for 3.6 million mt of the total production. This tonnage is available at a cost of production below $43/mt. Finland, Norway, and Sweden account for 1.5 million mt of the total annual 1985 production; 75 pct of this tonnage is also available below $43/mt. Cyprus, Japan, and Turkey account for 457,000 mt of the total; only 33 pct of their production is available below the $43/mt cost of production.

Figure 12 illustrates estimated potential annual production of pyrite concentrate from 1985 to 1995. It is assumed that over the long run, all costs of production, including a 15-pct DCFROR, will be recovered. Therefore, potential annual capacities are shown for selected cost ranges that include a 15-pct DCFROR. The curves reflect the production capacity of existing mines, including planned expansions. Also, all operations produce at full (100 pct) capacity over the life of the mine.

An initial increase to 6.8 million mt/yr between 1986 to 1990, results from the expansion of the Portugese operation, Aljustrel. Italy, Spain, and Portugal account for 96 pct of this annual production, averaging about 4.2 million mt between 1985 and 1995. Annual pyrite concentrate production from operations in the Scandinavian countries averages about 1.4 million mt over the same time period. Estimated annual production from Cyprus, Japan, and Turkey averages about 900,000 mt between 1985 and 1995.

CONCLUSIONS

Nine domestic and twenty-five foreign properties were evaluated to determine the total tonnage of potentially recoverable elemental sulfur and pyrite concentrate. Based on the January 1985 in situ demonstrated ore resources of these properties, there is an estimated 267 million mt of recoverable elemental sulfur resources in the 14 native sulfur deposits, and 417 million mt of recoverable pyrite resources, as concentrate, in the 20 pyrite deposits.

Estimated potentially recoverable elemental sulfur (from native sulfur deposits) amounts to nearly 196 million mt, 98 pct of which is available at an average cost of production (0-pct DCFROR) below the January 1985 international market price, $143/mt. Over 90 pct of these resources occur at operations that were producing as of January 1985. In addition, about 16 million mt would be available from four U.S. past producers and one U.S. nonproducing operation.

Estimated potentially recoverable pyrite concentrate available (from sulfide ore deposits) amounts to 289 million mt, averaging 46 pct S. Eighteen properties are producers, two are currently nonproducers, one of which is under development. Approximately 96 pct (278 million mt) of these pyrite concentrate resources are available below an average cost of production (0-pct DCFROR) equal to the January 1985 pyrite concentrate market price, $43/mt. An additional 12 million mt is available at average costs of production ranging from $50/mt to $80/mt of pyrite concentrate.

Elemental sulfur production from the producing properties will continue over the next decade. Potential average annual production from these operations during this time is about 5.4 million mt. Nearly all of this sulfur is available at an average cost of production below $108/mt (15-pct DCFROR). Potential annual production begins to decline in 1988, with another decline occurring in 1994. Some of the decline could be counteracted by the expansion of production capacities from remaining producers, such as Iraq, which have large demonstrated resources, although such an expansion would effectively shorten their producing lives.

In addition, five nonproducing operations might be brought into production. Most of their potential annual production, about 1.0 million mt, would be available below a cost of production of $128/mt (15-pct DCFROR). Future supplies of elemental sulfur will most likely depend on increases in the production of secondary sulfur sources; i.e., recovered elemental sulfur from high-sulfur crude oils and sour gas from Canada, Mexico, the Middle East, and the United States.

Italy, Portugal, and Spain will dominate the pyrite concentrate industry well into the next century, with approximately 240 million mt pyrite concentrate available, at a potential average annual production of 3.6 million mt. This tonnage is available below an average cost of production of $43/mt (15-pct DCFROR). Additionally, an estimated tonnage of 49 million mt at a potential annual production of nearly 2.0 million mt is available from Cyprus, Japan, the Scandinavian countries, and Turkey. Most of this tonnage is also available below a cost of production of $43/mt.

These operations, however, must supply pyrite concentrates at a market price that is cost competitive to H_2SO_4 production from elemental sulfur. Substantial increases in pyrite concentrate output other than to meet internal or local H_2SO_4 demand is unlikely, owing to the supply of sulfur from its many primary and secondary sources.

REFERENCES

1. Morse, D.E. Sulfur. Ch. in Mineral Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 783-797.
2. Toon, S. Sulphur, A Sweet or Sour Future? Ind. Miner., No. 122, Feb. 1986, pp. 16-37.
3. Buckingham, D.A. Availability of Elemental Sulfur and Pyrite Concentrate—Market Economy Countries. A Minerals Availability Appraisal. BuMines IC 9106, 1986, 16 pp.
4. Bodenlos, A.J. Sulfur. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 605-618.
5. Hanna, M.A. Geology of the Gulf Coast Salt Domes. Ch. in Problems of Petroleum Geology, Sidney Powers Memorial Volume. Am. Assoc. Pet. Geol., 1934, pp. 629-678.
6. Barker, J.M., D.E. Cochran, and R. Semrad. Economic Geology of the Misraq Native Sulfur Deposit, Northern Iraq. Econ. Geol., v. 74, 1979, pp. 484-495.
7. British Sulphur. Iraq. Ch. in World Sulphur Resources. 2d ed., 1974, pp. 155-157.
8. ______. Turkey. Ch. in World Sulphur Resources. 2d ed., 1974, pp. 52-56.
9. Strauss, G.K., J. Madel, and F.F. Alosno. Exploration Practice for Strata-Bound Volcanogenic Sulphide Deposits in the Spanish-Portuguese Pyrite Belt: Geology, Geophysics, and Geochemistry. Springer-Verlag, 1977, 93 pp.
10. British Sulphur. Portugal and Spain. Ch. in World Sulphur Resources. 2d ed., 1974, p. 32.
11. Bear, L.M. The Mineral Resources and Mining Industry of Cyprus. Ministry of Commerce and Industry, Geol. Surv. Dep., Republic of Cyprus, Bull. 1, 1980, 208 pp.
12. Helovuori, O. Geology of the Pyhasalmi Ore Deposit, Finland. Ch. in Economic Geology and the Bulletin of the Society of Economic Geologists. V. 74, No. 5, Aug. 1979, pp. 1084-1101.
13. Olsen, J. Genesis of the Joma Statiform Sulfide Deposit, Central Norwegian Caledonides. Paper in Proceedings of the Fifth Quadrennial IAGOD Symposium. E. Schweizerbart'sche Verlagsbuchhaudlung (Nagele U. Obermiller), Stuttgart, Fed. Rep. of Germany, 1980, pp. 745-757.
14. Wilson, M.R. The Geological Setting of the Sulitjelma Ore Bodies, Central Norwegian Caladonides. Econ. Geol., v. 68, 1973, pp. 307-316.
15. Mining Magazine. Tverrfjellet—Norway's Highest Mine. V. 139, July 1978, pp. 14-23.
16. British Sulphur. Sweden. Ch. in World Sulphur Resources. 2d ed., 1974, pp 46-52.
17. Yilmaz, A. Geology of the Cayeli-Madenkoy Copper-Zinc Deposit and the Problems Related to Mineralization. Ch. in Bulletin of the Mineral Research and Exploration Institute of Turkey. Oct. 1977, No. 89, pp. 10-23.
18. Morse, D.E. Sulfur. Sec. in BuMines Minerals Commodity Summaries 1986, pp. 154-155.
19. Haynes, W. Brimstone: The Stone That Burns. Van Nostrand, 1959, 308 pp.
20. Green Markets. McGraw-Hill, v. 8, No. 1-51, 1984, 10 pp.
21. ______. McGraw-Hill, v. 9, No. 1-51, 1985, 10 pp.

TIN AND TANTALUM

INTRODUCTION

The most widely used tin applications are in the plating and solder industries. The United States is the world's largest producer and consumer of tin plate averaging about 2.5 million mt/yr production and 2.7 million mt/yr consumption since 1983. The United States is also the world's largest producer of secondary tin, tin scrap, and tinplate scrap, with 25 pct of its total 1983 consumption originating from secondary tin (*1*).[1] Other applications for tin are as an alloying agent in the production of pewter and brass and in the tin-based chemicals industry. Substitutions of other metals and plastics, caused by tin's high price, have contributed to a gradually decreasing tin consumption.

Tantalum is used for corrosion-resisting laboratory apparatus, in the manufacture of electronic equipment, in superalloys and carbides, and in ceramics and specific high-temperature applications.

Market economy country (MEC) consumption of primary tin metal was approximately 151,000 mt in 1985, with the United States and Japan accounting for the largest volume of metal consumed. Consumption in the United States has been steadily declining in the last 10 yr from a high of 59,000 mt in 1973 to less than 38,000 mt in 1984 (1984 was the largest domestic tin-consuming year in 3 yr) (*1*).

Tin is a bluish-white mineral whose native metallic element is Sn. The term is used loosely to designate cassiterite and concentrates containing cassiterite with minor amounts of other minerals. Tantalum is a metallic element that occurs in various rare minerals and principally in columbite-tantalite. It is a brittle, lustrous, hard, heavy, gray metal.

The analyses in this study include mining, milling, concentrating, and beneficiating to a refined tin metal product. Also included is the cost of transportation to the refinery. Tantalum is included in this chapter because most of the world's tantalum production is derived from the slag generated by tin-concentrate smelting; however, because tantalum provides no credit to the tin miners, in situ grades and recovery data were not gathered.

Most of the information in this chapter is an updated summary of Bureau of Mines Information Circular 9086 "Tin Availability—Market Economy Countries. A Minerals Availability Program Appraisal" (*2*). Additional information on the world tin industry is available from other Bureau publications (*3-6*).

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

GEOLOGY AND RESOURCES

GEOLOGY

The most common and economically viable tin mineral in the Earth's crust is cassiterite, SnO_2. Cassiterite is derived from granites and occurs in economically viable lode deposits as granite-hosted greissens, skarns, hydrothermal (vein), and replacement-type deposits. Weathering of these granites in lateritic environments often leads to the accumulation of economically significant placer tin deposits. One of the richest tin provinces in the world is a belt of placers approximately 1,600 km long and 200 km wide. It stretches from Indonesia, through Malaysia and Thailand, and into the Yunan Province in China. This belt includes offshore placer deposits that are also developed and mined. In the Kinta Valley, Malaysia, some weathered granites are being mined by alluvial methods even though the cassiterite has not been transported or accumulated as placer deposits.

Additional geologic information on tin and tantalum deposits is available from the U.S. Geological Survey (*7-8*).

RESOURCES

Lode deposits in Bolivia, England, Australia, and other countries make up over 25 pct of the world's tin resources. Placer deposits (including highly weathered, in situ, lode deposits mined by alluvial methods) make up the remainder of the world's demonstrated resources. The most productive alluvial deposits in the world are in southeast Asia and Brazil. As shown in table 1, approximately 21.8 billion mt of demonstrated tin resources containing 3.5 million mt tin have been estimated within MEC's. Figure 1 compares the tin reserve base with the resources evaluated in this study. The world reserve base is estimated to be 3 million mt of contained tin while the MEC deposits evaluated in this study account for an estimated 3.5 million mt. Resource estimates may be understated because of deposits known to exist in China and the U.S.S.R.; resource data from both of these countries are difficult to obtain.

Current world reserves of tantalum are approximately 34,500 mt, with Australia, Thailand, and Brazil being the world's major producers. Tantalum-rich slags from smelted tin concentrates are the major source of tantalum. Tantalum-rich slags from Thailand, which assay up to 15 pct tantalum pentoxide (Ta_2O_5), may account for as much as 30 pct of world tantalum production. These slags are also produced in Indonesia, Malaysia, Nigeria, Rwanda, Singapore, Zaire, and Zimbabwe.

Properties evaluated in this study are listed in table 2.

Table 1.—Summary of MEC demonstrated tin resources evaluated for this study, as of January 1985

Country	Number of deposits	In situ ore: Tonnage,[1] 10^6 mt	In situ ore: Grade, pct	Tin,[1] 10^3 mt: Contained	Tin,[1] 10^3 mt: Recoverable
Argentina	1	W	W	W	W
Australia	8	298	0.101	300	175
Bolivia	26	239	.077	184	120
Brazil	17	134	.047	63	58
Burma	2	24	.070	17	8
Canada	1	58	.198	112	59
Indonesia	7	4,417	.019	832	660
Japan	2	5	.277	13	7
Malaysia	36	14,603	.009	1,252	1,032
Namibia	2	53	.141	75	56
Nigeria	1	129	.015	19	15
Peru	1	W	W	W	W
South Africa, Rep. of	2	9	.621	56	27
Thailand	25	1,766	.021	366	237
United Kingdom	6	35	.351	1	
				23	82
United States	1	25	.151	38	19
Zaire	1	7	.292	21	17
Zimbabwe	1	12	.209	24	15
Other	([2])	2	2.480	38	33
Total or av	140	21,814	.016	3,535	2,620

W Withheld to avoid disclosing company proprietary deposit data; included in "other."

[1]Rounded.

[2]Represents the aggregated ore from Argentina and Peru.

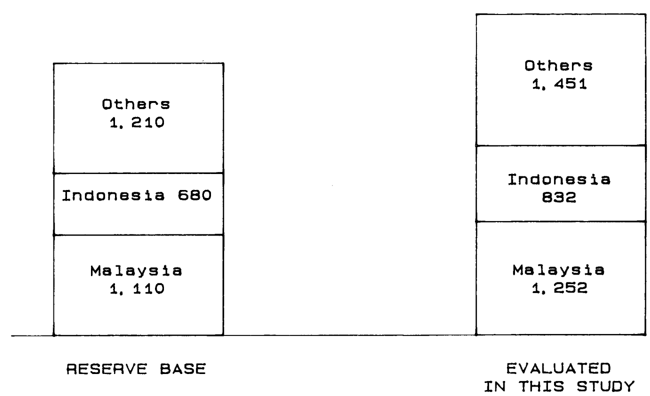

Figure 1.—Estimates of world tin resources (thousand metric tons of contained tin).

Table 2.—MEC tin properties included in this study

Country and deposit	Owner and/or operator	Mining method	Current status[1]
Argentina: Pirquitas	Sociedad Minera Pirquitas Pichetti y, Cia., S.A.	Underground	P
Australia:			
Ardlethan	Aberfoyle Ltd.	Open pit	P
Baal Gammon	Newmont Holding Pty. Ltd.	..do	N
Cleveland	Aberfoyle Ltd.	Underground	P
Greenbushes	Greenbushes Tin NL	Open pit	P
Mt. Bischoff	CRA Exploration Pty. Ltd. (51 pct)	..do	N
Mt. Garnet-Ravenshoe	Oakbridge Ltd..	Dredge	P
Pioneer	Triako (45.25 pct), Buka (45.25 pct)	Gravel pump	P
Renison	Renison Goldfields Consolidated Ltd.	Underground	P
Ruxton Area	Metals Exploration Ltd.	Open pit	P
Taronga	Newmont Holdings Pty., Ltd.	..do	N
Bolivia:			
Atoroma	Empresa Minera Atoroma Ltda.	Underground	P
Avicaya	..do	..do	P
Bolivar	Corporacion Minera de Bolivia (COMIBOL)	..do	P
Carocoles	..do	..do	P
Cerro Rico	Banco Minero de Bolivia (BAMIN)	..do	P
Chojlla	International Mining Co	..do	P
Chorolque	COMIBOL	..do	P
Colavi	..do	..do	P
Colquiri	..do	..do	P
Comsur	Compania Minera del Sur S.A. (COMSUR)	Dredge	P
El Centenario	COMIBOL	..do	N
Estalsa	Estalsa Boliviana	Gravel pump	P
Do	..do	Dredge	P
Huanuni	COMIBOL	Underground	P
Japo	..do	..do	P
Kellguani	Empresa Minera Kellguani	..do	P
La Reyna	Sabino Llave	..do	P
Milluni	COMSUR	..do	P
Morococala	COMIBOL	..do	P
Potosi	..do	..do	P
San Jose de Ayata	Empresa Minera Pabon Ltda	..do	P
San Juan	Waldo Sarmiento	..do	P
Santa Barbara	Santa Barbara Mining Co	..do	P
Santa Fe	COMIBOL	..do	P
Suka	Compania Minera Suka Ltd.a	..do	P
Totoral	Cia. Minera Orlandini Ltd.a	..do	P
Viloco	COMIBOL	..do	P
Yana Mallcu	Empresa Minera Yana Mallcu	..do	P
Brazil:			
Cachoeirinha-Bom Futuro	Moacyrmotta Brumadinho	Dredge	P
Candeias-Sao Domingo	Empresas Brumadinho S.A.	..do	P
Ceriumbras	Best Metais e Soldas	..do	P
Cortez	Brascan Recursos Naturais S.A.	..do	N
Duduca	..do	Gravel pump	N
Massangana	Paranapanema S.A. Mineracao, Industria e Construcao, operator, local owner.	..do	P
Mocambo	Mineracao Sao Jose	..do	N
Novo Mundo	Brascan Recursos Naturais S.A.	..do	P
Pitinga	Paranapanema	..do	P
Poco	Brascan Recursos Naturais S.A.	..do	P
Potosi	..do	..do	P
Potosi Hill	..do	Open pit	P
Rhodia-Espeng	Canopus	Gravel pump	P
San Laurenco	Empresas Brumadinho S.A.	..do	P
Sao Francisco	Paranapanema S.A. Mineracao, Industria e Construcao, operator, local owner.	..do	P
Sao Raimundo	..do	Dredge	P
Serra Branca	Mineracao Gondwana	Open pit	N
Taboquinha	Brascan Recursos Naturais S.A.	Gravel pump	N
Burma:			
Mawchi	Burmese Government	Underground	P
Tenasserim Valley gravel pump	..do	Gravel pump	P
Canada: East Kemptville	Rio Algom Ltd.	Open pit	N
Indonesia:			
Bima dredge	P.T. Riau Tin Mining	Dredge	P
Kelapa Kampit	P.T. Broken Hill Pty. Indonesia	Underground	P
P.T. Koba Tin	Kajuara Mining Corp. (Pty.) Ltd	Gravel pump	P
P.T. Tambang Timah (Banghah)	P. T. Tambang Timah	Dredge	P
Do	..do	Gravel pump	P
P.T. Tambang Timah (Belitung)	..do	Dredge	P
Do	..do	Gravel pump	P
Japan:			
Akenobe	Akenobe Mining Corp. Ltd	Underground	P
Suzuyama Tin	Kyowa Mining Co	..do	P
Malaysia:			
Austral Amalgamated	Austral Amalgamated Tin Bhd	Dredge	P
Ayer Hitam	Ayer Hitam, Tronoh, MMC	Gravel pump	P
Do	..do	Dredge	P
Berjuntai	Malaysian Mining Co	..do	P
Bidor Malaya	Bidor Malaya Tin Sdn. Bhd.	..do	P
Gopeng Consolidated gravel pumps	Gopeng Consolidated PLC	Gravel pump	P
Johor State	Multiple ownership	..do	P
Kedah gravel pump mines	Privately owned	..do	P

See footnote at end of table.

Table 2.—MEC tin properties included in this study—Continued

Country and deposit	Owner and/or operator	Mining method	Current status[1]
Malaysia—Continued			
Killinghall	Ramuda Sdn. Bhd. and Straits Trading Co	Gravel pump	P
Kinta Kellas dredge	Kinta Kellas Tin Dredging Bhd	Dredge	P
Kramat dredge	Kramat Tin Dredge Bhd	..do	P
Kuala Langat	Kumpulan Perangsang Selangor	..do	N
Lower Perak dredge	Lower Perak Tin Dredging Bhd	..do	P
Malayan dredge	Malaysian Mining Co	..do	P
Mambang Di-Awan	Gopeng Consolidated, Syarikat Permodalan	..do	P
ML4	Kumpulan Perangsang Selangor	..do	P
Modal Sri Pandan	Pahang Development Corp. (70 pct), Conzinc Riotinto Malaysia (30 pct).	..do	P
Pacific Tin Consolidated	Pacific Tin Consolidated Corp. (85 pct)	..do	P
Pahang gravel pump	Multiple ownership	Gravel pump	P
Perak State	..do	..do	P
Perangsang Berjuntai	Permodalan National Bhd. (70 pct), Berjuntai Tin Dredging Bhd. (30 pct).	Dredge	P
Petaling	Petaling Tin Bhd	..do	P
Rahman Hydraulic	Rahman Hydraulic Tin Bhd	Gravel pump	P
Selangor dredge	Selangor Dredging Bhd	Dredge	P
Selangor gravel pumps	Multiple owners	Gravel pump	P
Southern Kinta Consolidated	Malaysia Mining Co	Dredge	P
Southern Malayan dredge	..do	..do	P
Sungei Besi Mines	Sungei Besi Mines Ltd	Open pit	P
Sungei Lembing	Pahang Consolidated PLC	Underground	P
Syarikat Lombong Sebina	Pahang Development Corp. (70 pct), Conzinc Riotino Malaysia (30 pct).	Dredge	N
Timah Dermawan	Malaysian Mining Co. (40 pct), Tronoh (30 pct), Perak State (30 pct).	..do	P
Timah Langat	Timah Langat Bhd	..do	P
Do	..do	Gravel pump	P
Do	..do	Dredge	P
Timah Matang	Malaysia Mining Co	Dredge	P
Tronoh Mines Malaysia	..do	..do	P
Namibia:			
Brandberg West	South West Africa Ltd	Open pit	N
Uis	Industrial Minerals Mining Corp. (Pty) Ltd	..do	P
Nigeria: Amalgamated tin mine, Bisichi	Nigerian Mining Corp	Dredge	P
Peru: San Rafael	Minsur, S.A	Underground	P
South Africa, Republic of:			
Rooiberg Mine	Rooiberg Tin Ltd	..do	P
Union Tin Mines	Union Tin Mines Ltd	..do	P
Zaaiplaats tin mine	Zimro and Zaaiplaats Tin Mining	..do	P
Thailand:			
Aokam	Aokam Thai Ltd	Dredge	P
Bangrin tin dredge	Fairmont State Ltd	..do	P
Batun Mine	Phuket Mining Co. Ltd	Gravel pump	P
Bodan dredge	Thai Government (Offshore Mining Organization	Dredge	P
Central Region gravel pump	Multiple local ownership	Gravel pump	P
Charintr	Charintr Mining Co	..do	P
Eurothai Mine	Eurothai Mining Co	Open pit	P
Hok Chong Seng	Hok Chong Seng	Gravel pump	P
Hok Chong Seng suction dredge	..do	Dredge	P
Labu Mine	Sawad Wattonayagorn	Open pit	P
Narai dredge	Billiton Nederland BV (operator), Thai Government (owner)	Dredge	P
Ngan Thawi Brothers Co	Ngan Thawi Brothers Co	Gravel pump	P
Do	..do	Dredge	P
Nok Hoog Mine	Sombat Co., Ltd	Gravel pump	P
Northern Region gravel pumps	Multiple local ownership	..do	P
Phangnga Suction Boats	Thai Government	Dredge	P
Pinyok Mine	Bandit Tantivit	Open pit	P
Sethasup Karnrae Co. Ltd	Sethasap Karnral Co	Dredge	P
Siamese Tin Syndicate Ltd	Fairmont State Ltd	..do	P
Sinchon Mine	..do	Underground	P
Sierra Mining Co	Thai Nationals and Pacific Tin	Gravel pump	P
Southern Region gravel pumps	Multiple local ownership	..do	P
Tongkah Harbour	Tongkah Harbour Ltd	Dredge	P
Yip In Tsoi—Yala pump	Yip In Tsoi	Gravel pump	P
Yip In Tsoi—Haad Yala Ope	Yip In Tsoi Pit	Open pit	P
United Kingdom:			
Geevor tin mine	Geevor Tin Mines Ltd. (RTZ Ltd.)	Underground	P
Marine Mining	Marine (Cornwall) Mining Ltd. (consortium)	Dredge	N
Redmoor	Southwest Minerals Ltd	Underground	N
South Crofty Pendarves district	South Crofty PLC	..do	P
Wheal Concord property	Wheal Concord Ltd	..do	P
Wheal Jane-Mt. Wellington	Carnon Consolidated Tin Mines Ltd. (RTZ Ltd.)	..do	P
United States: Lost River	Bering Straits Native Corp	Open pit	N
Zaire: Kivu Mine	Sominki	Gravel pump, open pit	P
Zimbabwe: Kamative tin mine	Industrial Development Corp. of Zimbabwe	Underground	P

[1] P, producer; N, nonproducer or undeveloped.

U.S. AND WORLD HISTORICAL PRODUCTION

Since the suspension of tin trading on the London Metal Exchange (LME) in October 1985, many of the higher cost operations include in this study have halted production in expectations of low tin prices being established when trading resumes. Historic production, in 5-yr intervals from 1950 to 1985, is illustrated in figure 2.

MEC production of primary tin metal in 1985 was 157,750 mt, with Malaysia, Brazil, Indonesia, and Thailand, respectively, being the top four producers and contributing over 65 pct of the total (*2*).

INTERNATIONAL TIN AGREEMENTS

Most of the MEC tin producers are members of the International Tin Council created in 1953. This organization oversees tin exports and prices. Several factors led to the suspension of tin trading on the world's major tin markets in late 1985 and a 50-pct reduction in the price of tin. This section will briefly explain the history of the ITC in order to place into perspective the current status of the tin industry.

The First International Tin Agreement (ITA) of 1931-33, and the two following agreements (1934-36 and 1937-41), were established in reaction to the collapse in world tin prices, largely brought about by the Great Depression. The agreements were set up to control tin supplies and to stabilize tin prices through a system of Government-controlled quotas. In 1938, a tin inventory, or buffer stock, with floor and ceiling price levels, was adopted to prevent rapid tin price fluctuations. Price ranges in the buffer stock were set to LME quotes. Members of the International Tin Committee (established by the first ITA) would contribute to the buffer stock on a percentage basis of their production.

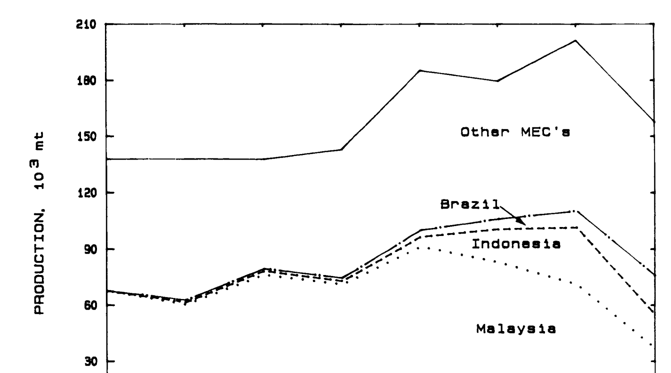

Figure 2.—MEC primary tin metal production, 1950-1985.

The First Post-War International Tin Agreement was established in 1953 and created the International Tin Council (ITC). Brazil, Bolivia, and China, three major tin producers, and the United States, a major consumer, are not members of the current agreement.

In order to pursue their objectives, the ITC was empowered with two important tools: control of exports and management of the buffer stock. In the Sixth (postwar) Agreement, starting in 1982, the conditions under which the Buffer Stock Manager would operate were outlined and included the responsibility that, if the tin market price were equal to or lower than the floor price, the manager must buy tin until the price is above the floor price or until purchasing funds are exhausted.

Major production increases from non-ITA tin producers (there was a 200-pct tin production increase in Brazil between 1980 and 1985) and a gradually decreasing MEC consumption of tin (due to less tin used in tinplating, competition from aluminum, and the replacement of tin by other substitute materials) resulted in decreasing demand in the early 1980's and an ITC floor price that was becoming difficult to maintain. By 1985, a 40-pct cutback on tin exports imposed on several producing members had been in effect for several years and the Buffer Stock Manager was buying tin to maintain a floor price of approximately \$5.50/lb to \$6/lb refined tin. In October 1985, the world's major tin trading markets, the LME and the Kuala Lumpur Tin Market (KLTM), suspended tin trading in reaction to the Buffer Stock Manager's statement that purchasing funds were exhausted, no more credit existed, and the ITC had ceased its buffer stock support.

EXTRACTION AND PROCESSING TECHNOLOGY

Tin is recovered essentially by four types of mining methods: gravel pumps and dredging (for alluvial deposits), and open pit and underground methods (for lode deposits). Tin recovered from dulong washers accounted for about 5 pct of the total tin produced in Malaysia in 1982. Tin produced from dulong washers was not evaluated in this study.

ALLUVIAL DEPOSITS

Gravel Pumps

Gravel pumps have been an important tin mining method for at least 70 yr; more than 30 pct of the world's tin is currently recovered using gravel pumps. Approximately half of Malaysia's and Indonesia's tin production, one-third of Thailand's, and most of Brazil's tin production originates from gravel pump operations.

A gravel pump consists of three primary sections: monitors (high-pressure water nozzles), pumping stations, and a concentrating section. Gravel pump monitors are designed to excavate exposed faces of virgin ground, tailings, and previously dredged ground, especially around exposed limestone pinnacles. Exposed faces can range in thickness from 1 to over 60 m. High-pressure water directed through the nozzles of the monitor serves to break up the tin-bearing ground. The monitors are generally placed about 35 m from the exposed face. The resulting slurry flows through channels to the pumping station.

The pumping station consists of an excavation or sump that is designed to receive the slurry. The sump serves to separate driftwood, stones, clay, and other undesirable materials from the gravel. The gravel is pumped from the bottom of the sump to a trommel. (A trommel is a revolving, generally cylindrical screen that separates out the oversized material.) The screened material is then directed to the palongs which are long (up to 50 or 60 m), inclined sluices lined with riffles. Fine-grained material requires use of a longer palong in order to allow time for the fines to settle out. The riffles capture fragments of cassiterite as well as other accompanying heavy minerals, such as zircon, apatite, monazite, and rutile. The flow of slurry is periodically stopped so the material collected in the riffles can be removed. The collected material (tin concentrate) is then dewatered and fed to a jig plant. (A jig is a mechanical device designed to utilize gravity for separation of tin—essentially a box with a screen bottom by which the action of water currents agitates the tin concentrate, leaving the most dense material at the bottom of the box.) The waste material is returned to the mine site to be used as landfill. The concentrates are then directed to the tin shed. (See "Tin Shed" section.)

Dredging

Onshore dredges have been used to recover tin from alluvial deposits since the turn of the century. A dredge's basic function is to serve as a self-contained, floating excavating machine on which is mounted a screening and jigging plant. There are basically two types of dredges in common use: bucket-line dredges and suction dredges. Both types require slight modifications to mine offshore.

The bucket-line dredge is the most common type of dredge. It consists of a hull, generally constructed of pontoons, upon which are mounted the digging mechanism and on-board concentrator plant.

The digging mechanism on a bucket-line dredge consists of a series of buckets connected to a chain mounted on a ladderlike structure. As the buckets rotate around the ladder in a Ferris wheel fashion, material is excavated and dropped into a distribution chute. At times the dredge is used to remove overburden in order to reach the "pay zone." The removal of material ahead of the dredge produces a pond on which the dredge can advance. The dredge's heading is guided by an on-board cable attached to a land anchor. Most dredges are operated by electricity (supplied by cable) but in remote areas, diesel-electric generators or, more rarely, steam-driven generators are used.

Offshore dredges started operating off the coast of Thailand in about 1907. The success of the operation encouraged the construction and use of offshore dredges in Indonesia. An attractive aspect of offshore dredging is that much of the overburden has been winnowed out by wave and tidal action, leaving behind heavier alluvial sands and gravels. However, bad weather, tidal currents, and wave action cause offshore dredges to mine at substantially lower efficiencies than their onshore counterparts, resulting in higher unit costs.

Suction dredges are much less common than bucket-line dredges. Large suction dredges find their greatest application in relatively shallow waters. A suction hose surrounded by high-pressure water jets or cutterheads delivers material dislodged by the high-pressure water or cutters to the surface. Among the disadvantages of suction dredging are the high amount of slimes produced and the occasional clogging of the cutters and suction pipe by debris.

After the ore is screened, it is sprayed by high-pressure hoses to break up clay balls and other loosely aggregated material. The material is then rescreened, and the undersize material is sent to a series of jigs. The final tin concentrate generally assays between 20 to 30 pct and yields about a 95-pct Sn recovery. The concentrate is further processed at a land-based tin shed.

Tin Shed

The tin shed is a centrally located treatment plant designed to further upgrade tin concentrates recovered from alluvial deposits to about 70 to 75 pct Sn (90-95 pct cassiterite). The plant consists of a relatively simple series of gravity methods, including jigs and tables. In some cases, a tin shed can be relatively complicated, using acid leaching to remove carbonates, flotation to remove sulfides (most commonly pyrite and arsenopyrite), and magnetic or electrostatic methods to remove iron minerals, zircon, ilmenite, columbite, and monazite. Some of these minerals have economic value but are not generally sold on a regular basis. After upgrading, the concentrate is usually dried and then shipped to the smelter.

LODE DEPOSITS

Underground and open pit mining methods are employed to mine most of the lode tin deposits evaluated for this study. Ores associated with lode deposits are mineralogically more complex than ores of alluvial deposits. Lode deposits are often associated with pyrite, pyrrhotite, arsenopyrite, and various silicate minerals. Additional minerals such as wolframite, scheelite, chalcopyrite, galena, sphalerite, stibnite, bismuthinite, gold, and silver, may occur in sufficient quantities to warrant recovery as a coproduct or byproduct. In nearly all cases, gravity methods are used to recover the tin. However, flotation circuits may also be part of the beneficiation circuit if the ore is complex. As in most beneficiation plants, the ore is first crushed. After crushing, a high-grade concentrate may be isolated by hand sorting for direct shipment to a smelter. Hand sorting is practiced at some mines in Bolivia and Southeast Asia. More typically, the ore is sent through a complex series of heavy mineral separation steps including jigs, tabling, and hydroclassifiers. Depending on the complexity of the ore, several grades and compositions of concentrates may be produced.

PROCESSING

Fuming of Concentrates

Over the past few years, direct fuming of low-grade tin concentrates has become an important addition to tin smelting technology. Fuming is performed when the tin grade is between 5 and 25 pct. The fuming process converts the tin-in-concentrate to a gaseous stage, from which a tin oxide dust is recovered. The tin in cassiterite reacts with sulfur present in pyrite and other sulfides to form stannous sulfide (SnS), which, in the presence of air, oxidizes back into a stannous oxide (SnO_2) or fume. The recovered fume (tin dust) yields a 45- to 60-pct Sn concentrate. The fume is then converted to metallic tin in a conventional smelting and refining complex.

Smelting

Medium-grade concentrates usually originate from lode deposits that include large amounts of impurities. Smelting results in two products: a crude tin metal, which is poured off and sent to the refiner, and large volumes of slag.

Smelting is performed in reverberatory, rotary, or electric smelter furnaces. The choice is generally dictated by economic rather than technical factors. For example, most Malaysian smelters use oil-fired reverberatory or rotary plants rather than electric smelters owing to the ready availability of oil. In some African countries, electricity is used because of the availability of inexpensive hydroelectric power.

The first smelting step consists of heating a concentrate to the point where it becomes molten. At this point, an iron-rich slag and a crude tin metal can be separated. The impure tin, which generally assays over 90 pct Sn, is directed to the refinery. The iron-rich slag is directed to a smelting furnace where higher temperatures and a reducing environment are produced. These conditions result in separation of the melt into a reject slag and hardhead, a tin-iron alloy. The hardhead is then returned to the initial smelting furnace, and the process is repeated. Hardhead usually assays about 60 to 70 pct Sn and 25 to 40 pct Fe. The discarded reject slag assays only 0.5 to 0.8 pct Sn and 8 to 12 pct Fe.

Refining

Tin produced from the smelting of tin concentrates contains minor amounts of impurities, most importantly, iron, lead, copper, bismuth, arsenic, antimony, cobalt, nickel, zinc, and cadmium. In order to attain a purity of 99.8 pct Sn, virtually all of the impurities must be removed by the refining process. The refining of tin is performed by either pyrometallurgical or by electrolytic processes.

Pyrometallurgy

In the liquation process of pyrometallurgical refining techniques, impure tin is heated to just above its 232° C melting point. At this temperature, an iron-tin phase remains as a solid when the relatively pure tin is poured off. The remaining iron-tin material (dross), is returned to the smelter complex. This step is repeated until separation of the solids is nearly complete. Vacuum distillation is a relatively new development in pyrometallurgical tin refining. The process requires that molten tin be contained in a vessel of dense graphite at high temperatures (1,100°-1,300° C) and be subjected to a vacuum. Using this process, impurities can be selectively distilled by application of specific temperatures and lengths of time. The vacuum distillation process is likely to replace some electrolytic refining plants.

Electrolytic Refining

Removal of impurities can also be accomplished by electrolytic refining. Electrolytic refining is generally classified into two general types, using either acid or alkaline electrolytes.

Both processes consist of immersing impure tin anodes and pure tin cathodes in a bath of electrolytes. The passage of a direct electric current through the tank or cell causes the anode tin to dissolve and deposit on the cathode. Anode impurities must be removed from time to time to maintain the effectiveness of the electrolyte.

BYPRODUCTS OF TIN SMELTING AND REFINING

The smelting and refining processes are capable of isolating several salable byproducts. Some facilities in Bolivia, Malaysia, the United Kingdom, and the United States recover lead, bismuth, antimony, copper, and tungsten. An important byproduct of tin processing is tantalum, either contained in slag or as a pentoxide. No credit is paid to the mines for the tantalum content in concentrate owing to its low assay levels.

PRODUCTION COSTS

Costs were determined for mining, milling, smelting, refining, transporation (f.o.b. refinery), taxes, and royalties. Capital costs estimated for undeveloped deposits reflect the estimated total investment required to develop a deposit and bring it into production.

All costs and investments were aggregated by country and by production status. Data were further aggregated by mining methods to facilitate comparisons.[2] Production costs presented are weighted-averages calculated over a deposit's life. These costs include labor, materials, energy, administration, and transportation.

Of the 140 deposits evaluated, 124 were identified as producing. The weighted-average mining and beneficiation cost for all producing tin mines is $1.10/mt ore (table 3). This cost applies to approximately 80 pct of the recoverable tin metal estimated to exist in MEC's evaluated in this study.

Almost 85 pct of the recoverable tin metal in MEC's is from Malaysia, Thailand, and Indonesia. The weighted-average mining and beneficiation costs estimated for producing deposits in these three countries are the lowest in this study, at $1.40/mt ore or less.

The estimated mining and beneficiation cost among the 16 undeveloped deposits or regions is $1.70/mt ore. This value is weighted heavily by Malaysia's Kuala Langat deposit, which is scheduled to go on line in the late 1980's. When this multiple dredge operation reaches its projected capacity in the 1990's, it will be the largest tin producer in Malaysia.

[2] In order to maintain the proprietary nature of certain data, several deposits were aggregated in "others" sections in the tables and in the discussion.

The average mining and beneficiation cost among producing deposits is $4.10/lb refined tin (table 4; fig. 3). The lowest cost tin-producing country is Brazil, where gravel pump and dredge operations produce tin from relatively high-grade deposits. In situ grades in these operations range from two to five times higher than average in situ grades

Table 3. Tin average feed grade and mining and beneficiation costs for selected MEC's[1]

Status and country	Number of mines	Av feed grade, pct Sn[3]	Ore cost,[2] $/mt Mining	Ore cost,[2] $/mt Beneficiation	Ore cost,[2] $/mt Total
Producing mines:					
Australia	5	0.069	3.70	2.60	6.30
Bolivia	25	.238	9.00	3.30	12.30
Brazil	12	.040	1.40	.60	2.00
Indonesia	7	.019	.80	.60	1.40
Malaysia	34	.009	.60	.10	.70
South Africa, Rep. of	2	.590	10.60	8.60	19.20
Thailand	25	.021	.70	.50	1.20
United Kingdom	4	.711	30.40	14.20	34.60
Other Africa[4]	4	.060	2.00	1.00	3.00
Others[5]	6	.211	7.40	2.20	9.60
Subtotal or av	124	.015	.80	.30	1.10
Undeveloped deposits[6]	16	.029	.90	.80	1.70
Grand total or av	140	.016	.80	.30	1.10

[1] Based on 1982 data; costs updated to constant January 1985 U.S. dollars.
[2] Weighted-average basis.
[3] Rounded.
[4] Includes Namibia, Nigeria, Zaire, and Zimbabwe.
[5] Includes Argentina, Burma, Japan, and Peru.
[6] Includes properties in Australia, Bolivia, Brazil, Canada, Malaysia, Namibia, United Kingdom, and United States; costs not presented by country because of the small number of deposits.

Table 4.—Tin production costs for selected MEC's[1]

(All costs are in January 1985 U.S. dollars per pound of refined tin on a weighted-average basis)

Status and country	Operating cost: Mine	Operating cost: Mill	Operating cost: Other[2]	Byproduct credit	Net operating cost	Recovery of capital[3]	0-pct DCFROR: Taxes and royalties[4]	0-pct DCFROR: Total cost[5]	15-pct DCFROR: Taxes and royalties[6]	15-pct DCFROR: Return on investment[7]	15-pct DCFROR: Total cost[8]
Producing mines:											
Australia	3.30	2.30	0.50	1.30	4.80	0.60	0.10	5.50	0.50	0.50	6.40
Bolivia	2.70	1.00	1.40	1.10	4.00	.60	2.10	6.70	2.40	.40	7.40
Brazil	1.60	.70	.20	0	2.50	.60	.70	3.80	.90	1.50	5.50
Indonesia	2.10	1.50	.20	0	3.80	.70	.20	4.70	.50	.50	5.50
Malaysia	3.90	.80	.10	0	4.80	.20	.20	5.20	.40	.20	5.60
South Africa, Rep. of	1.50	1.30	.00	0	2.80	.40	.00	3.20	.30	.40	3.90
Thailand	2.20	1.40	.10	.10	3.60	.30	1.60	5.50	1.90	.20	6.00
United Kingdom	2.30	1.10	.90	1.00	3.30	.80	.20	4.30	.70	.60	5.40
Other Africa[9]	1.80	.90	.20	.20	2.70	.60	.10	3.40	.60	.60	4.50
Others[10]	1.90	.50	2.70	2.00	3.10	.60	.10	3.80	.80	.10	4.60
Subtotal or av	2.90	1.20	.30	.20	4.20	.40	.40	5.00	.70	.60	5.90
Undeveloped deposits[11]	2.10	1.90	.90	1.10	3.80	1.60	.50	5.80	2.50	2.40	10.20
Grand total or av	2.80	1.20	.40	.30	4.10	.60	.40	5.10	.90	.60	6.20

[1] Based on 1982 data; costs updated to constant January 1985 U.S. dollars.
[2] Includes smelting, refining, and transportation of concentrates to the smelter and refinery.
[3] Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure as of January 1985, and investments required over the life of the operation.
[4] Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 0-pct DCFROR.
[5] Equal to sum of net operating costs, taxation, and capital recovery determined at a 0-pct DCFROR.
[6] Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 15-pct DCFROR.
[7] Revenue increase per pound necessary to obtain a 15-pct DCFROR.
[8] Equal to sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery *plus* the per pound increase necessary to provide a 15-pct DCFROR after taxation.
[9] Includes properties of Namibia, Nigeria, Zaire, and Zimbabwe.
[10] Includes properties in Argentina, Peru, Japan, and Burma.
[11] Includes properties in the United States, Australia, Bolivia, Brazil, Canada, Malaysia, Namibia, and United Kingdom.

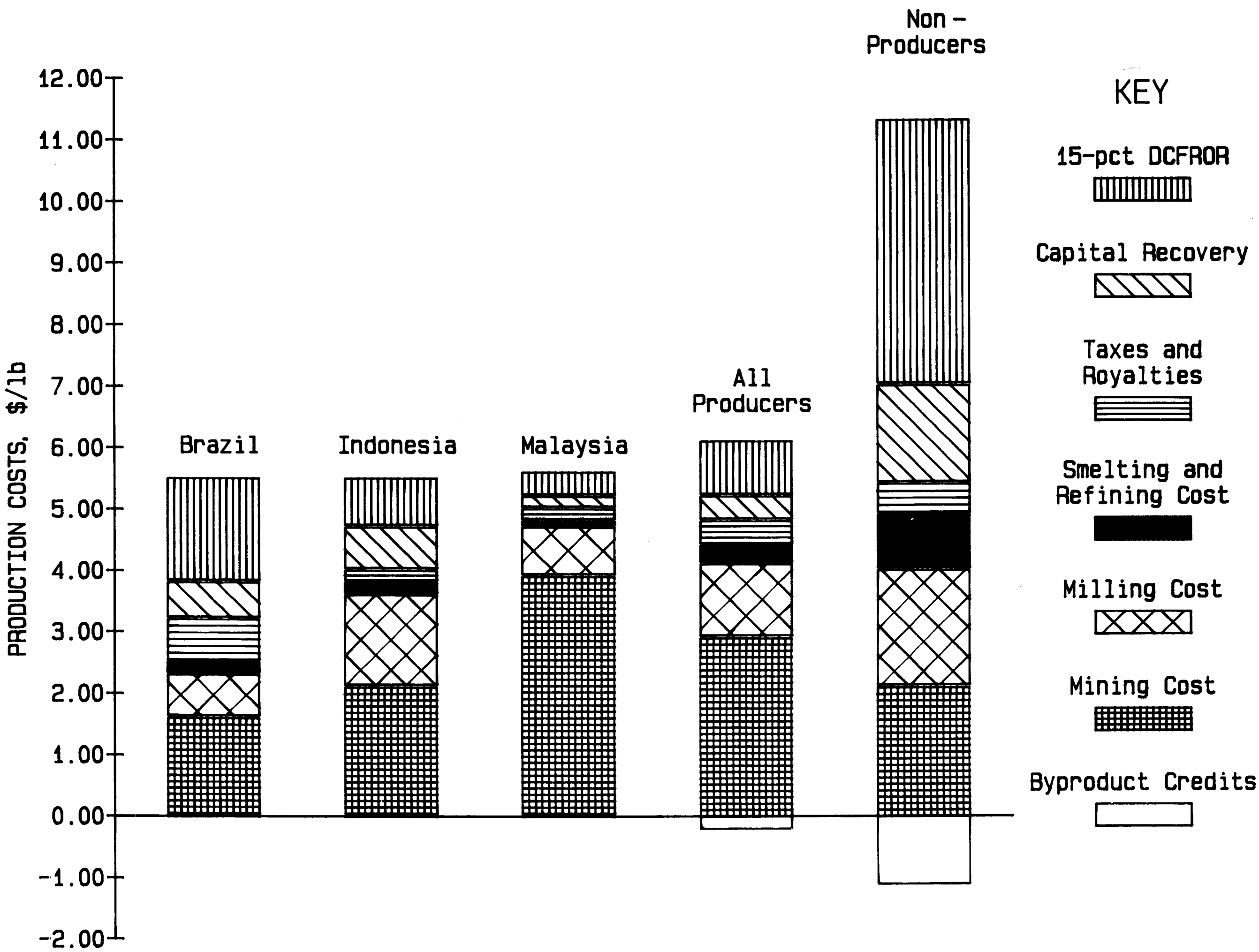

Figure 3.—Tin production costs in selected MEC's (January 1985 U.S. dollars).

for gravel pump operations in southeast Asia. Large tin resources, low production costs, and increasing tin production, have allowed Brazil to become a major producer in the international tin industry.

COSTS BY MINING METHOD

Gravel Pumps

Over 12 billion mt of tin ore is estimated to be recoverable from producing gravel pump operations evaluated in this study (table 5). This accounts for almost 65 pct of the recoverable tin metal available from all evaluated MEC producing deposits. Potential production from Perak State, Malaysia, accounts for almost 40 pct of the recoverable tin metal among producing gravel pump operations and over 20 pct of the recoverable tin metal from all producing mines evaluated in this study.

Weighted-average mining and beneficiation costs estimated for gravel pump operations range from $0.80/mt ore in Malaysian deposits to an aggregated cost of $3.40/mt ore for deposits in Australia, Bolivia, Burma, and Zaire.

Weighted-average mining and beneficiation cost among gravel pump operations is $4.60/lb of refined tin, with Brazil having the lowest cost at $2.40/lb refined tin (table 6). With an average in situ grade of 0.009 pct Sn, Malaysian deposits are the lowest grade deposits (among evaluated countries) producing tin by gravel pump methods. Although byproduct tantalum is recovered with tin in Indonesia, Malaysia, and Thailand, byproduct recovery data from alluvial deposits are incomplete and were not included in this study.

The total refined tin production cost (0-pct discounted-cash-flow rate of return [DCFROR]) for all producing gravel pump operations is about $5.30/lb. The highest cost gravel pump operations are in Thailand at $6.40/lb, with Brazil's $3.80/lb being the lowest cost

Dredges

Dredging operations account for over 30 pct of the recoverable tin metal among producing MEC deposits. Dredging is also the lowest cost producing method among all tin mining types. Operating costs for dredges vary depending on the type of dredge (onshore or offshore), the characteristics of the ore body, and the depth and type of overburden. Offshore dredges have more down time because of prolonged monsoon seasons, and usually operate only about 9 months out of the year. Consequently, offshore costs

Table 5.—Tin resources, feed grades, and mining and beneficiation costs, by mine method, for selected MEC's[1]

Mining method and country	Demonstrated resources, 10^6 mt	Feed grade, pct	Ore cost,[2] $/mt Mining	Ore cost,[2] $/mt Beneficiation
Gravel pump:				
Brazil	67.1	0.043	2.30	(3)
Indonesia	1,239.1	.023	1.90	(3)
Malaysia	10,504.7	.009	.80	(3)
Thailand	764.1	.020	1.50	(3)
Others[4]	50.7	.077	3.40	(3)
Total or av	12,623.6	.011	1.00	(3)
Dredge:				
Indonesia	2,510.6	0.015	1.10	(3)
Malaysia	2,336.4	.007	.40	(3)
Thailand	802.8	.019	.80	(3)
Others[5]	369.8	.012	.90	(3)
Total or av	6,019.6	.012	.80	(3)
Underground:				
Bolivia	15.1	.791	31.00	11.30
South Africa, Rep. of.	8.5	.590	11.00	8.60
United Kingdom	5.8	.711	30.40	14.80
Southeast Asia[6]	5.2	1.107	35.20	11.70
Others[7]	31.0	.620	18.70	11.10
Total or av	65.8	.703	23.00	11.20
Open pit:				
Australia	45.8	.056	10.70	5.80
Thailand	3.5	.532	7.90	2.80
Others[8]	59.5	.132	2.90	2.00
Total or av	108.7	.113	6.30	3.60

[1] Based on 1982 data; costs updated to constant January 1985 U.S. dollars; data do not add to totals shown because of independent rounding.
[2] Weighted-average basis.
[3] Combined with mining cost.
[4] Includes properties of Australia, Bolivia, Burma, and Zaire.
[5] Includes properties in Australia, Bolivia, Brazil, and Nigeria.
[6] Includes properties in Burma, Indonesia, Malaysia, and Thailand.
[7] Includes properties in Argentina, Australia, Japan, Peru, and Zimbabwe.
[8] Includes properties in Brazil, Malaysia, and Namibia.

are slightly higher than onshore tin dredging costs. Mining and beneficiation costs average about $0.80/mt ore, about $0.20/mt less than gravel pump operations (table 5).

Indonesian deposits contain over 320,000 mt of recoverable tin metal; this represents the largest reserves among all the MEC dredge operations evaluated. Indonesia's P.T. Tambang Timah is the largest integrated tin producer within the MEC's. The total production cost (0-pct DCFROR) for dredges in Indonesia is about $4.80/lb refined tin.

The average dredge operation can produce tin metal at a total production cost (0-pct DCFROR) of about $4.50/lb. Of the 40 producing dredge operations evaluated in this study, Malaysia has the lowest weighted-average total production cost, at $3.70/lb tin.

Underground

Among the 39 producing MEC underground deposits evaluated, the weighted-average mining and beneficiation cost is about $34.20/mt ore; however, the average tin grade from the underground mine is much higher than that from all other producing tin deposits and yields more tin per metric ton of ore mined. This results in a total production cost (0-pct DCFROR) of $4.80/lb refined tin.

Because of penalties imposed at the smelter on concentrates from lode deposits (e.g., Bolivian "underground" operations), smelter-refinery costs for underground operations are higher than for surface operations where concentrates are produced from relatively pure, alluvial deposits. The weighted-average smelter-refinery cost for producing underground MEC deposits is about $1.10/lb tin, compared with $0.10/lb and $0.20/lb tin, respectively, for concentrates from gravel pump and from dredge operations.

The average total production cost for all underground operations is weighted heavily by Bolivian production costs. At $7.00/lb tin (0-pct DCFROR), Bolivia's total production cost appears slightly higher than underground operations; as explained in the "Availability" section, Bolivia's total production cost is underestimated because of the tremendous devaluation of Bolivian currency since the cost data were collected.

Open Pit

The eight producing open pit mines evaluated in this study account for less than 1 pct of the recoverable tin in MEC's with an annual production of over 400,000 mt ore yielding over 4,500 mt/yr tin. The major portion of the $4.90/lb total production cost (0-pct DCFROR) is in the mining and beneficiating.

The largest producing open pit mine evaluated in this study is the Uis tin mine, Namibia. The burden of the total production cost in this mine is in mining and beneficiation. With one of the longest projected production lives of any of the deposits evaluated in this study and an estimated total production cost of less than $4/lb tin metal (0-pct DCFROR), Uis should continue to be a major open pit tin producer.

SUMMARY

Total production costs (0-pct DCFROR) for producing tin mines average about $5/lb tin. These costs are weighted heavily by operations in countries such as Malaysia, where large volumes of tin are produced from relatively low-cost gravel pump and dredge operations; Malaysia accounts for almost 40 pct of the estimated recoverable tin metal and 30 pct of the annual tin production among evaluated tin-producing countries.

The largest portion of total production costs is in mining and beneficiation. The most significant factor affecting mining and beneficiation costs, on a dollar per pound of refined tin basis, is the feed grade.

Because Brazil's resources are underestimated (it has expanded production since the data were gathered), relatively low-cost, high-grade alluvial operations should allow Brazil to enhance its role in the tin industry. Undelineated resources and additional resource data not available for this study, plus production increases of its low-cost tin operations, will maintain Brazil's status as one of the world's major tin producers.

Table 6.—Tin production costs, by mine method, for selected MEC's[1]

(All costs are in January 1985 U.S. dollars per pound of refined tin on a weighted-average basis)

Mining method and country	Operating cost			Byproduct credit[3]	Net operating cost	Recovery of capital[4]	0-pct DCFROR		15-pct DCFROR		
	Mine	Mill	Other[2]				Taxes and royalties[5]	Total cost[6]	Taxes and royalties[7]	Return on investment[8]	Total cost[9]
Gravel pump:											
Brazil	2.40	([10])	0.20	Unk	2.60	0.50	0.70	3.80	0.90	0.30	4.30
Indonesia	4.00	([10])	.20	Unk	4.20	.30	.10	4.70	.40	.30	5.30
Malaysia	4.90	([10])	.10	Unk	5.00	.30	.10	5.40	.40	.20	5.90
Thailand	4.40	([10])	.10	Unk	4.50	.10	1.80	6.40	2.00	.20	6.80
Others[11]	2.60	([10])	.20	Unk	2.80	.80	.30	3.90	.60	.80	5.00
Average	4.60	([10])	.10	Unk	4.70	.30	.30	5.30	.50	.30	5.80
Dredge:											
Indonesia	3.60	([10])	.20	Unk	3.80	.80	.20	4.80	.60	.50	5.70
Malaysia	3.10	([10])	.10	Unk	3.20	.00	.50	3.70	.50	.10	3.80
Thailand	2.50	([10])	.20	Unk	2.70	.30	1.40	4.40	1.80	.30	5.10
Others[12]	3.80	([10])	.20	Unk	4.30	.50	.30	4.80	.70	1.70	6.90
Average	3.30	([10])	.20	Unk	3.50	.50	.50	4.50	.80	.50	5.30
Underground:											
Bolivia	2.90	1.10	1.50	1.10	4.40	.40	2.20	7.00	2.50	.30	7.60
South Africa, Rep. of	1.50	1.30	.10	0	2.90	.30	.00	3.20	.30	.40	3.90
United Kingdom	2.30	1.10	.90	0	4.30	.00	.20	4.50	.70	.40	5.40
Southeast Asia[13]	1.10	.40	.30	0	1.80	1.20	.20	3.20	.50	2.10	5.60
Others[14]	1.90	1.10	1.30	1.30	3.00	.90	.20	4.10	.50	.30	4.70
Average	2.00	1.00	1.10	.70	3.40	.80	.60	4.80	.90	.40	5.50
Open pit:											
Australia	10.20	5.50	.50	7.50	8.70	1.00	.10	9.80	1.40	1.70	12.90
Thailand	2.10	.80	.20	.20	2.90	.50	1.30	4.70	1.50	.10	5.00
Others[15]	1.40	1.00	.10	0	.50	.00	.50	3.00	.10	.10	3.20
Average	3.70	2.10	.20	1.90	4.10	.70	.10	4.90	.50	.50	5.80

Unk Unknown.
[1] Based on 1982 data; costs updated to constant January 1985 U.S. dollars.
[2] Includes smelting, refining, and transportation of concentrates to the smelter and refinery.
[3] Because of insufficient byproduct data, byproduct credits from placer deposits were not incorporated into the cost calculations.
[4] Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure as of January 1985, and investments required over the life of the operation.
[5] Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 0-pct DCFROR.
[6] Equal to sum of net operating costs, taxation, and capital recovery determined at a 0-pct DCFROR.
[7] Includes property, State, Federal, and severance taxes and royalties, where applicable, calculated at a 15-pct DCFROR.
[8] Revenue increase per pound necessary to obtain a 15-pct DCFROR.
[9] Equal to sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery, *plus* the per pound increase necessary to provide a 15-pct DCFROR after taxation.
[10] Combined in with mining costs.
[11] Includes properties in Australia, Bolivia, Burma, and Zaire.
[12] Includes properties in Australia, Bolivia, Brazil, and Nigeria.
[13] Includes properties in Burma, Indonesia, Malaysia, and Thailand.
[14] Includes properties in Argentina, Australia, Japan, Peru, and Zimbabwe.
[15] Includes properties in Brazil, Malaysia, and Namibia.

AVAILABILITY

TOTAL RECOVERABLE

Total availability curves for all MEC deposits evaluated and individual curves for Indonesia, Malaysia, and Thailand, at a 0- and 15-pct DCFROR's, are illustrated in figure 4. Over 2.2 million mt tin is estimated to be recoverable at total production costs ranging up to $17.50/lb (0-pct DCFROR). Almost 88 pct of the demonstrated tin resource is from producing mines.

On the 0-pct DCFROR curve, approximately 290,000 mt tin is potentially recoverable at costs ranging up to $3/lb, 628,000 mt at costs ranging up to $4/lb, 1.3 million mt at costs up to $5/lb, and 2.3 million mt at costs up to $7/lb. Approximately 50 pct of the total recoverable tin is available at costs under $5/lb at a 0-pct DCFROR. The estimated production costs for mines operating at less than full capacity in countries under ITC production quotas would be slightly higher if they operated at full capacity. By reducing capacities, these mines were able to decrease labor and associated costs (by eliminating a shift or halting a recovery-beneficiation circuit) and therefore decrease the total cost of production.

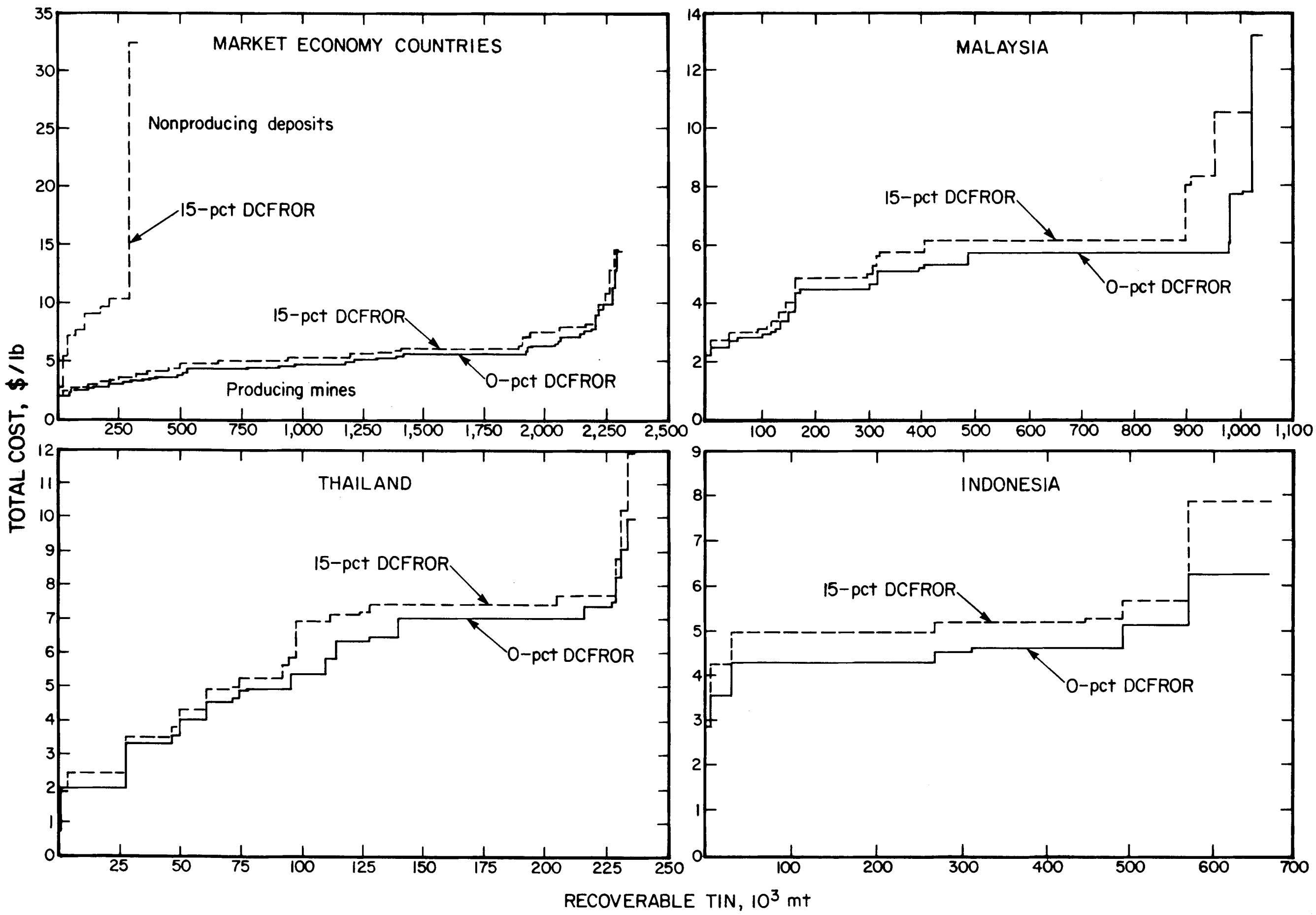

Figure 4.—Potential total tin available from selected MEC's (January 1985 U.S. dollars).

The curve for Malaysia shows 1.0 million mt of potentially recoverable tin from 36 deposits at costs ranging from \$1.60/lb to \$12.90/lb at a 0-pct DCFROR. At a 0-pct DCFROR, about 127,000 mt of tin is potentially available at total production costs under \$3/lb, 165,000 mt at costs up to \$4/lb, 317,000 mt at costs under \$5/lb, and almost 980,000 mt at costs up to \$7/lb.

The curve for Indonesia shows 660,000 mt of tin potentially recoverable from seven mines (or regions) at costs ranging from \$2.80/lb to \$6.30/lb at a 0-pct DCFROR. Over 490,000 mt of this tin is estimated to have a total production cost under \$5/lb at a 0-pct DCFROR.

ANNUAL CAPACITY

An annual availability curve (fig. 5) was constructed for producing mines in MEC's. The annual curve graphically illustrates production potential at different cost levels. It is not intended as a projection of actual production during the years shown, but does illustrate annual production potential at estimated capacity rates.

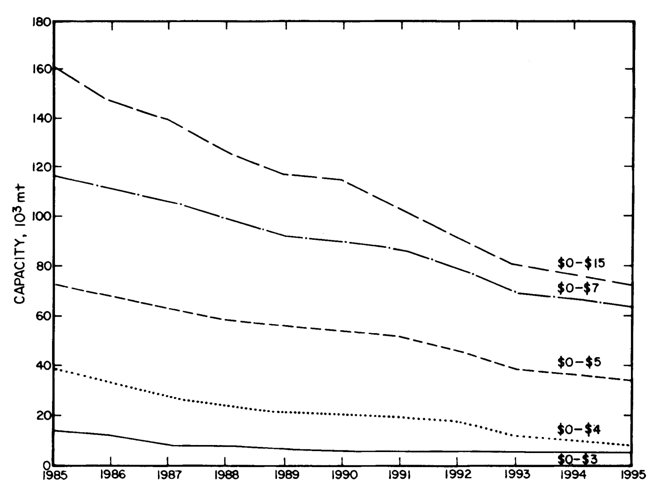

Figure 5.—Potential annual tin production from producing mines in MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

Table 7 indicates 1985 tin production potential (at capacity levels) from producing mines. Production costs were calculated at the 0-pct DCFROR level. As shown, a total of nearly 148,000 mt is potentially available, mainly from Malaysia, Thailand, and Indonesia. Implicit in these data are the assumptions that all operations will operate at full capacity and they will be able to sell all primary and secondary products generated.

Table 7.—Potential 1985 refined tin capacities from producing mines at selected production cost levels, including 0-pct DCFROR, metric tons

Price range . . $/lb . .	0-3.00	0-3.50	0-3.75	0-4.00	0-4.50	>4.50[1]
Australia	0	0	5,710	5,710	5,710	8,470
Bolivia	300	1,470	1,470	5,470	5,470	14,050
Brazil	2,630	2,930	2,930	4,020	[2]4,370	7,570
Indonesia	3,740	3,740	5,980	5,980	12,270	26,530
Malaysia	10,270	11,520	13,330	13,850	23,270	48,300
Thailand	1,440	2,050	2,310	3,130	3,130	28,000
United Kingdom	0	1,380	1,380	1,380	1,380	5,470
Others[3]	5,290	5,290	5,290	5,290	5,290	[4]9,220
Total	23,670	28,380	38,400	43,450	59,970	147,680

[1] Includes capacity production from all producing operations evaluated. In January 1984 U.S. dollars, this range extends up to $19/lb tin metal.

[2] Lack of data precludes Brazil's increased production from being included. Estimated capacity is approximately 22,000 mt refined metal, or 14,000 mt greater than Brazilian capacity noted in this table. The estimated weighted-average production cost for all Brazilian operations (0-pct DCFROR) is $4.20/lb tin metal.

[3] Includes capacity production from Argentina, Burma, Namibia, Nigeria, Peru, and the United Kingdom.

[4] Includes capacity production from Argentina, Burma, Namibia, Nigeria, Peru, the United Kingdom, as well as Japan, the Republic of South Africa, Zaire, and Zimbabwe.

The most significant development since data were gathered in 1982 that is not reflected in table 7 is the increased tin production from Brazil. Estimated Brazilian tin production was 22,000 mt tin metal in 1985, a significant increase from the 1982 estimated production of 7,600 mt. The weighted-average production cost (0-pct DCFROR) for all Brazilian tin is estimated to be $4.20/lb.

Potential annual production of tin from producing mines (and regions) in MEC's at a 0-pct DCFROR from 1985 to 1995 is shown in figure 5. The curves reflect the production capacity of existing mines, including planned expansions when known. The curves could not take into account ITC sales quotas, production cutbacks mandated by market conditions, or smuggled tin, because these factors are likely to vary on an annual basis and would be difficult to project. Because actual production was below capacity levels at the time of the study (at least on a countrywide basis among ITC members), it is not likely that potential annual production will decline to the extent shown in the curves, since much production potential was being deferred. Furthermore, these curves are based on a static resource estimate and do not reflect the fact that mineral resource estimates historically have increased over time or remained relatively constant owing to ongoing exploration programs in existing mines and the discovery of new ore bodies. This is particularly important in countries such as Bolivia, where underground mines tend to maintain reserve estimates that often exceed no more than 3 to 5 yr of production at current mining levels. This is partially due to the nature and complexity of the ore bodies, and partially due to the added cost of blocking out reserves much further ahead of current production.

Figure 5 shows a gradual decline in production capacity among producing mines, from 161 million mt in 1985 to about 75 million mt in 1995, as the demonstrated resources of a number of mines become exhausted. As mentioned previously, such a decline is unlikely for some producers, but there is concern among low-cost producers, such as Malaysia, that new capacity from existing mines will be more expensive to develop than currently available capacity because ore grades at dredging operations have been declining. In order to expand future tin production, the Malaysians will likely have to resort to even more costly underground lode mining in mountainous areas of Malaysia that are currently under exploration.

CONCLUSIONS

The 140 deposits evaluated for this study represent a demonstrated in situ tonnage of 21.8 billion mt of ore containing 2.6 million mt of potentially recoverable tin metal. According to resource estimates of the countries studied, Malaysia, Indonesia, and Thailand have the largest demonstrated tin resources, accounting for over 70 pct of the total recoverable tin from the deposits evaluted in MEC's. The delineation of new reserves and production increases has established Brazil as one of the largest tin producers in the world.

Almost half of the potentially recoverable MEC tin is accounted for in gravel pump operations, followed by dredge, underground, and open pit operations. Dredges have the lowest estimated weighted-average total production costs per pound of refined tin, followed by underground mines, gravel pumps, and open pit mines.

Increased production from the very low cost tin operations in Brazil, plus their nonparticipation in the ITA, has established Brazil as a dominant MEC tin producer.

REFERENCES

1. Tin International. Statistical Update. V. 58, No. 12, 1985, pp. 451-464.

2. Bleiwas, D.I., A.E. Sabin, and G.R. Peterson. Tin Availability—Market Economy Countries. A Minerals Availability Appraisal. BuMines IC 9086, 1986, 50 pp.

3. Carlin, J.F., Jr. Tin. Ch. in Mineral Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 847-858.

4. ______. Tin. Ch. in BuMines Minerals Yearbook 1985, v. 1, pp. 963-974.

5. ______. Tin. Sec. in BuMines Minerals Commodity Summaries 1986, pp. 166-167.

6. Cunningham, L.D. Tantalum. Sec. in BuMines Minerals Commodity Summaries 1986, pp. 158-159.

7. Sainsbury, C.L., and B.L. Reed. Tin. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 637-651.

8. Parker, R.L., and J.W. Adams. Niobium (Columbium) and Tantalum. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 443-454.

TITANIUM, HAFNIUM, AND ZIRCONIUM

INTRODUCTION

For many years titanium has primarily been used as a source for pigments in the form of titanium dioxide (TiO_2). Its whiteness, high refractive index, and light-scattering ability make it an excellent whitening agent for paints, paper, rubber, plastics, and other miscellaneous items. More recently, though, titanium metal has become important in the defense and aerospace industries because of its high strength to weight ratio and its resistance to corrosion, although this accounts for only 5 pct of world TiO_2 consumption. Small quantities of TiO_2, in the form of rutile, are also used for welding rod coating. Ceramic capacitors, for electronics, use small quantities of titanium. The United States is presently one of the world's largest producers of TiO_2 pigments and metal, although it imports significant quantities of the raw materials (TiO_2 minerals) for their manufacture.

Titanium is the ninth most abundant element in the Earth's crust, and occurs in many mineral species. Only a few of these minerals contain enough titanium to be of commercial importance. They are ilmenite ($FeTiO_3$), rutile (TiO_2), anatase (TiO_2), and leucoxene (an upgraded alteration product of ilmenite).

The majority of the world's rutile, typically about 95 pct TiO_2, is produced in Australia, the Republic of South Africa, Sri Lanka, and Sierra Leone. Rutile can be used in titanium metal plants, in chloride process pigment plants, or for the manufacture of welding rods.

Ilmenite and leucoxene are much more abundant than rutile. Ilmenite may contain as much as 50 to 70 pct TiO_2, leucoxene as much as 87 pct TiO_2 (and even 91 pct from some Australian operations). Ilmenite is generally used in sulfate-process pigment plants, although one producer uses the higher grade ilmenites in chloride-process plants. Ilmenite is also upgraded to synthetic rutile for use in chloride plants, or is smelted to produce a titanium slag containing 80 to 85 pct TiO_2, plus a byproduct pig iron concentrate. Titanium slag can feed sulfate plants and, if high enough grade, chloride plants.

Titanium deposits often contain other minerals that may be recovered with the titanium and improve the economics of the operation. These minerals include zircon, monazite, garnet, hafnium, sillimanite, and kyanite.

This chapter analyzes the availability of each titanium mineral separately. The analysis goes to the concentrate form of each mineral and does not include the processing of the concentrates into a pigment or metal. Analyses are presented in terms of metric tons of concentrate. Included are discussions of the availability of zirconium and hafnium.

Most of the information presented in this chapter is an updated summary of Bureau of Mines Information Circular 9061 "Titanium Minerals Availability—Market Economy Countries. A Minerals Availability Appraisal" (*1*).[1] Additional information on the titanium industry worldwide is available from other Bureau publications (*2-7*).

GEOLOGY AND RESOURCES

GEOLOGY

Ilmenite, theoretically 53 pct TiO_2 and 47 pct FeO, can be found in numerous mineral occurrences. Its titanium content may often be greater than the theoretical amount because of oxidation and removal of iron and calcium, as frequently occurs in sand deposits with titanium contents of as much as 65 to 70 pct TiO_2. This form is known as altered ilmenite (or leucoxene, arizonite, or pseudorutile). Rutile is also found in numerous mineral occurrences; it is theoretically 100 pct TiO_2, although it seldom contains greater than 95 pct because of impurities. Other titanium minerals occasionally found in economic concentrations are anatase, brookite, and perovskite.

The economic titanium minerals ilmenite, rutile, anatase, and potentially economic perovskite occur in a variety of different deposit types. These include several types in hard rocks (igneous gabbro-anorthosite assemblages, alkalic igneous rocks, unusual metamorphic rocks), weathered and hydrothermally altered rocks, and placer deposits.

Gabbro anorthosite assemblages, typically of Precambrian age, commonly contain ilmenite (locally with rutile) disseminated and/or as massive segregations. Intergrowths with magnetite are a major problem. Major deposits are in Norway, Canada, and the United States.

Alkalic igneous rocks (of any age) may contain rutile, anatase, or perovskite, all commonly with chemical impurities. Alteration by weathering may produce a higher quality ore, as in Brazil.

Metamorphic rocks of eclogite facies contain rutile and are important resources in Italy and the U.S.S.R. Aluminous metamorphic rocks and hydrothermally altered rocks contain large low-grade resources of rutile.

In the United States, rock-hosted deposit types previously mentioned have collectively been estimated (*8*) to contain 67 million mt of TiO_2 by the U.S. Geological Survey and Bureau titanium specialists (their estimates include

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

low-grade resources and a larger number of deposits than this study).

Placer deposits of titanium minerals include shoreline-complex sands of modern and former shorelines, and fluvial placers. Shoreline-complex sands contain the more important resources and include beach deposits, aeolian (dune) deposits, and other related sand deposits. Any titanium mineral resistant to abrasion and weathering, such as rutile and ilmenite, or formed by weathering, such as altered ilmenite, may be concentrated along with other resistant heavy minerals such as zircon and monazite, in the manner that is commonly observed on many beaches. These deposits may be of any age but most resources are present in Tertiary to modern deposits with relic depositional topography. Important deposits are in Australia, the Republic of South Africa, and the United States. Placer deposits in the United States have been estimated (*8*) to contain 49 million mt of TiO_2.

RESOURCES

Estimates of demonstrated titanium resources for the market economy country (MEC) deposits used in this study are 445 million mt of contained TiO_2 (table 1 and figure 1). Although Australia has the largest share of demonstrated ore resources (42 pct of the world total), these low-grade beach sand type deposits account for only a small share (11 pct) of the total contained TiO_2 in the demonstrated resources. Nevertheless, Australia's mines and deposits account for a large share of current world production of titanium concentrates (approximately 40 pct for ilmenite and 50 pct for rutile). The world titanium reserve base is estimated to be 455 million mt of contained TiO_2 (converted from 300 million st contained titanium) (*3*). Of this total, approximately 60 million mt contained TiO_2 from Egypt, China, Malaysia, and the U.S.S.R., were not evaluated in this study because of a lack of cost data.

Regionally, titanium resources are widespread, with North America and Europe accounting for over 50 pct of the contained TiO_2 in the deposits included in this study. Table 1 also shows the large potential of anatase resources located in Brazil.

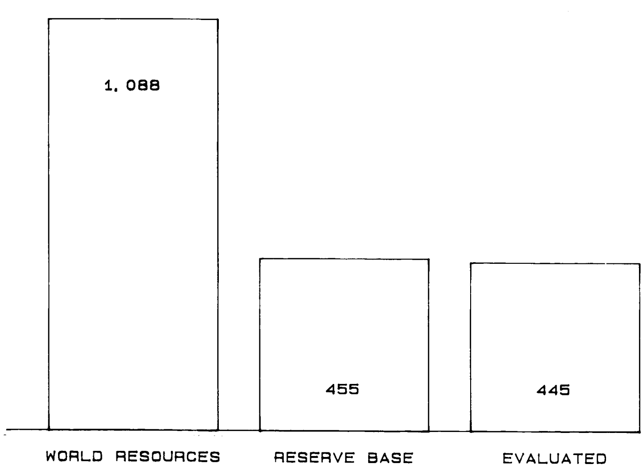

Figure 1.—Estimates of world titanium resources (million metric tons of TiO_2).

Table 1.—Summary of MEC titanium resources evaluated for this study, as of January 1985[1]

	Ore-sand, 10^6 mt		Contained TiO_2, 10^6 mt		Av TiO_2 grade,[3] wt pct
	Demonstrated	Inferred	Demonstrated	Inferred[2]	
NORTH AMERICA					
United States:					
Rutile			1.4	Unk	0.24
Ilmenite	1,705	223	39.5	8.9	2.33
Leucoxene			1.0	Unk	.15
Canada: Ilmenite	239	114	72.8	8.8	30.46
Total or av	1,944	337	114.7	17.7	5.90
SOUTH AMERICA					
Brazil:					
Anatase			84.5	36.8	18.12
Rutile	605	669	.1	.5	.11
Ilmenite			17.7	19.4	12.78
Total or av	605	669	102.3	56.7	16.91
EUROPE					
Finland: Ilmenite	9	19	1.2	2.6	13.50
Italy:					
Rutile			20.7	17.5	4.37
Ilmenite	473	400	8.5	7.2	1.79
Norway: Ilmenite	492	Unk	88.5	Unk	18.00
Total or av	974	419	118.9	27.3	12.21
ASIA					
India:					
Rutile			4.3	2.4	.89
Ilmenite	483	455	30.0	41.6	6.21
Leucoxene			.6	Unk	.27
Sri Lanka:					
Rutile	65	Unk	.8	Unk	1.17
Ilmenite			3.7	Unk	5.69
Total or av	548	455	39.4	44.0	7.19
AFRICA					
Sierra Leone: Rutile	120	17	1.9	.3	1.61
South Africa, Rep. of:					
Rutile	655	1,538	2.2	5.1	.33
Ilmenite			14.9	35.1	2.28
Total or av	775	1,555	19.0	40.5	2.45
OCEANIA					
Australia (East Coast):					
Rutile	3,048	1,461	6.9	4.7	.23
Ilmenite			11.7	4.4	.38
Australia (West Coast):					
Rutile			2.8	2.8	.52
Ilmenite	578	813	23.4	70.9	4.05
Leucoxene			2.0	1.0	.37
New Zealand:					
Rutile	73	924	.1	.9	.09
Ilmenite			3.4	42.5	4.60
Total or av	3,699	3,198	50.3	127.2	1.36
Grand total or av	8,545	6,633	444.6	313.4	5.17
Total by commodity:					
Rutile	Unk	Unk	41.2	34.2	Unk
Ilmenite	Unk	Unk	315.3	241.4	Unk
Leucoxene	Unk	Unk	3.6	1.0	Unk
Anatase	Unk	Unk	84.5	36.8	Unk

Unk Unknown.
[1] In situ resources.
[2] Inferred contained TiO_2 tonnage not necessarily based on the listed demonstrated grade and therefore cannot be calculated from the average TiO_2 grade.
[3] Represents demonstrated level resource grade. The total average TiO_2 grade for each region is a weighted average of all titanium minerals from that region. Not all deposits in each region contain each titanium mineral; therefore, calculation of contained TiO_2 from the demonstrated tonnage and the average grade is not possible for all regions.

From the 61 deposits studied (table 2), additional resources of at least 313 million mt of contained TiO_2 is estimated to exist at the inferred level. The majority of this is located in Australia and New Zealand (41 pct), Brazil (18 pct), India (14 pct), and the Republic of South Africa (13 pct). The remainder is found in the United States, Canada, Finland, Italy, and Sierra Leone. The total inferred resources represent a 70-pct addition to the demonstrated amount.

Table 2.—MEC titanium properties included in this study

Country and properties	Ownership	Current Status[1]	Deposit type[2]	Mining method	Milling method	Products[3]
Australia						
New South Wales:						
Bridge Hill Ridge	Mineral Deposits Ltd	P	PL	Dredging	Magnetic-electrostatic	R, I, Z
Evans Head	State of New South Wales	EXP	PL	..do	..do	R, I, Z
Munmorah	Associated Minerals Consol. Ltd.	PP	PL	..do	..do	R, I, Z, RE
Stockton Bight	Mineral Deposits Ltd	EXP	PL	..do	..do	R, I, Z
Tomago Sand Pits	R Z Mines (Newcastle) Ltd	P	PL	..do	..do	R, I, Z
Yuraygir National Park	State of New South Wales and Federal Government.	PP	PL	..do	..do	R, I, Z
Queensland:						
Agnes Waters						
Cooloola	Mineral Deposits Ltd	EXP	PL	..do	..do	R, I, Z
	State of Queensland and	PP	PL	..do	..do	R, I, Z, RE
Curtis Island	Federal Government.					
Fraser Island	Murphyores Holdings Ltd	EXP	PL	..do	..do	I, R, Fe, Z
Gladstone Mainland	Murphyores Dillingham	PP	PL	..do	..do	R, I, Z, RE
Moreton Island (MDL)	Murphyores Holdings Ltd	EXP	PL	..do	..do	I, R, Fe, Z
Moreton Island	Mineral Deposits Ltd	EXP	PL	..do	..do	R, I, Z
(Murphyores).	Murphyores Holdings Ltd	EXP	PL	..do	..do	R, I, Z
North Stradbroke Island	Consolidated Rutile Ltd	P	PL	..do	..do	R, I, Z, RE
Western Australia:						
Allied Eneabba	E.I. du Pont de Nemours and others.	P	PL	Strip level	..do	R, I, L, Z, RE
Australind	Associated Minerals Consol. Ltd.	EXP	PL	Ground sluicing	..do	I, R, L, Z
Barrambie	Ferrovanadium Corp	EXP	HR	Open pit	Magnetic[4]	S, Fe
Cable Sands	Kathleen Investments	P	PL	Dredging	Magnetic-electrostatic	I, R, Z, RE
Capel	Associated Minerals Consol. Ltd.	P	PL	..do	..do.[5]	I, R, L, SR, RE
Cataby	Metals Exploration-Alliance Minerals.	EXP	PL	..do	..do	R, I, Z, RE
Eneabba	Associated Minerals Consol. Ltd.	P	PL	..do	..do.[5]	I, R, L, SR, RE
Gingin	Westralian Sands-Lennald Oil	EXP	PL	..do	..do	R, I, L, Z
Jurien Bay, Cooljarloo.	TiO_2 Corporation NL	DEV	PL	..do	..do	I, R, L, Z, RE
North Capel	Westralian, Tioxide, Ishirara	P	PL	..do	..do.[6]	I, R, L, Z, RE
Scott River	State of Western Australia	EXP	PL	..do	..do	I, R, L, Z
Yoganup Extended, Boyanup, Tutunup.	Westralian, Tioxide, Ishirara	P	PL	Open pit	..do.[6]	I, R, L, Z, RE
Brazil:						
Bananeira	Mineracao Itaqui	EXP	HR	Open pit	Flotation	A
Camaratuba	Titanio do Brazil	P	PL	Strip level	Magnetic-electrostatic	R, I, Z
Campo Alegre de Lourdes	Cia. Bahiana de Pesquisa Metais	EXP	HR	Open pit	..do.[4]	S, Fe
Catalao	Metais de Goiasfertil	DEV	HR	..do	Flotation	A
Tapira	Cia. Vale do Rio Doce	DEV	HR	Strip level	..do	A
Canada:						
Allard Lake	QIT-Fer Et Titane (SOHIO)	P	HR	Open pit	Magnetic-electrostatic[4]	S, Fe
Pin-Rouge Lake	Laurentian Titanium Mines	EXP	HR	..do	..do.[4]	S, Fe
Finland: Otanmaki	Rautaruukki OY	P	HR	Sublevel stoping	Flotation	I, Fe, O
India:						
Chavara (IREL)	India Rare Earths, Ltd	P	PL	Strip level	Magnetic-electrostatic	R, I, L, Z, RE
Chavara (KMML)	Kerala Minerals and Metals, Ltd	P	PL	Dredging	..do	R, I, L, Z, RE
Manavalakurichi	India Rare Earths, Ltd	P	PL	Strip level	..do.[5]	I, R, SR, Z, RE
Orissa-Chatrapur	..do	P	PL	Dredging	..do.[5]	I, R, Z, RE, SR
Italy: Piampaludo	Mineraria Italiana Spa Milan	EXP	HR	Strip hillside	Flotation	R, I, G
New Zealand: Barrytown	Fletcher-Challenge Ltd	EXP	PL	Dredging	Magnetic-Electrostatic[5]	I, R, SR, Z, RE
Norway: Tellnes and Tyssedal	Titania (NL Industries) and KSI (DNN Industries).	P	HR	Open pit	Gravity-flotation[4]	I, S, Fe, O
Sierra Leone: Mogbwemo	Sierra Rutile (NORD Resources)	P	PL	Dredging	Magnetic-electrostatic	R
South Africa, Republic of:						
Richard's Bay	QIT, Union Corp, IDL, others	P	PL	..do	..do.[4]	S, R, Fe, Z
Sri Lanka: Pulmoddai	Ceylon Mineral Sands Corp. (Government).	P	PL	..do	..do	I, R, Z, RE
United States:						
Arkansas: Magnet Cove	Numerous private owners	PP	HR	Open pit	..do	R
California: Ione	North American Refractories	([7])	PL	Dredging	Magnetic	I, Z
Colorado: Powderhorn	Buttes Gas and Oil	EXP	HR	Open pit	Magnetic-electrostatic	P, RE
Florida						
Green Cove Springs	Associated Minerals Ltd	P	PL	Dredging	..do	R, I, L, Z, RE
Highland Operation	E.I. du Pont de Nemours	P	PL	..do	..do	M, Z
Trail Ridge Operation	..do	P	PL	..do	..do	M, Z
Georgia:						
Brunswick-Altamaha	Union Camp Corporation	EXP	PL	..do	..do	M, R, Z, RE
Cumberland Island	U.S. National Park Service	EXP	PL	..do	..do	M, Z
New Jersey: Manchester	ASARCO	PP	PL	..do	..do	I
New York: MacIntyre Dev	N L Industries	([8])	HR	Open pit	Flotation	I, Fe

See footnotes at end of table.

Table 2.—MEC titanium properties included in this study—Continued

Country and properties	Ownership	Current Status[1]	Deposit type[2]	Mining method	Milling method	Products[3]
United States—Con.						
North Carolina:						
N L Industries	N L Industries	EXP	PL	Open pit	Magnetic-electrostatic	I
Oklahoma:						
Otter Creek Valley	Numerous private owners	EXP	PL	Placer mining	Magnetic	I
Tennessee:						
Silica Mine	Tennessee Silica Sand	(7)	PL	Open pit	Magnetic-electrostatic	I, R, L, Z, O, RE
Oak Grove	Ethyl Corporation	EXP	PL	Dredging	..do	I, R, Z, RE
Virginia:						
B.F. Camden Anomaly	Private ownership	PP	HR	Open pit	..do	I
Piney River	S.U. Wilkcens, Jr.	EXP	HR	Open pit—sublevel caving.	Flotation	I
Wyoming: Iron Mountain	Rocky Mountain Energy, Anaconda	EXP	HR	Open pit	Gravity-magnetic[4]	S, Fe

[1] PRD, producer; PP, past producer; DEV, developing deposit; EXP, explored prospect.
[2] PL, placer (or sand); HR, hard rock.
[3] The first product listed was assumed to be the primary product for this study. R, rutile concentrate; Fe, iron, magnetite, or pig iron; I, ilmenite concentrate; L, leucoxene concentrate; S, titanium slag; SR, synthetic rutile concentrate; A, anatase concentrate; RE, rare-earth oxide concentrate (monazite); G, garnet, O, other miscellaneous (sulfides, precious metals, vanadium, pyrite, etc.).
[4] After beneficiation, the ilmenite concentrate is smelted in an electric furnace.
[5] After beneficiation, some or all of the ilmenite concentrate is further upgraded in a synthetic rutile plant.
[6] Will be producing synthetic rutile also, starting 1987.
[7] Producers of silica sand, but not heavy minerals. Assumed as explored prospect for this study.
[8] Presently only producing a magnetite concentrate.

U.S. AND WORLD HISTORICAL PRODUCTION

Figure 2 illustrates the production of titanium concentrates worldwide in 5-yr intervals for the years 1950 through 1985 (estimated). The illustration shows the dramatic increase of production levels over the past 35 yr for all titanium concentrates.

Figure 2 shows that over the past 35 yr, Australia, Norway, and the United States have accounted for the majority of all world ilmenite and leucoxene production, ranging from a low of just over 60 pct in the late 1950's, and early 1960's, to a high of over 80 pct in the late 1960's and early 1970's. Presently these three countries account for an estimated 64 pct of world ilmenite and leucoxene production. Over the 35-yr period illustrated, it is of importance to note that while ilmenite and leucoxene production has consistently increased in Australia and Norway, U.S. production increased to a peak in the mid-1960's and has been declining ever since. Also, it should be noted that the "other" category primarily includes India, which was the second largest producer in 1950 and 1955, behind the United States.

Figure 2 also illustrates the production history for rutile over the past 35 yr (note that a small quantity of production from the United States is not included because in recent years it has been unreported by the Bureau for confidentiality purposes). It is apparent that Australia dominated world rutile production until the late 1970's and early 1980's, when production commenced in Sierra Leone and the Republic of South Africa. Presently Australia is estimated to account for just over one-half of world rutile production, with Sierra Leone and the Republic of South Africa making up most of the remainder.

Until the late 1970's and early 1980's, all titanium slag production was from Canada (fig. 2). Presently, Canada produces nearly two-thirds of all slag and the Republic of South Africa produces the remainder. The Republic of South Africa is expected to increase its share of world slag production in the near future.

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

Titanium ore can be mined by both surface and underground methods, although surface mining, principally of sand deposits, is most commonly used. Of the 61 deposits in this study, all but two used or were proposed to use one of three surface mining methods: dredging, strip level, or open pit. Table 2 lists various data for the titanium mines and deposits included in this study.

Dredging is the typical surface mining method for placer beach sand deposits. This method is used for deposits in Australia (east coast), Sierra Leone, the Republic of South Africa, and the United Sates. Types of dredges most often used are cutterhead suction dredges and, in some cases, bucket-ladder dredges. Preliminary concentration may take place on the dredge or on barges alongside, using gravity separation devices. A typical wet mill with gravity circuits may have gravity separation devices carrying out rougher,

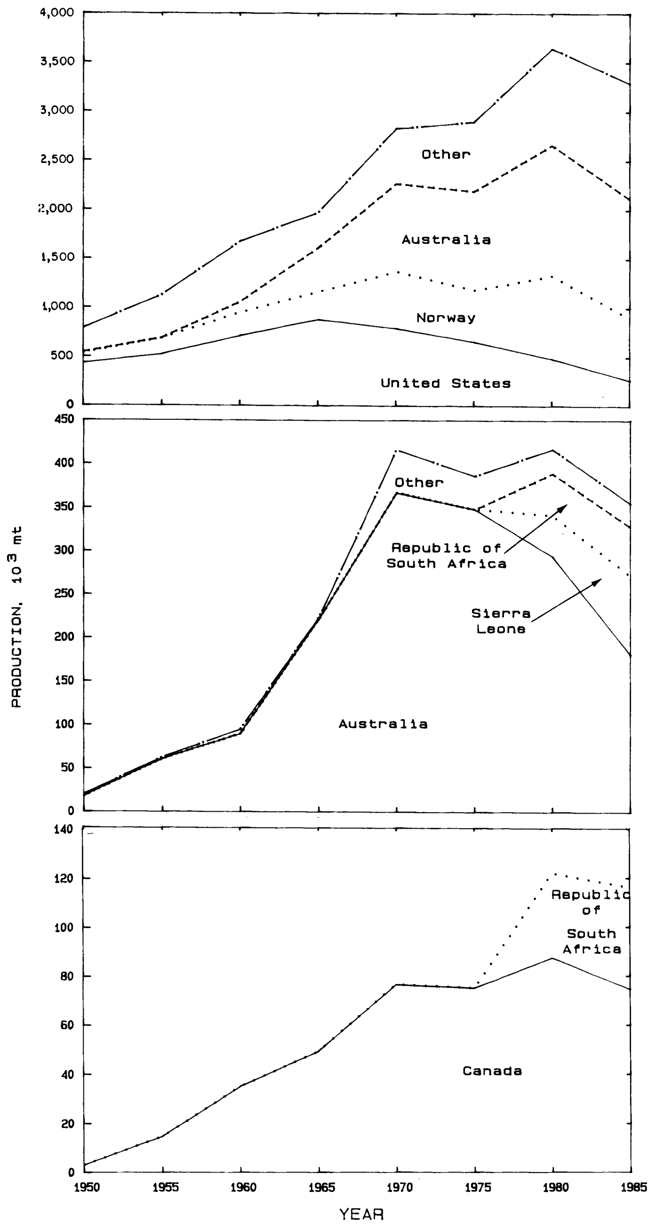

Figure 2.—World production of ilmenite and leucoxene concentrates (top), rutile concentrate (middle), and titanium slag (bottom), 1950-85,

cleaner, and recleaner duties, with banks of spirals upgrading recleaner concentrate.

Factors such as ore body location (usually several kilometers inland), ore body size and shape, lack of an adequate water supply, and deposit lithology, may make the use of dredges for some placer deposits impractical. In those instances, other surface mining methods, including draglines and/or front-end loaders (FEL's) and trucks, are used. These methods are typical of those used in mines in West Australia and Sri Lanka.

Open pit or underground methods are used on all hard-rock titanium deposits. Finland's Otanmaki Mine, using a sublevel stoping method, is the only underground mine producing titanium in this study. Open pit methods that require little or no blasting and use FEL's and trucks for ore and waste haulage, are proposed for deposits in Brazil and the United States. Hard-rock deposits in Canada, Italy, Norway, and the United States require, or would require, extensive blasting. Diesel or electric shovels and trucks or FEL's and trucks are proposed for ore and waste haulage.

The only exception to the preceding mine descriptions are India's coastal beach sand deposits, where natural concentrate beach sand is skimmed by hand using shovels, and buckets, baskets, or handcarts are used to transport the ore to stockpiles. The ore is then taken by conveyor, hand-pushed mine car, or canoe directly to a dry mill.

PROCESSING

Titanium ores mined from placer beach sand deposits are processed through preliminary concentration in wet mills, with final concentration taking place in dry mills. In wet mills ore is separated, using wet-gravity methods, into a heavy mineral fraction containing the titanium raw materials and a lighter mineral fraction (tails). Wet mills can be land-based or floating on water. Rough, heavy mineral concentrate from the wet mill is transported usually by truck or barge to the dry mill for further separation and concentration. The specific flowsheet of a dry mill depends considerably on the type of ore and heavy mineral assemblage to be recovered. Dry mills use various stages of high-tension electrical separation, induced roll magnetic separation, and additional gravity separation methods to produce specific titanium raw material (ilmenite, rutile) and other heavy mineral (zircon, monazite) concentrates. Variations to typical wet and dry mill procedures are used throughout the world, based on the composition and type of mineral sand.

Hard-rock mines such as the MacIntyre development in New York (now only producing magnetite), the Tellnes ilmenite mine in Norway, the Otanmaki Mine in Finland (recently shut down), and the nonproducing Piampaludo deposit in Italy, use or would use flotation methods to recover titanium raw materials. Three nonproducing hard-rock deposits—one each in Brazil, Canada, and the United States—have proposed operations that would use gravity and/or magnetic and high-intensity electrical separation methods to recover titanium concentrate.

All the recovered titanium concentrates from either wet or dry mill or flotation processes are for pigment production using either the chloride or sulfate process. In addition, ilmenite can be used in the production of titanium slag or synthetic rutile, which in turn feed pigment plants or titanium metal plants.

The abundance of ilmenite and scarcity of rutile has led to the research and development of processes for upgrading ilmenite to a low-iron, high-titanium (90-97 pct TiO_2) product called synthetic rutile. Most upgrading proceses fall into four major groups: direct oxidation, often followed by reduction and acid leaching, a pyrometallurgical method, a carbonyl process, and direct acid leaching.

PRODUCTION COSTS

Table 3 and figure 3 show the average operating costs for selected world MEC titanium operations included in this study (expressed as dollars per metric ton of concentrate). The costs for titanium operations in other countries were not included in the table because of the limited numbers of deposits in those countries.

RUTILE

For primarily rutile operations, the mine operating cost primarily represents the cost of dredging and the wet milling (particularly for Australia). For the producers in Australia, this cost is just over $180/mt, while in India and Sri Lanka the mining cost is only one-quarter of that. Mine costs in India and Sri Lanka are considerably less than in Australia because mining is very labor intensive (in India mining is done by hand shoveling) and labor is inexpensive. The mine cost is just over $260/mt for the Australian nonproducers, primarily because of the lower ore grades. Mill costs for primary rutile operations are primarily for dry milling. For producers, this cost averages from $88/mt to $95/mt in both Australia and India and Sri Lanka, and only increases to $118/mt for the Australian nonproducers. The high tax cost for rutile producers in India and Sri Lanka is due to the high federal corporate income tax rate in those countries. The byproduct credits for rutile mines and deposits represent the revenues generated primarily by ilmenite, zircon, leucoxene, and monazite (rare earths). In the rutile operations evaluated, these credits are significant revenue generators, offsetting much or all of the operating costs. The capital recovery costs are lower for the producers because much of their investment is already written off.

ILMENITE

The mine cost for primary ilmenite mines (producers) averages $8/mt in Australia, increasing to $37/mt for the nonproducers; the mill cost for producers is $5/mt and increases to $17/mt for nonproducers. In both cases, this increase is primarily related to the lower ore grades of the nonproducers (therefore requiring more feed and greater upgrading to produce the same product).

It was estimated in the study (but not shown in table 3) that the average synthetic rutile operating cost for plants (producing or developing) in Australia and India is approximately $220/mt of synthetic rutile. Since nearly 2 mt of ilmenite concentrate, at a market price of approximately $40/mt, is necessary to produce 1 mt of synthetic rutile, the cost to produce synthetic rutile ($80 + $220 = $300/mt) is comparable to the 1985 market price of rutile. The 1985 market prices for rutile and synthetic rutile were $360/mt and $350/mt, respectively.

Table 3.—Rutile and Ilmenite production costs for selected MEC's.

(All costs are in January 1985 U.S. dollars per metric ton concentrates on a weighted-average basis)[1]

	Operating cost		Transportation[2]	Byproduct credit[3]	Net operating cost	Recovery of capital[4]	0-pct DCFROR		15-pct DCFROR		
	Mine	Mill					Taxes and royalties[5]	Total cost[6]	Taxes and royalties[7]	Return on investment[8]	Total cost[9]
Primary rutile (natural):											
Australia:											
Producers	183	95	6	239	45	25	16	86	40	37	147
Nonproducers	261	118	35	188	226	57	26	309	157	163	603
India and Sri Lanka:											
Producers	46	88	17	473	−322	44	171	([10])	189	33	([10])
Primary ilmenite:											
Australia:											
Producers	8	5	2	19	−4	11	2	9	3	5	15
Nonproducers	37	17	17	49	22	14	2	38	20	22	78

[1] Costs are expressed in terms of dollars per metric ton of appropriate concentrate; i.e., rutile or ilmenite.
[2] Represents the transportation cost to pigment plant or local port or market.
[3] Includes all byproduct revenue credits for the operation.
[4] Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure, and reinvestments required over the life of the operation.
[5] Includes property, State, Federal, and severance taxes, and royalties, where applicable, calculated at a 0-pct DCFROR.
[6] Equal to sum of net operating costs, taxation, and capital recovery determined at a 0-pct DCFROR.
[7] Includes property, State, Federal, and severance taxes, and royalties, where applicable, calculated at a 15-pct DCFROR.
[8] Revenue increase per ton necessary to obtain a 15-pct DCFROR.
[9] Equal to sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery *plus* the per ton increase necessary to provide a 15-pct DCFROR after taxation.
[10] Revenues from the numerous byproducts at these operations are more than sufficient to cover all costs and still attain a 15-pct DCFROR.

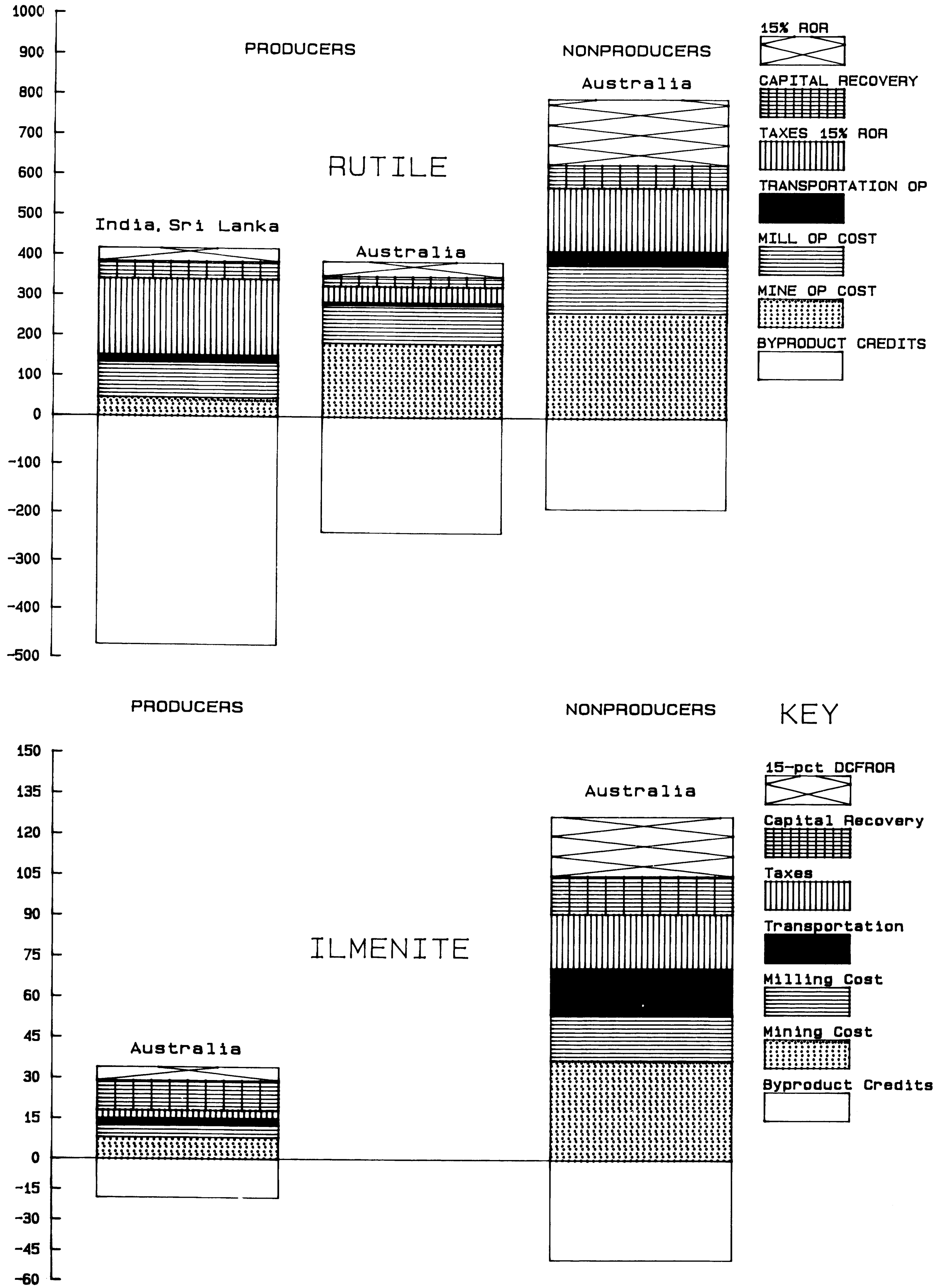

Figure 3.—Rutile and ilmenite production costs for selected MEC's (January 1985 U.S. dollars).

AVAILABILITY

If an operation had more than one *titanium* product, a "primary" product was selected for the price determination. The primary product was defined as the titanium product that generated the greatest revenues. All other products were assumed to be the byproducts. No revenues were generated for byproduct ilmenite, which was assumed as a stockpiled source.

TOTAL RECOVERABLE

Rutile

The 39 deposits containing rutile that were analyzed in this study (17 producers, 22 nonproducers) contain 28.5 million mt of recoverable rutile concentrates, with an average grade of 95 pct TiO_2. These resources were either the primary product of rutile mines or occurred as a coproduct from mines producing ilmenite. Rutile resources available as a primary product are 24.9 million mt, 87 pct of the total.

Estimated recoverable rutile resources located in Australia were 9 million mt, or 32 pct of the total recoverable rutile contained in deposits analyzed in this study, with producing mines in Australia accounting for about 47 pct of the total rutile for Australia. Rutile concentrates recoverable from mines and deposits located in India and Sri Lanka were almost 4 million mt, nearly 13 pct of the total rutile available. Rutile concentrates located in the United States were about 1.2 million mt, or only 4 pct of the total. Other countries with deposits containing recoverable rutile concentrates were Sierra Leone, the Republic of South Africa, Brazil, and Italy. The Italian Piampuludo deposit has the largest future potential of all the nonproducing deposits studied.

The tonnage of rutile concentrates potentially available from the producing mines and nonproducing deposits analyzed in MEC's for which rutile was a primary product or a major coproduct or byproduct is shown on figure 4. Production costs for operating mines are bracketed by lower and upper cost levels. The lower level reflects costs at a 0-pct discounted-cash-flow rate of return (DCFROR), while the upper level reflects total cost, including a 15-pct DCFROR.

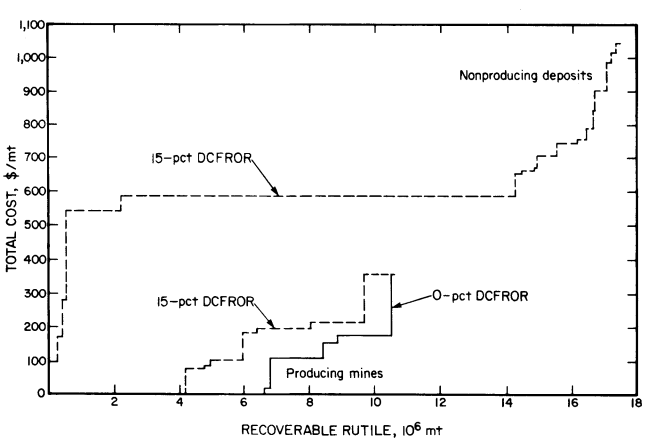

Figure 4.—Potential total rutile available from MEC's (January 1985 U.S. dollars).

The cost curve for nonproducing deposits includes a 15-pct DCFROR. (These production cost levels are explained in the methodology section at the beginning of this Bulletin.)

Figure 4 illustrates that of the 28.5 million mt of recoverable rutile concentrates in the MEC's, producing mines only account for 38 pct (10.7 million mt) of this total. All the tonnage from the producing mines is available at total costs less than the January 1985 market price for rutile of approximately $365/mt, even at a 15-pct DCFROR. However, only 3 pct of the rutile from the nonproducing deposits is available at total costs less than the January 1985 market price. This 3 pct includes a developing mine in Australia as well as three nonproducers whose mining in Australia is environmentally restricted.

Ilmenite

It is estimated that 181 million mt of ilmenite concentrate, containing approximately 54 pct TiO_2, could be recovered from the demonstrated resources of 17 primary product ilmenite mines and deposits, 22 primary product rutile mines and deposits, and 3 mines that feed synthetic rutile operations (the ilmenite at these synthetic rutile mines is that which does not end up feeding the synthetic rutile plants). This indicates an abundance of ilmenite resources based on an estimated 1985 world production level of 3.2 million mt. The analysis assumed that for many potential byproduct sources the ilmenite would be stockpiled rather than sold. Resources of ilmenite that were selected as being a part of a stockpile are those that are presently being stockpiled or which would likely be stockpiled. Most often these stockpiles would be high in chromium content and therefore would not be a desirable product (most are located in Australia). These tonnages were part of the ilmenite resources but no revenues from the sale of ilmenite were credited to the production costs.

Resources associated with primary product ilmenite mines accounted for 60 pct of the total recoverable ilmenite concentrate (or 109 million mt). Ilmenite resources located in Europe contained by far the greatest portion of all primary product ilmenite potentially recoverable (68 pct).

The total recoverable ilmenite from primary ilmenite deposits is shown in figure 5. The total quantity shown is less than 109 million mt because some ilmenite was associated with costs greater than $200/mt, the maximum cost shown. The curve shows that over 79 million mt of ilmenite concentrate was potentially recoverable, primarily from European deposits, at a cost less than the January 1985 market prices of ilmenite in the United States and Australia, $41/mt and $34/mt, respectively. This is over 72 pct of the total primary ilmenite potentially available.

The total availability curve for byproduct iilmenite concentrate from primary product rutile mines is also illustrated on figure 5. For these operations, the availability of ilmenite is dependent on the production of rutile. Ilmenite potentially recoverable as a byproduct from primary product rutile mines was over 65 million mt. This is about 36 pct of the total ilmenite recoverable from all sources. India and Sri Lanka had 28 million mt, 43 pct of the total, and Australia had 23 million mt, or 36 pct of the total. About 41 million mt of byproduct ilmenite (63 pct of the total) was potentially recoverable at a primary product rutile cost less than the January 1985 rutile concentrate market price,

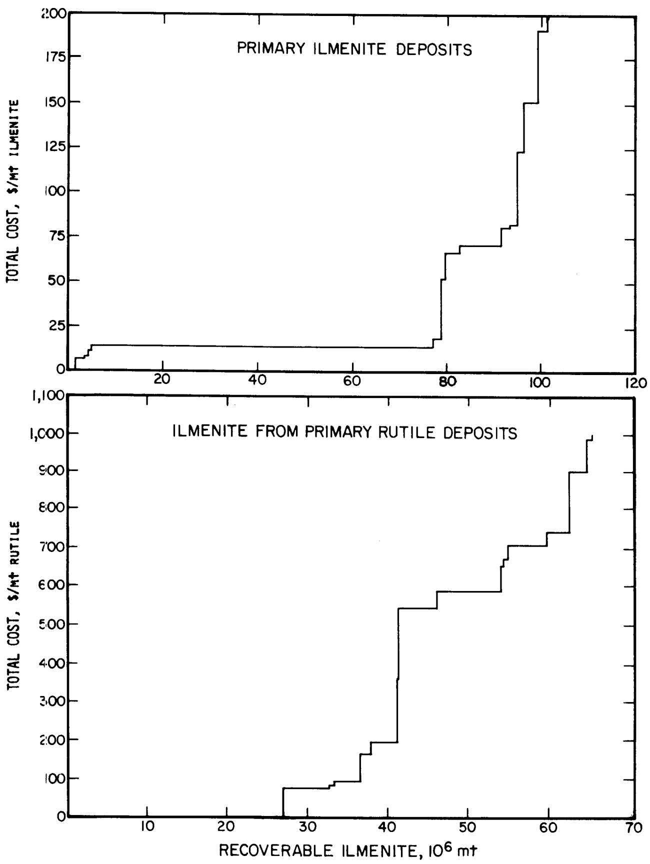

Figure 5.—Potential total ilmenite available from MEC's (January 1985 U.S. dollars; 15-pct DCFROR included).

which was $364/mt. Much of the byproduct ilmenite is presently being stockpiled. Revenues from the stockpiled ilmenite were not included in the analysis.

Some mines produce ilmenite for shipment to synthetic rutile plants, as well as ilmenite concentrate to market. More than 7 million mt of ilmenite concentrate was potentially recoverable as the remaining portion of unused ilmenite from these operations. This is less than 4 pct of all ilmenite potentially available. All of this was potentially recoverable at costs at or below the January 1985 market price for synthetic rutile, $350/mt.

This study indicates that ilmenite resources are vast and that the majority of low-cost ilmenite is contained in European deposits. It also demonstrates that there is a significant amount of low-cost ilmenite that is not being produced or sold. This ilmenite is associated with primary product rutile operations and was stockpiled rather than sold under the market conditions prevailing in January 1985. Much of this ilmenite has a high content of chrome, which is considered a deleterious material.

Other Titanium Sources

It is estimated that approximately 2.5 million mt of leucoxene concentrates, containing approximately 67 pct TiO_2, were potentially recoverable as a byproduct of primary product rutile, primary product ilmenite, or the synthetic rutile operations included in the study. These resources made up only a small portion of the titanium resources included in this analysis. Curves were not drawn for leucoxene because of the small number of deposits.

It is estimated that 21.1 million mt of synthetic rutile concentrates, containing approximately 93 pct TiO_2, was potentially available from seven properties in which ilmenite was being used to feed synthetic rutile plants.

The synthetic rutile operations included in the analysis represent those with current or proposed production of synthetic rutile from ilmenite. The plants included were located in Australia and India; the only nonproducer was located in New Zealand (two properties in Australia are developing). Synthetic rutile plants located in the United States, Japan, Malaysia, and Taiwan were not included because the source of the ilmenite processed could not be determined.

Much of the ilmenite potential, discussed in earlier sections of this chapter, could be processed to produce synthetic rutile if new plants were built. The world resources of ilmenite are very large, and the costs to produce ilmenite and convert it to synthetic rutile are quite comparable to those for the production of rutile, therefore, the potential production of synthetic rutile could be significant in the future. Owing to the fact that the longrun availability of rutile is limited, the production of synthetic rutile is expected to grow.

It is estimated that 148.1 million mt of titanium slag, containing at least 71 pct TiO_2, is potentially recoverable from the seven properties included in this study. Slag potentially recoverable from two properties in Canada was 83.5 million mt, 56 pct of the total slag included in this study. Slag potentially recoverable from the Republic of South Africa was 8.7 million mt, almost 6 pct of the total slag included in this study. Producing operations (plus the developing one in Norway), all of which have a cost of production below the January 1985 market prices of $193/mt for Sorel slag and $209/mt for Richards Bay slag, had resources of 120.9 million mt potentially recoverable. This is 82 pct of the total titanium slag potentially recoverable from properties included in this study.

The quantity of titanium slag that potentially is recoverable, as with that of ilmenite, is large. Some slag is high grade and has been used in place of rutile as a feed for chloride pigment production. Additional high-grade slag may be potentially recoverable in the future, as was recently achieved with the upgrading of the Sorel slag in Canada.

Although it has a slightly lower grade, anatase is a potential replacement for rutile. Anatase potentially recoverable from three properties in Brazil was 62 million mt concentrate. Two of the Brazilian properties, Tapira and Catalao, were being developed during 1985. Because there is no published market price for anatase, cost comparisons are not possible. However, the total cost of production estimated for these deposits is between the present market prices of rutile and ilmenite, $364/mt and $34/mt, respectively.

It is estimated that 6.7 million mt of a mixed ilmenite-leucoxene concentrate was potentially available from four properties in the southeastern United States (two producing and two nonproducing). This high-grade concentrate (greater than 60 pct TiO_2) is used or would be used for the specially designed Du Pont chloride plants. Du Pont owns the two producing properties. Because the product was produced and consumed in integrated operations, total cost comparisons are not possible.

ANNUAL CAPACITY

Rutile

Potential 1985 production at full capacity from operating MEC mines studied is illustrated in figure 6. These mines had the capacity to produce an estimated 418,000 mt of rutile concentrate in 1985. The curve is bracketed by a lower operating cost level and an upper 0-pct DCFROR cost level, which includes a return of capital but no profit. At costs of approximately $364/mt, roughly equivalent to the January 1985 price for rutile concentrate, all of the 418,000 mt was available. The estimated MEC production of rutile concentrate in 1985 (excluding the United States, which is withheld to provide confidentiality) was approximately 355,000 mt.

Figure 7 shows the potential annual production capacities for the 1985-95 period for rutile mines that were in production at the time of this analysis (1985). The curve, representing annual production at operations with costs of production less than $360/mt (only slightly less than the approximate January 1985 market price of $364/mt), gradually declines until the year 1993, when it declines at a greater rate. By the year 1995, an estimated 290,000 mt could be produced, which is approximately 82 pct of the 1985 level. However, this decline could be offset if new resources are found, if inferred tonnages become demonstrated, or production begins from properties that currently are not producing, such as the higher cost rutile mines or the Brazilian anatase deposits.

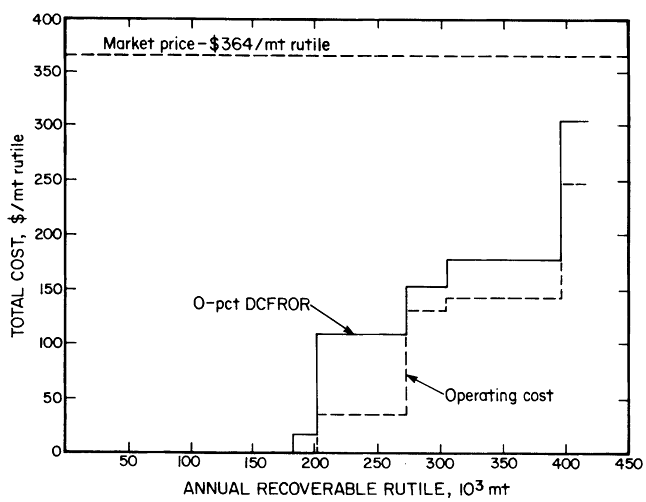

Figure 6.—1985 rutile capacity from producing mines in MEC's (January 1985 U.S. dollars).

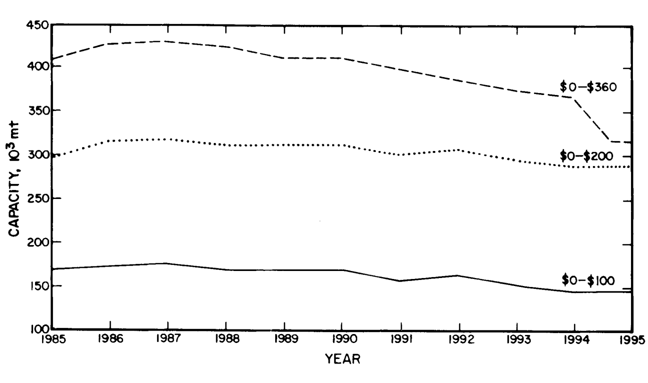

Figure 7.—Potential annual rutile production from producing mines in MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

Ilmenite

Annual production from producing operations of both primary product and byproduct ilmenite, with costs near or below market prices in 1985, could potentially have been 2.9 million mt. This quantity may not match actual production figures because it assumes mines are operating at full capacity, and includes byproduct ilmenite, some of which is assumed in this analysis to be stockpiled and not sold. In addition, most published data for ilmenite production include ilmenite going to slag or synthetic rutile, while this analysis includes such ilmenite separately.

Other Titanium Sources

Annual availability of leucoxene concentrate as a byproduct of producing primary product rutile operations, primary product ilmenite operations, or from mines feeding synthetic rutile operations with costs near or below the market price, could have potentially been 88,000 mt in 1985. Actual production values for leucoxene are not reported; therefore, no comparison with 1985 production can be made.

Annual availability of synthetic rutile from the producing operations (and ones presently developing) included in this study could potentially be approximately 400,000 mt. Because not all synthetic rutile plants are included in this study, comparisons with 1985 production levels cannot be made.

Annual availability of titanium slag from producing operations could potentially have been a maximum of 1.3 million mt in 1985. This compares closely with the 1985 world production estimate of 1.2 million mt of slag. At that level, slag production could be maintained into the next century.

No anatase mines were operating as of January 1985, although development of two continued. Annual availability of anatase from these operations could potentially be a maximum of over 700,000 mt. This large tonnage is a significant potential replacement source for rutile, and could be produced for many years.

AVAILABILITY OF BYPRODUCT ZIRCONIUM AND HAFNIUM

In addition to the various titanium minerals recoverable from beach sand operations, zircon concentrates often contribute significantly to the economic viability of an operation. It is estimated that zircon concentrates (which contain approximately 65 pct ZrO_2) that could be recovered from 40 titanium mines and deposits included in this study total just under 23 million mt. Australia accounts for nearly two-thirds of the total, while U.S. mines and deposits account for 16 pct. At the January 1985 market price for zircon concentrate of $182/mt in the United States (although only about $94/mt in Australia), approximately 14 million mt was available (or over 60 pct of the total), nearly two-thirds of which is from Australia.

At or near the January 1985 market price for titanium concentrates, over 600,000 mt of zircon was available in 1985 from the producing mines included in this study. This nearly equals the 610,000 mt produced in 1985 (excluding the United States, U.S.S.R. and China) (7). Owing to the fact that a majority of zircon is produced from rutile mines, production from which this study has shown to be declining in the next 10 yr, the annual availability of zircon concentrates from producing mines should also decline.

The availability of hafnium can be directly related to the availability of zircon, since they are nearly always found together in nature. As a rule of thumb, there is typically 2 pct Hf found with zirconium (or 2-pct HfO_2 of the amount of ZrO_2). Therefore, based on the quantity of zircon concentrates this study has shown to be potentially available (approximately 23 million mt) as of January 1985, a total of 460,000 mt of hafnium (HfO_2) is also potentially available.

CONCLUSIONS

Titanium is used primarily in the form of titanium dioxide as a source of pigments. Titanium metal is considered a strategic and critical material for the United States because of its defense and aerospace applications. In an attempt to assess the worldwide availiability of titanium mineral resources, the Bureau of Mines evaluated 61 mines and deposits in MEC's. The mines and deposits in this study include all known resources of titanium at the demonstrated resource level that met the study criteria and can be mined and processed with current technology, as of January 1985.

This analysis has determined that in the long run, resources of rutile are limited (only 10.7 million mt concentrates from producers and 17.8 million mt from the nonproducers). It has also determined that there are various other sources of high-grade titanium. Anatase deposits in Brazil have the potential to be a significant source (containing 62 million mt concentrate) and are, in fact, developing; slag deposits in Canada and the Republic of South Africa could increase capacity, and an increase in the Republic of South Africa is soon to occur; higher cost rutile deposits could be developed; and, most importantly, an increase in world synthetic rutile capacity, based on the large resources of ilmenite (181 million mt of total concentrate), could become the most significant future resource of high-grade titanium concentrate (this, too, is also presently occurring).

REFERENCES

1. Fantel, R.J., D.A. Buckingham, and D.E. Sullivan. Titanium Minerals Availability—Market Economy Countries. A Minerals Availability Appraisal. BuMines IC 9061, 1986, 48 pp.
2. Lynd, L.E. Titanium and Titanium Dioxide. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 168-169.
3. ______. Titanium. Ch. in Minerals Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 859-879.
4. Lynd, L.E., and R.A. Hough. Titanium. Ch. in BuMines Minerals Yearbook 1985, v. 1, pp. 975-989.
5. Lynd, L.E. Ilmenite. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 72-73.
6. ______. Rutile. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 132-133.
7. Adams, W.T. Zirconium. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 180-181.
8. Force, E.R., and L.E. Lynd. Titanium Mineral Resources of the United States—Definitions and Documentation in Contributions to the Geology of Mineral Deposits. U.S. Geol. Surv. Bull. 1558B, 1984, pp. B1-B11.

TUNGSTEN

INTRODUCTION

Worldwide, tungsten is marketed in four major intermediate forms: scheelite or wolframite concentrate, ammonium paratungstate (APT), ferrotungsten, and artificial scheelite. The addition of natural or artificial scheelite to steel imparts properties of greater hardness, wear resistance, and high heat resistance. It is an important constituent of steel for tools, drill bits, and other similar applications. Nearly all APT is used to produce tungsten carbide, which is cemented, usually with cobalt, to form various cutting and wear-resistant products. Most tungsten carbide is used for metalworking machinery and by the mining and oil industries, for which tungsten carbide's high melting point, high compressive strength, hardness, and resistance to oxidation are necessary.

All properties evaluated were assumed to produce one or more of three major products (concentrate, APT, or ferrotungsten) as the first marketable product. For each property, it was determined which product forms were actually or likely to be produced, based on the character of the concentrate and market considerations. While it is possible to process ores from many of the operations into any of the three products, cost and market considerations generally determine the most profitable product from each property.

The method used in this evaluation for creating a comparable homogeneous product was to process nonstandard (low-grade or high-impurity) concentrates to APT or ferrotungsten, depending on ore mineralogy. Production from scheelite properties would likely be concentrate; properties containing ferberite as the major ore mineral were evaluated as ferrotungsten properties. All remaining properties were evaluated as APT properties, even though some may actually sell concentrates of various grades and qualities. Availability is presented in terms of the amount of tungsten contained in the evaluated product. The costs of converting low-grade or otherwise poor quality wolframite concentrates into a usable product are reflected in the average total cost of production estimates for APT properties.

Because of the paucity of data regarding centrally planned economy country (CPEC) deposits and operations, only market economy country (MEC) properties were evaluated.

In contrast to many basic commodity markets, there are no terminal market price quotations for tungsten ore. Almost all transactions involve a milled or processed form of tungsten, with sales negotiated between producers and consumers or merchants and users. Marketing and pricing practices for processed tungsten products are more diversified than for concentrates, the prices for which are contained in the *Metal Bulletin*. APT and ferrotungsten prices are obtained by using the *Metal Bulletin* published prices and adjusting for conversion charges.

Most of the information presented in this chapter is an updated summary of Bureau of Mines Information Circular 9025 "Tungsten Availability—Market Economy Countries. A Minerals Availability Program Appraisal" (*1*).[1] Additional information on the domestic and foreign tungsten industry is available from other Bureau publications (*2-4*).

[1] Italic numbers in parentheses refer to items in the list of references at the end of this chapter.

GEOLOGY AND RESOURCES

GEOLOGY

Tungsten occurs in three main types of economically important deposits: (1) contact metamorphic-metasomatic scheelite-bearing tactites (skarns), (2) wolframite-bearing quartz veins, and (3) deposits of volcanogenic origin. Tungsten also occurs in pegmatites, placers, brines, and deposits associated with porphyries. In general, tungsten mineralization is genetically associated with felsic igneous rocks such as granodiorite, quartz monzonite, and granite.

Tactites are formed through high-temperature replacement and recrystallization of calcareous sedimentary rocks at or near the contact with an igneous intrusion. Tactites generally have distinct boundaries, but commonly, scheelite mineralization is erratically distributed. Although tactite bodies are frequently small and irregular in shape, the largest known tungsten occurrence, Shizhuyuan, China, contains an estimated 500,000 mt of tungsten. Important tactite deposits evaluated for this study include Sangdong in the Republic of Korea, Pine Creek in California, and Cantung in Canada.

Tungsten-bearing vein deposits are widely distributed geographically, and account for more than half of the world's reserves (*2*). The veins occur as discrete bodies, vein swarms, or stockworks, usually found in roof zones associated with acidic igneous intrusions. Tungsten is mainly present as wolframite, huebnerite, or ferberite, but scheelite may also occur. The greatest commercial concentration of vein-type deposits is in southeastern China, where the famous Xihuashan Mine is located. Other well-known vein deposits included in this study are those of Bolivia and Thailand.

Tungsten deposits associated with porphyries include Mount Pleasant, Canada, and Climax, in Colorado, where tungsten is produced as a byproduct of molybdenum mining. Climax was not included in this evaluation because of the byproduct status of tungsten.

Two deposits of unique geologic character include Mittersill, Austria, and Searles Lake, CA. Mittersill is a stratiform deposit that apparently is related to submarine volcanic activity. Searles Lake, one of the largest known potential sources of tungsten in the United States, is a brine deposit that was not included in this evaluation. Additional geologic information on tungsten deposits is available from the U.S. Geological Survey (*5*).

RESOURCES

Figure 1 shows the amount of contained tungsten in properties evaluated for this study and the world reserve base estimate. The important countries shown separately are Canada, the United States, and Australia. This evaluation includes a total of approximately 1,315,000 mt of contained tungsten at the demonstrated level, compared with more than 1.6 million mt shown on the reserve base for all MEC's. An additional 1.8 million mt was not evaluated owing to a lack of cost data.

Table 1 shows the total amount of recoverable tungsten potentially available from evaluated properties. Canada contains the largest portion, accounting for more than half of the total. The United States accounts for approximately 11 pct, but the large resources in the Searles Lake and Climax deposits have not been included because tungsten is or would be produced as a byproduct. If included, the total amount of recoverable tungsten for the United States would probably be twice the amount shown.

Table 2 lists the properties evaluated for this study.

Table 1.—Summary of MEC demonstrated tungsten resources evaluated for this study, as of January 1985

Country	In situ ore, 10^6 mt	In situ grade, pct	Recoverable, 10^3 mt
Australia	73.5	0.20	91.4
Austria	3.5	.40	9.6
Bolivia	2.8	.54	8.7
Brazil	3.9	.42	10.5
Burma	.4	.40	.5
Canada	246.1	.27	507.3
France	2.7	.69	11.2
Mexico	4.5	.36	5.9
Namibia	3.3	.20	3.9
Peru	.7	.36	1.7
Portugal	5.5	.29	11.8
Republic of Korea	7.6	.68	42.9
Spain	28.1	.12	14.6
Sweden	1.2	.34	3.5
Thailand	3.9	.80	15.9
Turkey	13.3	.40	26.6
Uganda	.9	.19	1.2
United Kingdom	56.0	.13	43.5
United States	84.1	.17	103.3
Total or av	542.0	.25	914.0

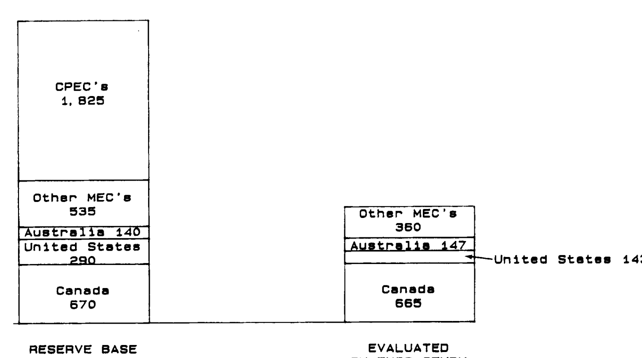

Figure 1.—Estimates of world tungsten resources (thousand metric tons tungsten content).

Table 2.—MEC tungsten properties included in this study

Location and property name	Owner and/or operator	Current status[1]	Primary product	Mining method[2]
Australia:				
Kara	Tasminex, McIntyre Mine	P	Natural scheelite	S
King Island	Peko-Wallsend Ltd.	P	Natural and artificial scheelite	U
Mount Carbine	Queensland Wolfram Ltd.	P	APT	S
Mount Mulgine	Minefields Exploration	N	APT	S
Torrington	Barix Proprietary Ltd.	N	APT	S
Austria: Mittersill	Metallgesellschaft, Voest-Alpine	P	APT	U
Bolivia:				
Bolsa Negra	Comibol	P	APT	U
Chambillaya	International Mining Co.	P	Ferrotungsten	U
Chojlla	..do	P	APT	U
Enramada	..do	P	APT	U
Kami	Comibol	P	APT	U
Pueblo Viejo	Senora Eva Thiel de Sara	P	APT	U
Tasna	Comibol	P	APT	U
Viloco	..do	P	APT	U
Brazil:				
Barra Verde	Mineracao Sertaneja Ltda.	P	Natural scheelite	U
Boca de Lage	Tungstenio do Brasil Minerios e Metais	P	Do	U
Brejui	Mineracao Tomas Salustino S.A.	P	Do	U
Zangarelhas	Tungstenio do Brasil Minerios e Metais	N	Do	U
Burma: Mawchi	Burmese Government	P	APT	U
Canada:				
Cantung	Canada Tungsten Mining Corp.	P	Natural scheelite	U
Logtung	AMAX, Logtung	N	APT	S
Mactung	AMAX Northwest Mining Co. Ltd.	N	Natural scheelite	U
Mount Pleasant	Billiton Canada, Sullivan Mining	PP	APT	U
France:				
Montredon	Bureau des Recherches Geologiques, Penarroya	PP	APT	S
Salau	Compagnie Metallurgique et Miniere; L'Omnivini des Mines.	P	Natural scheelite	U
Mexico:				
Baviacora	Tungsteno de Baviacora SA	P	APT	S
Los Verdes	Cia Minera Coronado SA de CV	N	APT	S
San Alberto	Draco	P	APT	S
Namibia:				
Brandberg West	Southwest Africa Co., Ltd.	PP	APT	S
Krantzberg	Nord Resources, Bethlehem Steel	PP	APT	U
Peru: Pasto Bueno	Fermin Malaga Y Santolalla	P	APT	U
Portugal:				
Borralha	Minas da Boralha Sarl	PP	Ferrotungsten	U
Panasqueira	Serra d Estrela	P	APT	U
Republic of Korea: Sangdong	Korea Tungsten Mining Co. Ltd.	P	Natural scheelite	U
Spain:				
Barruecopardo	Coto Minero Merladet SA	P	APT	S
La Parilla	Minero Bonilla	P	APT	S
Santa Comba	Coparix Minera SA	P	APT	U
Sweden: Yxsjoberg	Luossavaara Kiirunavaara AB	P	Natural scheelite	U
Thailand:				
Doi Mok	Sirithai Scheelite	N	APT	U
Doi Ngoem	Parasit Mining Co.	P	Ferrotungsten	S
Khao Soon	Siamerican Mining Co. Ltd.	PP	Do	U
Turkey: Uludag	Etibank	P	APT	C
Uganda: Nyamalilo	Ugandan Government	PP	Ferrotungsten	S
United Kingdom: Hemerdon	AMAX, Hemerdon Mining	N	APT	S
United States:				
Adamson	Panaminas	P	APT	U
Andrew	Curtis Tungsten Inc.	P	APT	C
Emerson	Teledyne Inc.; North Tempiute Mining and Development Co.; Union Carbide.	PP	APT	U
Indian Springs	Utah International	N	APT	U
Nevada Scheelite	Natural Resources Development Inc.	PP	APT	U
Pilot Mountain	Union Carbide	PP	APT	S
Pine Creek	..do	P	APT	U
Springer	Utah International (General Electric)	PP	APT	U
Strawberry	Teledyne	P	APT	U
Thompson Creek	R. M. Barrett	PP	APT	U
Tungsten Queen	Hecla Mining Co.	PP	APT	U

[1] P, producer; N, nonproducer; PP, past producer.
[2] S, surface; U, underground; C, combined.

U.S. AND WORLD HISTORICAL PRODUCTION

World mine production was estimated to total 45,100 mt of tungsten in 1985, of which 53 pct was from MEC's (fig. 2). China, the world's largest producer, accounted for nearly 30 pct of world production. Together, China and the U.S.S.R. produced about half of the world's total tungsten in 1985. In 1985, the United States accounted for slightly more than 2 pct of world production and about 5 pct of MEC production. The largest MEC producer was Canada, with an estimated 3,000 mt, or 14 pct of MEC production.

During the 1950-85 period, China has been the most important world producer, and has produced nearly a third of total world output since 1980. The U.S.S.R. has also accounted for a relatively high percentage of world production throughout the time period. Among individual MEC's, the United States, Australia, Canada, and Republic of Korea have been important producers, although United States production, as a percentage of the total, has declined considerably, from a high of 13 pct in 1965 and 1970, to 2 pct in 1985. U.S. production has declined considerably over the historical period shown, from a high of 7,090 mt in 1955 to 1,100 mt in 1985, because of relatively high production costs compared with those of other producing countries.

EXTRACTION AND PROCESSING TECHNOLOGY

MINING

Table 2 shows general mining methods for each evaluated property. Many of the smaller deposits are or would likely be mined simply by following mineralized outcrops from the surface to some depth, without great regard for grade control, productivity, or future mining plan. This approach is widely practiced in Bolivia, Burma, Mexico, Thailand, and several other less developed countries. For larger deposits in more industrialized countries, the mining method depends upon depth and geometry of the ore body, competency of the ore and country rock, ore grade, development of a mine plan, and required capital investment.

Eighteen of the properties evaluated, accounting for 28 pct of mine capacity, are or would likely be mined wholly or largely by surface methods. The size, efficiency, and degree of mechanization vary widely, from the small, labor-intensive Mexican mines to the large, highly mechanized open pit Mount Carbine operation in Australia.

About two-thirds of the total recoverable tungsten in evaluated properties is potentially available from the 36 underground mines and deposits. As with surface operations, the size, efficiency, and degree of mechanization vary widely. Most of the Bolivian mines are developed on steeply dipping, irregular veins that require conventional stoping methods. These mines range in ore capacity from 3,400 to 180,000 mt/yr. Cantung, an important Canadian mine, employs the room-and-pillar method at a mine ore capacity of 350,000 mt/yr.

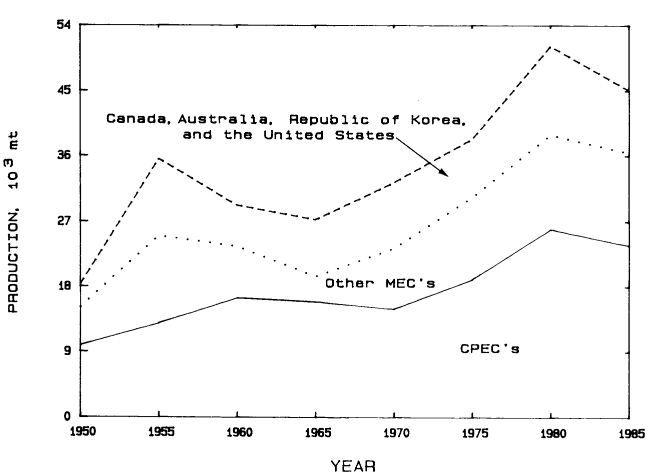

Figure 2.—World tungsten production, 1950-85.

PROCESSING

The brittle nature and high specific gravity of tungsten minerals, especially scheelite and wolframite, require careful processing in order to obtain acceptable recoveries. Generally, processing includes crushing and grinding, followed by gravity and/or flotation to produce a concentrate. Nearly all tungsten ores require gravity methods, although some scheelite ores can be floated. Some mines, such as Doi Ngoem in Thailand and some of the Bolivian operations, employ hand sorting prior to milling as a means of upgrading the ore.

Some operations produce tungsten concentrates that require additional treatment before they can be processed to APT or ferrotungsten. The purpose of this practice is to reduce energy and chemicals consumption. Major upgrading processes include flotation to remove base metal, arsenic, and molybdenum sulfides; magnetic separation to remove wolframite and garnet; electrostatic separation to remove cassiterite; roasting to eliminate arsenic, sulfur, flotation oils, and organics; grinding to ensure higher solubility; and acid leaching to remove carbonates.

PRODUCTION COSTS

Operating and production costs are shown on table 3 and figure 3 in terms of January 1985 U.S. dollars per pound of recovered tungsten. Costs are presented for countries and regions for which a sufficient number of properties were available so that individual property data are not disclosed. The weighted-average figures include both underground and surface operations for countries and regions shown, and also include a mix of final product types. For this reason, postmill processing costs are not directly comparable among the various country and region groups. For example, postmill costs are relatively high for the United States, because all U.S. properties were assumed to produce APT as the marketable product, while the majority of product available from Brazil and Peru is concentrate.

As expected, production costs for producers are generally lower than for nonproducers. The weighted-average net operating cost (mine, mill, transportation, and postmill processing) of $9.27/lb for United States producers, the highest of any country group shown, results from expensive mining methods employed at several U.S. operations, and the additional cost of postmill processing (to APT). Many of the non-U.S. properties produce a concentrate as the first marketable product, so their evaluation did not include a postmill processing cost. The low net operating cost for the "other" country grouping ($3.04/lb) is heavily influenced by two large, highly mechanized producers, Sangdong and Cantung, which together account for 90 pct of the total recoverable tungsten in the grouping.

Among nonproducers, several properties display net operating costs that are lower than those for producers, although in many cases, the large amounts of capital necessary to bring the properties on-line would prohibit development. As with producers, net operating costs for U.S. nonproducers are relatively high. This is in part a consequence of the fact that the selection process of properties for this study resulted in inclusion of a relatively large number of deposits in the United States, for which resource data are more readily available than for nonproducing deposits in foreign countries. Adequate resource and cost data are generally available only for foreign deposits that are likely to be developed in the foreseeable future, owing to their favorable cost position compared with producers. However, U.S. nonproducers, all underground, would be mined using relatively expensive methods dictated by the characteristics of the ore bodies, which are similar to those at producing U.S. operations that have high operating costs.

The extraordinarily high cost (including a 15-pct DCFROR) for some properties, notably those in Canada and Australia, results from the need to recover large amounts of capital over the producing life of the properties. This is especially the case for large, undeveloped properties in remote areas (i.e., requiring large infrastructure costs) that require relatively long lead times for development, such as Logtung, in Canada, and Mount Mulgine, in Australia.

Table 3.—Production costs for MEC tungsten properties

(January 1985 U.S. dollars per pound of tungsten)

Country	Operating cost: Mine	Operating cost: Mill	Post-mill processing[1]	Byproduct credit	Net operating cost	Recovery of capital[2]	0-pct DCFROR: Taxes and royalties[3]	0-pct DCFROR: Total cost[4]	15-pct DCFROR: Taxes and royalties[5]	15-pct DCFROR: Return on investment[6]	15-pct DCFROR: Total cost[7]
PRODUCERS											
Australia	1.85	1.28	1.27	0.43	3.97	0.70	0.04	4.71	0.82	1.86	7.35
Bolivia	2.63	.88	1.57	.27	4.81	.98	.91	6.70	1.22	.51	7.52
Brazil-Peru	1.97	1.00	.72	.18	3.51	1.84	.58	5.93	.94	.64	6.93
Europe	2.53	1.20	.94	.37	4.30	1.12	.23	5.65	.63	.56	6.61
United States	5.04	2.18	2.05	0	9.27	.58	.16	10.01	.35	.76	10.96
Other[8]	1.52	1.01	1.12	.61	3.04	.90	.11	4.05	.25	.68	4.87
NONPRODUCERS											
Africa	1.45	1.92	1.21	2.73	1.85	2.59	0.78	5.22	1.99	2.54	8.97
Australia	1.64	1.50	1.98	.22	4.90	2.72	.29	7.91	5.92	7.35	20.89
Canada	2.00	1.51	1.96	.49	4.98	2.92	.18	8.08	4.55	8.72	21.17
United States	3.64	2.98	4.10	0	10.72	1.13	.02	11.87	.61	2.03	14.49
Other[9]	.93	.61	1.06	0	2.60	2.41	.74	5.75	2.86	3.49	11.36

[1] Includes conversion to APT and transportation.
[2] Includes cost of recovering remaining undepreciated investments in exploration, acquisition, development, mine and mill plant and equipment, and infrastructure as of January 1985, and investments required over the life of the operation.
[3] Includes property, State, Federal, and severance taxes and royalties, where applicable.
[4] Equal to sum of net operatiing costs, taxation, and capital recovery determined at a 0-pct DCFROR.
[5] Taxes calculated at a 15-pct DCFROR.
[6] Revenue increase per pound necessary to obtain a 15-pct. DCFROR.
[7] Equal to sum of net operating costs, taxation generated at a 15-pct DCFROR, capital recovery *plus* the per pound increase necessary to provide a 15-pct DCFROR after taxation.
[8] Includes Canada, Mexico, and Republic of Korea.
[9] Includes Brazil, Mexico, and Thailand.

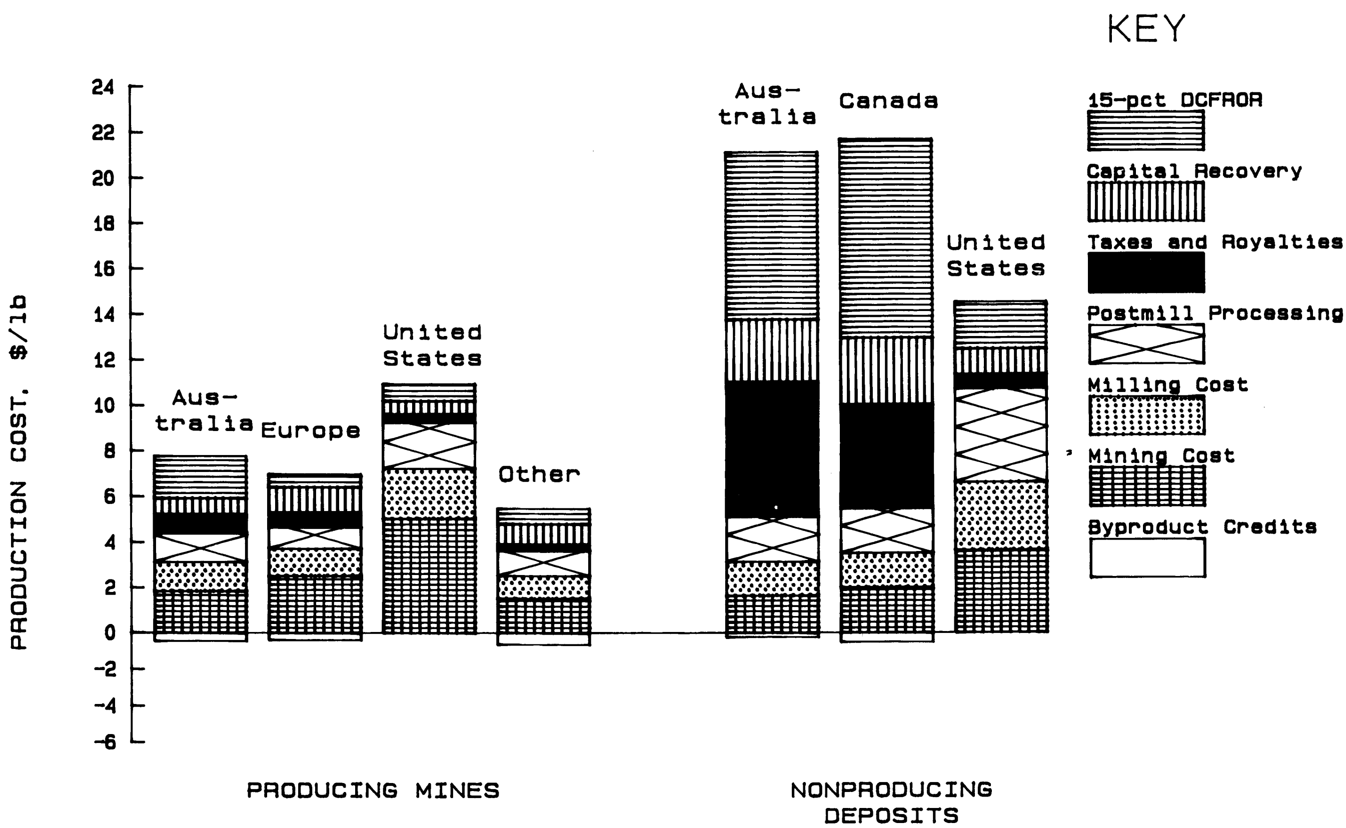

Figure 3.—Tungsten production costs for selected MEC's (January 1985 U.S. dollars).

AVAILABILITY

As part of this analysis, it was determined what the likely first marketable product was for each property, based on ore minerology, market considerations, and other information about the operation. The three main products were APT, natural or artificial scheelite, and ferrotungsten. Only the APT curves are presented because it would be possible to identify individual properties on the scheelite and ferrotungsten curves. Availability of scheelite and ferrotungsten are discussed in a narrative format.

TOTAL RECOVERABLE

Twenty-six properties were evaluated as APT producers, accounting for a total of approximately 140,000 mt of recoverable tungsten contained in APT (fig. 4). The weighted-average cost to produce the total amount is $8.43/lb W at a 15-pct DCFROR ($7.47 at 0-pct DCFROR). This compares with the January 1985 market price of approximately $5.00/lb. Of the 140,000 mt, more than 60 pct is contained in five properties: Uludag, Turkey; Panasqueira, Portugal; Mount Carbine, Australia; and Pine Creek and Emerson, United States. U.S. properties contain approximately 51,000 mt of recoverable tungsten (as APT) at the demonstrated level, with a weighted-average production cost of $11.00/lb W at a 15-pct DCFROR.

Thirteen properties were evaluated as nonproducers that would likely produce APT as the primary marketable product. The total amount of tungsten potentially available from the 13 properties is 342,000 mt, which could be produced at a weighted-average cost of about $22.00/lb W (15-pct DCFROR). Nearly half of the total amount is contained in Logtung, Canada. The four U.S. properties evaluated as APT nonproducers (Tungsten Queen, Pilot Mountain, Thompson Creek, and Indian Springs) account for 15 pct of the 342,000 mt at all nonproducing APT properties, and could be produced at a weighted-average cost of $14.50/lb W (15-pct DCFROR) contained in APT.

Nine properties were evaluated as natural or artificial scheelite producers, accounting for about 96,000 mt of recoverable tungsten contained in scheelite concentrate. Of this total, nearly 70 pct is contained in two properties, Sangdong, Republic of Korea, and King Island, Australia. The weighted-average cost of producing scheelite from the nine producers is $5.23/lb W at a 15-pct DCFROR. The January 1985 market price was $4.65/lb W. None of the U.S. properties were evaluated as scheelite producers.

Only two properties, Mactung, Canada, and Zangarelhas, Brazil, were evaluated as nonproducers that would likely produce scheelite as the primary marketable product. Together, they contain approximately 320,000 mt of recoverable tungsten.

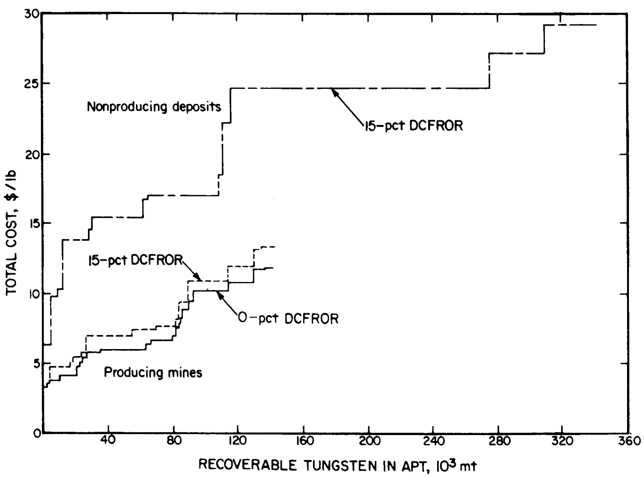

Figure 4.—Potential total tungsten, as APT, available from MEC's (January 1985 U.S. dollars).

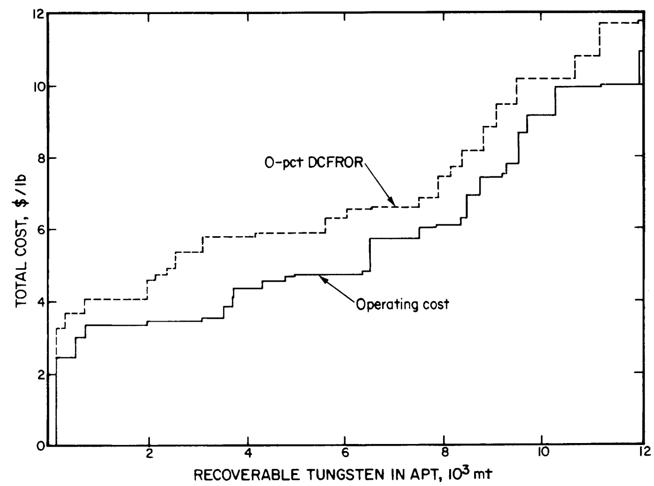

Figure 5.—1985 tungsten, as APT, capacity from producing mines in MEC's (January 1985 U.S. dollars).

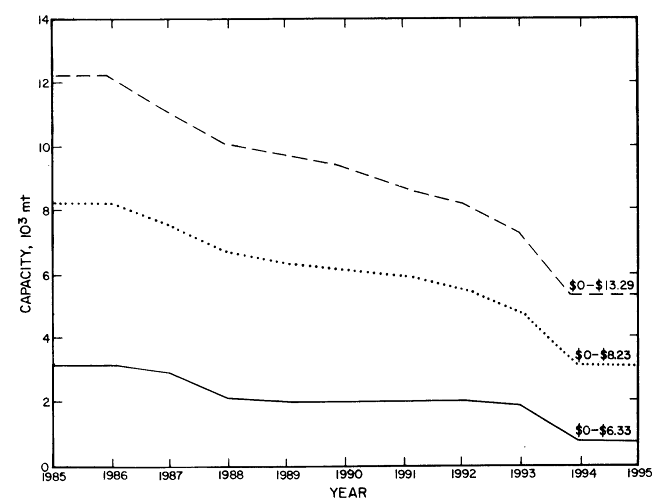

Figure 6.—Potential annual tungsten, as APT, production from producing mines in MEC's at selected production cost ranges, 1985-95 (January 1985 U.S. dollars; 15-pct DCFROR included).

There is about 16,000 mt of tungsten contained in the five properties that were evaluated as ferrotungsten producers and nonproducers. About 70 pct of the total is contained in one property, Khao Soon, Thailand, which is a past producer. The weighted-average cost for the three producers (Chambillaya, Bolivia; Borralha, Portugal; and Doi Ngoem, Thailand) is $9.95/lb W at a 15-pct DCFROR; for all five properties, the weighted-average cost is $10.30/lb W.

ANNUAL CAPACITY

Total MEC tungsten production in 1985 was estimated to be 21,400 mt. Producing properties evaluated for this study had an estimated combined capacity level of about 22,300 mt in 1985.

Potential 1985 production at estimated production levels for producing and temporarily shut down APT mines evaluated is shown in figure 5. There is a total of about 12,200 mt of tungsten potentially available. U.S. properties account for about 28 pct of the total.

The nine scheelite producers have an estimated 1985 production level of about 9,700 mt of tungsten, of which more than 70 pct is contained in three properties (Sangdong, King Island, and Cantung). About 60 pct of estimated 1985 production potentially available from producing mines, or more than 5,800 mt, is in properties with a weighted-average cost (including a 15-pct DCFROR) less than or equal to the January 1985 price of $4.65/lb W.

Figure 6 shows potential annual production of tungsten available as APT, from 1985 to 1995, from the demonstrated resources of producing mines in MEC's. The curves reflect estimated production levels of existing mines, including known planned expansions. About 25 pct of the 12,200 mt available in 1985 is available at a 15-pct-DCFROR cost equal to the January 1985 price of about $6.30/lb W contained in APT; less than 200 mt is available from U.S. properties at that cost.

After 1986, annual production from demonstrated resources could begin to decline rather quickly, and by 1995 projected production from current producers is only 40 pct of the amount potentially available in 1985. However, there are two reasons why the decline will likely be less than that shown. First, the current weak market means that some operations will continue to produce at less than capacity. Second, some properties, notably those in Bolivia, are likely to have resources in excess of the currently reported demonstrated quantity.

CONCLUSIONS

The United States, which presently accounts for only about 2 pct of world tungsten production, and 5 pct of MEC production, has operating and production costs that are among the highest of any producing country in the world. Unless market conditions improve considerably, it is likely that the United States will continue to be a relatively minor world and MEC producer.

The annual availability results show dramatically declining potential availability by 1995, although this is largely a result of depletion of currently demonstrated resources at several properties, such as those in Bolivia, where there are likely to be more resources at lower levels of probability that will be reclassified as demonstrated in the future. It is unlikely that the amount of tungsten potentially available annually from producing or temporarily closed mines will decline in the near future.

There are substantial amounts of demonstrated resources contained in undeveloped properties, several of which show operating costs that compare favorably with producers, that would be able to replace depleted resources in producing properties. However, large capital investments would be necessary to bring several of the properties into production.

REFERENCES

1. Anstett, T.F., D.I. Bleiwas, and R.J. Hurdelbrink. Tungsten Availability—Market Economy Countries. A Minerals Availability Program Appraisal. BuMines IC 9025, 1985, 51 pp.

2. Stafford, P.T. Tungsten. Ch. in Mineral Facts and Problems, 1985 Edition. BuMines B 675, 1985, pp. 881-894.

3. ______. Tungsten. Ch. in BuMines Minerals Yearbook 1985, v. 1, pp. 991-1005.

4. ______. Tungsten. Sec. in BuMines Mineral Commodity Summaries 1986, pp. 170-171.

5. Hobbs, S.W., and J.E. Elliott. Tungsten. Ch. in United States Mineral Resources. U.S. Geol. Surv. Prof. Paper 820, 1973, pp. 667-678.

☆ U.S. GOVERNMENT PRINTING OFFICE : 1987 O - 193-757 : QL 3