Bulletin 464

MINERAL-DRESSING CHARACTERISTICS OF THE RED IRON ORES OF BIRMINGHAM, ALA.

By

WILL H. COGHILL and G. DALE COE

FOREWORD

The primary purpose of this paper is to give the results of a recent investigation, by the Bureau of Mines, of the nature of the Red Mountain (Birmingham, Ala.) iron deposits.

To lay a background the authors have, by quoting unstintingly from other publications, traced the development of the Birmingham iron industry. They have, however, tried to refrain from going too far afield in quoting borrowed—though related—material. The layman might be pleased if the nature of the ore or of the beneficiated product were herein correlated with blast-furnace operation. He would like to know the amount of ore, coke, and fluxing stone that is sent daily to the furnace. This would have to be followed up by the grade of each constituent and the grade of the pig iron and slag. Then there is the blast, either natural or conditioned, that might well be measured in tons rather than in cubic feet. And, of course, the effect of change of grade of the constituents on capacity and cost of blast-furnace operation is of interest. The authors have not operated blast furnaces, so they leave that subject to others. And here, just as the discussion of the blast furnace was left in the hands of those who understand it, so is the matter of profit surrendered to those who are trained to cope with it.

From the standpoint of metallurgical recovery, possibly the best practice would be to continue to feed low-grade ore to the blast furnace that works with such phenomenal recovery rather than to practice milling, which will lose 8 percent of the resources. Economically, however, the operator is the best-qualified judge.

Here the object is to show the operator what has been done in the way of beneficiation and indicate a willingness to go with him in new undertakings that spring from this report.

In milling with well-trained crews treating 4,000 tons per day the cost of power, labor, and supplies is put at 75 cents per ton of ore. The concentrates will have 91 percent of the iron and 70 percent of the weight of the feed, making the concentrates cost about $1.00 per ton prepared for the sintering plant. If the ore is 36.0 percent Fe, 11.0 percent CaO, and 23.0 percent insoluble, the concentrates will be 47.0 percent Fe, 9.0 percent CaO, and 11.0 percent insoluble. Only the operator can balance this against his blast-furnace work.

Getting tuned up for milling in the Birmingham district will be a ponderous task. Persons associated with iron-ore mining have been reared in an atmosphere that enabled them to gain superlative training in mining and moving large quantities of material. Only lately have the operators been confronted with the task of securing the services of experienced milling men.

On the other hand, milling is uppermost in the minds of western operators who have to treat metal sulfide ores. In fact, they are so "milling-minded" that they have been guilty of building mills before the mines were sufficiently developed. When mill operators of years' experience are needed, they are as easy to obtain as experienced miners.

In the Birmingham district the number of men that have been brought up in a milling atmosphere is negligible. Because of this condition, a crew has to be trained in each and every step. The cost of this training will make the usual expense of sending "sonny" to college seem small. Possibly an analogy will clarify this point: Suppose two men of good mining experience had to open and operate a mine but were denied the services of workmen who not only were not miners but who had never seen a mine. Such men would not know the difference between a stull and a stope, a crib and a crab, or a winze and a winch. Troubles would come without end, but it is from men of similar unpreparedness that mill crews must be recruited. Many years will pass before milling in the Birmingham district catches up with mining.

CONTENTS

TABLES

ILLUSTRATIONS

MINERAL-DRESSING CHARACTERISTICS OF THE RED IRON ORES OF BIRMINGHAM, ALA.[1]

By WILL H. COGHILL[2] AND G. DALE COE[3]

INTRODUCTION

The scope of this paper is such that it was deemed advisable to group the contents into several main sections. They are:

Section I.   Geography and Geology.
Section II.   Historical Review.
   (a)   Mining and Smelting in Alabama.
   (b)   Milling.
   (c)   Summary of Milling Research.

These two sections are primarily a bibliography and summary of the research on the Birmingham Red Mountain ores up to the middle 1940's.

Section III.   Status of the Birmingham Iron Industry.

This section points out the extent to which the findings of the iron-ore research have been applied on a commercial scale. Also, a very brief description of the dependent and allied industries is included. The value of the Birmingham iron ore is greatly enhanced by the proximity of coal and dolomite. Thus the coal and coke industry and the dolomite industry are closely allied with and to a large extent dependent upon the possibilities of economic utilization of the Red Mountain iron ores.

Section IV.   Beneficiation Investigations of the Red Mountain Iron Ores.

This is a record of the most recent, hitherto unpublished work of the Southern Experiment Station of the Federal Bureau of Mines, Tuscaloosa, Ala., on the chemical and physical composition not only of the Big seam but of most of the iron-bearing rock strata of Red Mountain.

ACKNOWLEDGMENTS

1

SECTION I. GEOGRAPHY AND GEOLOGY

The commercial red iron ores in the Birmingham district are of marine provenance. They crop out along Red Mountain from Sparks Gap (fig. 1) about 3 miles southwest of Bessemer to Red Gap near Trussville about 15 miles northeast of Birmingham, a total distance of about 30 miles. The ridge is about 400 feet high. Nearly all of the mining is in the 12 miles between the southern limits of Bessemer and Birmingham. The beds at the outcrop dip to the southeast at angles varying from 10° to 30° but gradually flatten with depth.

Burchard (1),[4] in 1933, said:

The Birmingham district may be the longest-lived iron-mining district in the United States. * * * At the present rate of production the iron-ore reserves of the Birmingham district should last more than 300 years, but those of present commercial grade in the Lake Superior district appear to be limited to about 30 years. * * *

The abundance and fortunate grouping of the raw materials for iron and steel making, a climate in which out of door work can be carried on throughout the year, a practically unlimited area suitable for manufacturing sites, topography favoring access by railroads, and low-cost water transportation by way of the Black Warrior River to the Gulf of Mexico are among the natural advantages enjoyed by the Birmingham district.

One of the earliest (1897) descriptions of the geology of the Red Mountain ore deposits is given by Smith (2), who said:

The soft ore usually extends from the outcrop or surface for 150 to 200 feet down on the d'p to where the "hard ore" sets in, though it in places extends down for nearly 300 feet. The distance that the "soft ore" extends down on the dip, however, is dependent on the nature or imperviousness of the cover, the "hard ore" occasionally beginning at the very outcrop. The passage from the one ore to the other is sudden, the line extending in points into the other.

The impurities of this ore, other than the carbonate of lime which when in constant and not too large quantities is not considered an impurity, as it is serviceable as a flux, often making the ore self-fluxing, are sandy and clayey materials. Usually, the sandy matter is in rounded grains that are disseminated all through the ore while the clayey matter, small in quantity, is in interstratified shaly seams. The ore is often oölitic or a mere mass of rounded and glazed siliceous grains of various sizes, flattened and cemented together in a ferruginous matrix. It is often very fossiliferous with "crinoidal stems, bryozoa," etc.

The "soft ore" for many years furnished almost all of the ore for the furnace burdens in the Birmingham district but of late years the "hard ore" has rapidly grown in popularity and use, until it now, at times, furnishes as much as 90 percent of the ore burden. This is as it should be, as the "hard ore" is in much larger quantity than the "soft ore."

The middle seam, known as the "Big" or "Red Mountain Seam," is the main seam both in the thickness and, as a general thing, in the quality of its good ore. It has furnished most of the red ore that has been mined in this region. It has been quarried for almost the whole distance of 11 miles from Birmingham to Bessemer. For the greater part of this distance it is in two benches of about 10 feet each in thickness that are separated from each other by an interstratified seam of clayey shales and sandstones from the fraction of an inch to 5 feet in thickness.

All of the ore that is mined between Birmingham and Bessemer is used in the 26 furnaces of the Birmingham district. It is used with a small admixture of limonite, with the exception of in 2 or 3 furnaces where it is used alone.

Eckel (3), in 1910, discussed the genesis of the deposits.

For many years the origin of the oölitic and fossil ores which occur in rocks of Clinton age has been much discussed. Disregarding minor points of difference it may be said that the various theories which have been advanced to account for

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[4] Italic numbers in parentheses refer to items in the references at the end of this report.
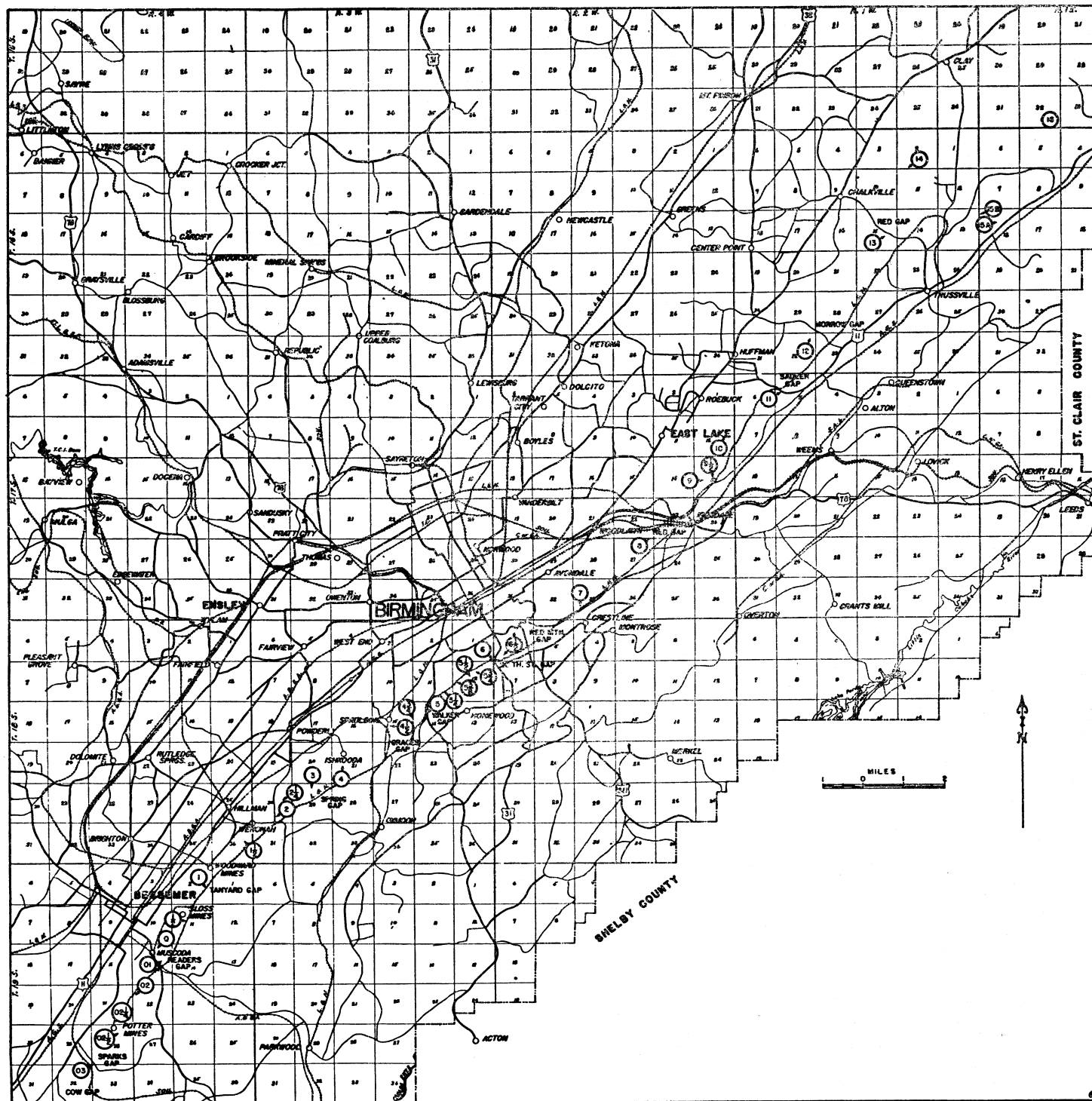
FIGURE 1.—Key to sampling of iron ores on Red Mountain, Jefferson County, Ala. Sampled points marked by circled numbers.

the origin of these ores can be reduced to three. These three opposing theories are, briefly stated, as follows:

1. *Original deposition:* The ores were formed at the same time as the rocks which enclose them, having been deposited in a sea or basin along with the limestone, sandstones, and shales which now accompany them.

2. *Residual enrichment:* The ore beds are merely the weathered outcrops of slightly ferriferous limestones, the lime having been leached out above water level, leaving the insoluble portion of the limestone in a concentrated form.

3. *Replacement:* The ores are of much later origin than their enclosing rocks, having been formed by the replacement of original beds of limestone by iron brought in by percolating waters   *   *   *   .

Examination of a typical specimen of hard oölitic ore, with the information furnished by chemical analysis, shows that the ore is made up of four principal ingredients—iron oxide, lime carbonate, free silica, and clay. Very small percentages of other constituents—iron sulfide, lime phosphate, and magnesium carbonate—also occur, but they may be disregarded at present.

The four principal constituents above named are arranged in a very definite and distinct way in the ore. Most of the silica is present as quartz, in rounded grains of various sizes; each of these grains is surrounded by one or more concentric layers of iron oxide, which is usually mixed with more or less fine clayey matter; while the lime carbonate surrounds and encloses the oölites thus formed. It will be noted that this description, taken by itself, points strongly in the direction of one particular theory of origin of ores, that of original deposition, while it throws great difficulties in the way of accepting the replacement hypothesis. The manner in which the ingredients of the ore are arranged, as above outlined, would seem to admit of but one possible series of events which could give rise to the formation of such an ore. There were, first of all, rounded sand grains and small quartz pebbles; these were next enveloped by a deposit of iron oxide (with some clay); and finally the iron-covered grains were cemented together by a deposit of lime carbonate.

Burchard *(4)*, in speaking about the ore beds being deposited along the shore lines in long, narrow bays, says:

These conditions may perhaps be accounted for by considering that the ore beds were deposited in long narrow bays and lagoons in comparatively shallow water, in which, by means of the sorting action of currents, the sediments were spread along the shore line in such a way that they are homogeneous in character for greater distances parallel to the shore line than normal to it. While the axes of folding may not extend exactly parallel to these ancient shore lines, it is believed that in general these directions coincide fairly closely in the southern Appalachian region. In other words, the axial directions of the folds denote the general directions of the old shore lines.

Burchard *(5)* speaks about the difficulty of separating broken shale from the ore, and styles as an "ore" a bed of ferruginous material carrying 25 percent iron. Certainly events since his writing (1912) would not justify raising this limit. (Are those who have had access to ores of twice this tenor aware that the pioneering days with richer ores will soon be over?) He writes as follows:

By the term "ore" in this report is meant such ferruginous material as may have value either at present or in the near future as a source of iron, whether or not it occurs in quantities sufficient to warrant mining. At present no ores carrying less than 25 percent of metallic iron are intentionally charged alone into blast furnaces, and ores as lean as this can not be used economically unless they carry more than enough lime to flux them and are also used in connection with richer ore. As there are in the southern Appalachians enormous reserves of ore carrying 25 percent or more of iron, it seems hardly necessary in this report to consider as an ore anything leaner than the 25-percent grade. It is, however, difficult to draw a rigid line. For instance, other things being equivalent, it would be more desirable to use as a flux a limestone carrying 15 to 20 percent of ferric oxide (10.5 to 14 percent of metallic iron) than one containing only 3 to 5 percent of ferric oxide, on account of its higher iron yield. Nevertheless, such a ferruginous limestone would not commercially be styled an ore, while a bed carrying 25 percent of metallic iron (35.5 percent of ferric oxide), although in itself hardly rich enough to be used alone for the manufacture of iron, would be conceded to be an

"ore." In view of this commercial distinction material carrying less than **20** percent of metallic iron will not be considered as an iron ore in this report * * *.

The distinguishing characteristic of the red ore is its occurrence in beds or thin lenticular masses of great linear extent, analogous to strata of shale, limestone, and sandstone, and it is interbedded with such rocks.

Regarding the mineralogy, Singewald (6), in 1917, had this to say:

Mineralogically the ores consist of red hematite and include two structural varieties, oölitic and fossil, which usually occur mixed. Usually the nuclei of the oölites are small grains of sand, about which successive layers of iron oxide, and frequently thin layers of silica and aluminous material, have been deposited. The fossil ore consists of fragments of such fossil forms as bryozoans, crinoids, corals, and brachiopods.

In addition to the iron oxide the ores carry varying percentages of calcium carbonate, silica, alumina, magnesium carbonate, and other constituents in minor amounts. Where surface waters have leached out the calcium carbonate the ore is porous and friable, and is known as "soft ore"; in contrast, the unaltered ore is called "hard ore." The "hard ores" naturally form the great bulk of the deposits, and those mined at present range in composition from 32 to 45 percent iron, 5 to 20 percent lime, 2 to 25 percent silica, 2 to 5 percent alumina, 1 to 3 percent magnesia, 0.25 to 1.5 percent phosphorus, from a trace up to 0.5 percent sulfur, and 0.5 to 3 percent water. Actually the ores average well under 40 percent iron, and their value depends chiefly on the relative proportions of lime and silica. Roughly speaking, a self-fluxing ore is one in which the lime is slightly in excess of the silica. As such an ore requires in smelting no addition of limestone, it is equivalent to a higher-grade siliceous ore that requires added flux. If the proportion of lime is in excess of that required for a self-fluxing ore, a quantity of siliceous ore can be added. This is done in the district for the most part by using a mixed charge of limy ore and brown ore.

The great bulk of the red ore of the district is high in silica and comparatively low in lime, so that the addition of considerable flux would be necessary in smelting. The iron content of the ore is so low, however, that the iron content of the furnace charge would be reduced enough to make use of the ores unfeasible. Only those ores, therefore, that are self-fluxing or nearly self-fluxing can be used at present.

How far the ore can depart from the self-fluxing type and still be workable depends entirely on the status of the iron industry. For example, the Upper bench of the Big seam at the Spaulding mine is workable when the demand for iron is good and unworkable when the demand slumps. The average composition of this ore is about 35 to 36 percent iron, 17 to 18 percent silica, and 12 percent lime, and it may be regarded as representing the dividing line between ore that is workable and ore that is not workable under average conditions.

Burchard (1) in 1933, said:

Locally the ore beds are largely composed of fine to coarse grains of silica, coated and cemented with ferric oxide. According as the ore is high or low in lime carbonate it is termed "hard" or "soft" ore. The soft ore has resulted from the leaching by downward-percolating waters of the soluble lime carbonate contained in the original hard ore. This alteration extends down the dip to varying distances that depend on the thickness and permeability of the cover. With the removal of the lime carbonate the relative percentages of the remaining less soluble constituents, mainly iron oxide, silica, and alumina, are increased.

Although the soft ore carried more iron, some of the hard ore has the advantage of containing enough lime to flux the silica that it contains.

Burchard (4), under the caption, Clinton Ore, discusses chemical composition and specific gravity:

In general, the hard and semihard ores used to-day [1910] in the Birmingham district range in percentages of major constituents as follows: Metallic iron, from 32 to 45 percent; lime oxide, from 5 to 20 percent; silica, from 2 to 25 percent; alumina, from 2 to 5 percent; magnesia, from 1 to 3 percent; phosphorus, from 0.25 to 1.5 percent; sulfur, from a trace up to 0.5 percent; and water, from 0.5 to 3 percent. The ore is therefore of non-Bessemer grade. Small quantities of manganese are found in the ore in places. The content of this mineral seldom exceeds 0.25 percent. In the soft ore the lime generally runs less than 1 percent, so that the percentages of the other constituents are proportionately higher.

The Clinton ore exhibits rather wide variation in specific gravity due (a) to variations in composition, and (b) to variations in structure. By experiments with 1-inch cubes and lumps of ore the specific gravities of certain southern Appalachian Clinton ores have been found to range from 2.93 to 3.56. The above figures correspond roughly to weights of from 183 to 225 pounds per cubic foot, and to volumes of 12.25 to 10 cubic feet per long ton.

Phillips (7), after his pioneering work in 1896, comes back in 1912 to tell more about the physical nature of the ores. He gives the reader the benefit of his observations about the size of sand grains in the respective ore seams and does not overlook the value of the intimate commingling of the ore with the lime that is to flux it:

The Ida seam [see table 1, fig. 2] lies from 5 to 30 feet above the Big seam, and the Irondale seam lies from 2 to 3 feet below the Big seam.

There is a remarkable difference in the physical appearance of the ore from these three seams. The Ida ore is fine-grained and closely knit, the siliceous pebbles are much smaller than in either of the other seams and the proportion of the ferruginous cement is larger.

The Irondale seam is not so close grained as the Ida, but finer grained than the Big seam. In respect to coarseness of grain it stands between the Ida and the Big seam. [See table 5 for order of coarseness.]

The siliceous pebbles in the Big seam are some times one-eighth of an inch in diameter, the general size, however, ranging from one-sixteenth to one-thirty-second of an inch. In all of the seams the pebbles are rounded, with a tendency, now and then, to lenticular shapes.

When boiled in acid and freed from adhering oxide of iron the pebbles are opaque. They are sometimes flattened and when so shaped are larger than usual, some of the flattened pebbles being one-half to one-inch in greatest diameter but only about one-eighth of an inch thick. It is occasionally observed that these flattened enclosures carry much more lime than the surrounding ore and that the lime (as carbonate) is well crystallized.

Physically, these three seams of ore are composed of more or less ferruginous pebbles held together by the cement of oxide of iron. By crushing and screening even the lowest grade ore it has been found possible to bring the iron in about 25 percent of the material from 32 percent to 53 percent, provided that the ore has been crushed to pass a 15-mesh screen. * * *

When large experiments were conducted by the writer on the magnetization of the low-grade, soft red ores it was constantly observed that a hard crust was formed on the surface of the pieces, and that the outside would be entirely converted into the magnetic oxide (showing a marked step towards reduction) while the center was comparatively unchanged. In the case of low grade limy ore it was observed that as soon as the carbonic acid came off freely the ore began to fall to pieces and the reduction was carried to the center.

In comparing the two ores another circumstance must not be lost sight of, and this is the intimate commingling of the ore and the lime that is to flux it. It would be impracticable to effect by artificial means such an intimate mixture of ore and lime as Nature has already provided in this material. This circumstance is of the greatest importance in any discussion of the relative value of the two ores, the soft and the hard or limy ore. While this latter ore requires a higher heat for fusion it is not on this account to be considered less easily reducible.

The reducibility of an ore depends far more upon its permeability or porosity than upon its fusing point and the loss of energy in the furnace is, for the most part, chargeable to lack of reducing power, rather than to lack of fusing power. There is such a thing as fusing an ore without reducing it or otherwise materially affecting it, but no ore can be reduced without a profound alteration in its chemical nature.

Cooke (8), in his microscopic study of many iron ores, sets forth their physical nature and includes the Birmingham ore as follows:

The majority of hematite ores are of sedimentary origin and are of economic value to the metallurgist only if enriched by weathering processes. The Lake Superior iron ores, which are chiefly hematite in nature, have resulted from the weathering and local concentration of pre-Cambrian, sedimentary, iron-bearing rocks. * * * Circulating solutions removed the silica and carbonates and

left hydrated ferric oxide, which, under arid conditions and perhaps also pressure, were converted to hematite.

The oölitic hematite deposits of the Clinton formation have a different origin. The consensus of opinion is that these ores have been formed by direct sedimentation. Weathering of ferruginous rocks yielded colloidal ferric hydroxide, which was transported by streams to shallow-water marine lagoons. Precipitation of the colloid around detrital quartz grains yielded oölitic ore, and precipitation around calcareous fossil fragments yielded fossiliferous ore. Others believe that the ores were formed by replacement of either oölitic limestones or calcareous marine organisms by iron-bearing solutions   *   *   *   .

The ore investigated was a hard, oölitic ore from Red Mountain near Birmingham. The following minerals were identified in polished and thin sections: Hematite, quartz, feldspar, calcite, siderite, apatite, and argillaceous material.

The hematite apparently is soft and in a polished section scratches easily, not because of inherent softness but because of the finely crystalline nature of the mineral. An oil-immersion objective shows the larger grains of hematite very clearly. The hematite is essentially micaceous in habit. Some platy hematite occurs scattered throughout the quartz and also in the calcite and siderite. The hematite is colored various shades of red and orange owing to its fine structure and high internal reflection. Some of it is massive enough to exhibit in reflected light its characteristic steely blue color.

The oölites consist of a central core which may be rounded quartz grain, a calcareous fossil skeleton, or an aggregate of specularite needles.   *   *   *

Concentric layers surround the oölitic nuclei; they alternate with layers of siliceous and argillaceous material and occasional layers of calcite.   *   *   *

The quartz and feldspar occur as well-rounded grains, either as cores to the oölites or as fragments cemented in the ore by calcite. The calcite and hematite cement the oölite together.   *   *   *

Diligent search for apatite in the thin sections and in the crushed ore was rewarded by the discovery of one grain. This is insufficient evidence to conclude whether the phosphorus-bearing mineral occurs principally as discrete grains in the cementing material or whether it occurs chiefly as a very fine grained dispersion in the oölites themselves.

Harder *(9)*, in the study of iron ores, recognizes organisms as a factor in their deposition:

As early as 1836 it was found by Ehrenberg that organisms play an important part in the formation of ocherous deposits of bog iron ore. Since that time these organisms have been studied by various investigators, who have determined their position in the classification of lower plant forms, have investigated the life histories of some of them, and have studied their morphology and their physiologic processes. Nearly all of this work, however, has been done from the standpoint of the biologist, and the geologic significance has received only minor consideration.

Harder concludes that:

According to present knowledge, there are three principal groups of iron-depositing bacteria—(1) those that precipitate ferric hydroxide from solutions of ferrous bicarbonate, using the carbon dioxide set free and the available energy of the reaction for their life processes; (2) those that do not require ferrous bicarbonate for their vital processes but that cause the deposition of ferric hydroxide when either inorganic or organic iron salts are present; and (3) those that attack iron salts of organic acids, using the organic acid radicle as food and leaving ferric hydroxide, or basic ferric salts that gradually change to ferric hydroxide.

The Red Mountain formation ranges in thickness from 200 to 300 feet along the outcrop in the Birmingham district. The various beds in geologic sequence have been grouped by Richard W. Smith, mineral technologist, Bureau of Mines, in table 1. A panorama of the outcrop is shown in figure 2.

FIGURE 2.—Red Mountain formation, Walker Gap, Green Springs Highway, Birmingham, Ala.

TABLE 1.—*Generalized section of the Red Mountain formation in the Birmingham district*

Age member	Character	Thickness (feet)
Unconformity		
Niagara (Clinton):		
Upper barren Clinton	Shale and nonferruginous sandstone	23–75
Upper Ferruginous sandstone....	Fine-grained ferruginous sandstone, at places shaly in part..........	0–35
Lower barren Clinton............	Shale and nonferruginous sandstone, occasional thin ferruginous sandstone	10–21
Ida seam	Iron ore and siliceous iron ore.....	1– 7
Middle Ferruginous sandstone...	Fine-grained ferruginous sandstone with alternate beds of shale and nonferruginous sandstone at most places	28–42
Big seam	Limy ore in upper bench, and siliceous in lower benches.........	7–25
Unconformity		
Medina (Albion):		
Irondale seam	Iron ore (at north end), siliceous iron ore, and ferruginous sandstone, somewhat shaly at places.	4–12
Upper barren Brassfield	Nonferruginous sandstone and shale	30–40
Alfretta "seam" (Lower Ferruginous sandstone)	Ferruginous sandstone and thin siliceous ore, shaly at places.......	1–12
Lower barren Brassfield	Nonferruginous sandstone and shale, occasional slightly ferruginous sandstone beds............	25–60

NOTE: The italicized seams are those that are dealt with in this report because of their significant iron content.

SECTION II. HISTORICAL REVIEW

MINING AND SMELTING IN ALABAMA

The beginning of things is always interesting. Rickard *(10)*, in his characteristically lucid manner, mentions the birthplace of the iron industry in North America and records the development of the iron and steel industry of Canada from 1735 to the present:.

The beginning of the iron industry on the North American continent must be credited to John Wilson, Jr., who, in 1646 took bog-iron ore from small lakes of glacial origin near Lynn, in Massachusetts, and smelted the ore with charcoal in little furnaces. It is more than a coincidence that the first smelting of iron in Canada, in 1735, was likewise based upon the use of bog-iron, which, being simply ferric peroxide, is easy to reduce.

Tuomey *(11)*, in touching on the early literature on the iron ores of Alabama, scarcely mentions their mineral-dressing characteristics, but his writings do deserve a place in any bibliography dealing with the industries of Alabama. He began his explorations in Alabama in 1847. He was chosen professor of mineralogy, geology, and agricultural chemistry in the University of Alabama in 1848. His First Biennial Report on the Geology of Alabama bears the date 1850. Those who write bulletins today will be impressed by that prompt performance.

Tuomey goes back as far as 1827 in giving a list of 13 papers on the geology of Alabama. The status of the iron-ore industry is best shown by quoting from two pages of his First Biennial Report.

There are eight bloomeries in the State, two of which are on Talladega Creek, and the others on the waters of the Cahawba.

Of the two high furnaces, one is in Bibb County; it has but recently been erected, so that its operations are, as yet, confined to the manufacture of pig iron and hollow-ware; the blast is urged by steam power, and the boiler is heated directly from the trundle-head. These works are situated within 8 or 10 miles of the Coosa, and from the convenience and good quality of the ore, and the abundance of fuel, they can scarcely fail of success, under ordinary management.

The Benton works are situated on Cane Creek, a short distance from the river; they have been for years in successful operation. An extension of the works, the introduction of the hot blast, and various other improvements are contemplated, which, when accomplished, will place this among the most complete establishments at the South. The following brief statement was politely furnished by one of the proprietors:

"Polkville, Benton Co., Ala., Sept. 26, 1849.

"We have a blast furnace, a puddling furnace, and forge, in operation. We turn out daily 6,000 lb. of iron: 2,000 lb. of which are put into hollow-ware and machinery-castings, 2,000 lb. into bar iron, and 2,000 lb. into pigs. We use 600 bushels of charcoal every 24 hours. Our iron-ore beds (some of them) are within 600 yards of the furnace. Our limestone is at the furnace, and in abundance. The nearest stone coal beds that have been worked, are 13 miles off. We are now preparing to put up a rolling mill, and think that in a short time, we will be able to roll iron successfully. Our establishment is 5 miles east of the Coosa River, opposite the Ten Islands, and 11 miles from Greensport. We ship our iron down the Coosa in flatboats to Wetumpka, Montgomery, and Mobile. We have found the articles we produce here of ready sale in either of those markets. We are prepared to make, turn off, and fit up all kinds of machinery, except fine castings for cotton mills, and will be very soon ready to furnish these."

Tuomey *(12)*, in his Second Annual Report, states:

About that time (1854) the public prints were filled with glowing accounts of mines of silver and copper, of unparalleled richness, discovered in various localities east of the Coosa.

Our first object was to examine these mines * * * . I remained, east of the Coosa, the whole summer, in discharge of the disagreeable duty of endeavoring to repress speculation * * * . Of all the duties that fall to the lot of the geologist this is certainly the least to be desired and that for which he receives the least credit. Very few men think it a favor to have their rich beds of copper transformed into mere iron ore, or their visions of immense wealth dispelled by a single blow of the hammer.

Armes *(13)* in her story associates the first blast furnace with the date of admission of Alabama to the Union:

A blast furnace * * * was built before Alabama was admitted to the Union. Coal-mining operations, brown-ore mining, forges, and furnaces antedate cotton mills.

By the year 1819, when Alabama was admitted to the Union, there was not a community in the State without its blacksmith shop * * * .

Iron used at the start by the frontier smiths was necessarily bar iron and blooms imported from England and Sweden * * * . It was steered up the Tombigbee River from Mobile * * * .

The earliest use of the Alabama iron ore itself was * * * to shoe horses of Andrew Jackson * * * .

In this region (Cahaba) was also a curious and dominating hill of red rock that was to the Indians so precious of color and quality that they traveled for many miles to get it. * * * It was their war paint and of use to them for dyeing.

With the dawning of the nineteenth century the hill of the wine-red rock was found to be a mighty range of iron ore, the greatest ore range of the South. * * * The hill of red ore was early called Red Mountain, and in the year 1871 the city named Birmingham was founded. * * *

Coal had been mined in Tuskaloosa County as far back as 1831, in the vicinity of the State university, and shipped in flatboats to Mobile. It was also sold at the university at 4 cents a bushel. This is one of the earliest dates recorded in coal operations in the State. * * *

Prior to this date, to wit, in 1858–59, Horace Ward had built at his furnace at Shelby a large rolling mill with a capacity of 12 tons of heavy and small-size finished bar iron. This was the first and parent rolling-mill plant for Alabama.

The enterprise was entered into by Mr. Ward in 1858, before the war was really anticipated, and in 1859 he completed the plant, and in December of that year made the first heat in his puddling furnaces. On April 4, 1860, the mill engine was started and all the machinery properly adjusted, and on the 11th of April, 1860, this mill turned out Alabama's first day's product of finished bar iron, the beginning of an era in her history as an iron manufacturing State.

In the winter of 1863 the "Old Roman" furnaces at Oxmoor went into blast, each making 10 tons of charcoal iron per day. At the same time Frank Gilmer opened the Helena coal mines. Milner says: "He sent thousands of tons of coal all over the South, and thousands of tons of Red Mountain pig iron were shot away in shot and shell at Charleston and Mobile." The entire output of the furnaces, "charcoal iron No. 1," was hauled to Selma by one little locomotive, "Willis J. Milner," a little broad-gage wood-burner, named for John T. Milner's father. * * *

In tracing the reconstruction of the iron works at Brierfield (Bibb County) one finds back of the enterprize Brig. Gen. Josiah Gorgas. The property was seized as contraband of war by the United States Government, and was sold at public auction, in January 1866. * * *

From an article describing these iron works, and written in 1868 by an officer of the United States Engineering Corps, these excerpts have been taken:

All of the structures are of the most substantial kind. First, within 100 yards of the railroad is the large rolling mill; within this there are three engines at work, one driving the "muck train," and intended also to drive the "nail plate train," a second which makes bar iron, and a third which pumps water, cuts off iron, and a machine for making buckles for cotton ties. Here are eight puddling furnaces, two heating furnaces, and four boilers supplying steam to the engines. The boilers are placed by the heating furnace, and the steam is made by the waste heat from those furnaces. The machinery all appears to work well, is placed on stone foundation, and is well disposed for work. The puddling furnaces will convert 16 gross tons of pig iron into muck bar in 24 hours, and these are daily converted into 20,000 pounds of bar iron, and 100 kegs of cut nails—the machinery for which is all on the spot, though not yet put up. * * *

The furnace is fed daily with 40 tons of ore, 9 or 10 tons of limestone, broken up small, and 2,500 bushels of charcoal, or, if coke be used, 25 or 30 tons of that, making the large aggregate of about 75 tons of material fed in daily. This is the limit of the capacity of the furnace and makes some 22 tons of iron daily, as the yield is something over 50 percent of the ore used.

On page 46 the author gives a picture under the caption:

Historic Spaulding mine on Red Mountain, where first iron ore in Jefferson County was dug, Republic Iron & Steel Co.

Pumpelly *(14)* gives us a monumental tome containing 1,025 pages with many maps. Pages 383 to 401 are devoted to notes on samples of iron ore collected in Alabama, by W. M. Chauvenet and by Bailey Willis.

The chemical analyses of the iron ores show that the "insoluble" may contain as much as 1.0 percent each of lime, magnesia, potassia, and soda.

Smith *(2)* gives us a valuable history of the iron-ore industry:

The iron-ore industry in Alabama (1897) has made wonderful progress within the last few years. Alabama did not take a prominent position in the list of iron-ore-producing States until 1889 when she jumped to second place, being next to Michigan. She held this second place up to 1894 when there was a decrease in her iron-ore production and she was turned down by Minnesota. She still (1897) holds third place in the list. Unlike, however, the two States that rank her in iron-ore production she smelts almost all her ores instead of shipping them to other States to be smelted. She is now fourth in the list of States in pig-iron production and stands a good chance of being second within a year or two. She has 34 coke furnaces and 12 charcoal furnaces that during 1895 had an output of 854,667 long tons and during 1896 had an output of 922,170 long tons of pig iron. She had invested in iron-ore mining in 1870 only about $40,000 while in 1890 this investment amounted to $5,244,906. Her iron-ore production for 1894, according to Part IV, Mineral Resources of the United States, 1894, was 12.57 percent of all

the iron ores produced in the United States or 1,493,086 long tons with a value of $1,240,895 at $0.83 per long ton. The greatest draw-back to the production of iron ores in this district is that they are too high in phosphorus. * * *

During 1895, according to W. B. Phillips, pig iron was made in Alabama at less than $6.00 per ton. The iron ores of this region therefore owe much of their importance to their most favorable location with respect to the other raw materials for the manufacture of iron. * * *

The ore, being in thick seams and so lying as to permit greatly of the use of gravity in most instances in the mining of it, can be most easily and cheaply mined. During a part of 1895, this ore with 46 percent of iron, was delivered in the stock houses at 50 cents per ton. * * *

MILLING

The argument for concentration hardly needs to be pressed because the district has been forward-looking; as far back as 1895 it was investigating beneficiation. At that time the famous Lake Superior district, instead of producing the present prodigious annual amount of nearly 100 million tons of iron ore, was only beginning production.

MAGNETIC CONCENTRATION

Phillips (15) began his work in roasting and magnetic separation in 1893 but refers to an English patent in 1792 for separating iron ores. Phillips obtained concentration but found that the phosphorus was most intimately mixed with the iron. He writes as follows:

In this paper I propose to give, as briefly as possible, the results already reached in converting a nonmagnetic ore into magnetic ore, and then concentrating it over a magnetic separator of the alternate-polarity type. * * *

During the late winter of 1893, Mr. George B. McCormack, at that time Assistant General Manager of the Tennessee Coal, Iron & Railroad Company, Birmingham, Ala., noticed that a piece of soft, red fossiliferous ore, that had fallen into a coke-oven flue at Pratt mines, and had been subjected, at a full red-heat, to the action of the reducing gases filling the flue, was quite black and resembled magnetite. A trial with a hand-magnet showed that it was magnetic; and after consulting with Mr. Alfred E. Barton, Superintendent of the Ensley furnaces, they concluded to try whether or no such ore could be made magnetic uniformly under the same conditions as produced magnetism in the small piece. Accordingly, they placed several lumps in the flue, and after a while observed that they became highly magnetic, and were capable of being concentrated; that is, on pulverizing, the magnetic portion was easily removed from the sandy portions, and it was possible, in this way, to increase the metallic iron in the ore from 47 percent to over 60 percent. * * * After satisfying themselves that the ore could be magnetized in this way, Messrs. McCormack and Barton secured patents on the process of magnetizing hematite ores and then concentrating them magnetically. * * *

I took charge of the matter for the Tennessee Coal, Iron & Railroad Co. in the spring of 1893, and since that time have done a great deal of work on it, both in the Laboratory and in an experimental plant erected at Ensley, Ala., treating 3,000 pounds of ore at a time; and at Bessemer, Ala., where, for some weeks, we magnetized ore in a Davis-Colby gas-fired kiln holding 110 tons.

After satisfying ourselves as to some of the most important points connected with the process, and concentrating ore from 44 percent of iron to 61 percent, the emergencies of the iron-trade at the time compelled us to forego further work in this direction and to use the kiln for calcining brown ore. During the meantime, however, we are preparing to enter upon the business of magnetizing and concentrating the lower grades of our iron ores upon a much larger scale than heretofore. The plant is being overhauled and enlarged, modifications of the kiln are in progress, and a new separator is being constructed, especially designed for this particular business. * * *

With a full knowledge of the subject and from an acquaintance of several years with the ore situation in the Birmingham district, I have no hesitation in saying that a concentrating-plant of the capacity mentioned above, would not experience any serious difficulty in obtaining low-grade ores suitable for concentration and at a price that would leave a fair profit, for 20 years. * * *

Taking the worst work we have done, 3 tons of ore to 1 ton of concentrates, the raw ore carrying 45 percent of iron and the concentrates 57 percent, the cost of the ore would be 75 cents, and adding 50 cents for other expenses we have a ton of 57-percent ore for $1.25. I mention these figures merely by way of illustration, and not as showing the actual cost of the operation. * * *

The kiln in use is of the Davis-Colby type, modified to suit the requirements of the case. This kiln is intended for gas-firing, the gas going into the combustion-chamber, taking fire there, and then passing through the ore and into a central vertical chamber occupying the center of the kiln, thence into the stack. The kiln is fired with producer gas, made in the Wellman producer. * * *

When the operation was proceeding satisfactorily, we got from the kiln 110 tons of ore per day of 24 hours, and worked in this way for several weeks. * * *

The average content of iron in the ore sent to the separator was 45 percent and of silica 30 percent. The average content of iron in the heads was 58.86 percent and of silica 11.51 percent; in the middlings 51.12 percent of iron and 21 percent of silica. * * *

So far, nothing has been said as to the removal of phosphorus. This element is present in the ore to about 0.30 percent, but it is not removed in the separation. It seems to be present as phosphate of lime, entirely amorphous, and most intimately mixed with the iron. We have not been able to remove it, or even to diminish it to any considerable extent. No matter how finely the ore is ground, the heads still carry more phosphorus than is allowed in Bessemer ore. It can be entirely removed by chemical means, and brought from 0.30 to 0.008 percent at one operation. It has been found that dilute sulfuric acid will dissolve out the phosphorus from the heads without affecting the content of iron seriously, and in this manner heads carrying from 58 percent to 60 percent of iron and 0.008 percent of phosphorus have been prepared.

Wilkens and Nitze *(16)* used Phillips' plant, but in order to practice magnetic separation without roasting they substituted an inclined drier for the magnetic kiln. They mention that the rounded siliceous grains all passed through 10-mesh but believed that their removal was not sufficient. In their report they say:

The richer, soft ores of this district, such as are used in the furnaces, average from 45 to 48 percent in iron and from 30 to 24 percent in insoluble matter. Such ores occur, however, only in few localities, which are limited in extent, and are now almost exhausted. By far the greater portion of the leached ore-beds consists of material running from 35 to 45 percent in iron and from 45 to 30 percent in insoluble matter.

This latter class of ore cannot be used in the furnaces to advantage, and is therefore practically worthless, unless the percentage of iron be raised by concentration, and at the same time the insoluble matter be proportionately decreased.

Structurally, the ores are as a rule fine-grained, the average size of the distinct particles being such as would pass through a 10-mesh screen. On examining the product of separation it is seen that the ore consists of:

    1. Rounded silica grains, which, owing to a coating of iron oxide are found by analysis to contain from 10 to 15 percent of iron.

    2. Rounded grains of more highly ferruginous material, running perhaps 30 percent in iron.

    3. A binding-material of hematite, which in itself carries a varying amount of insoluble matter, depending upon the locality of the ore, fineness of grain, etc.

It was found that a mere elimination of the sand-grains was not sufficient to raise the heads to the desired quality, but that the material of the second group, and in some cases a portion of the more siliceous matrix, had to be separated and classed either with the tails or as middlings, in order to obtain high-grade heads.

The tests on these ores were made under the auspices of the Tennessee Coal, Iron & Railroad Co., and were in direct charge of Dr. William B. Phillips, the consulting chemist of the company, who, for the past 3 years, has made a specialty of the problems involved in the concentration of the Birmingham ores. The plant used was the one at which Dr. Phillips had carried on his experimental work in magnetizing-roasting and concentration, and which was described in his paper on that subject. A small inclined drier was substituted for the magnetizing-kiln, for drying such ores as carried a large amount of moisture. It was found that the ore stored in the stock-house could generally be prepared and treated without

drying. A revolving screen, which divided the material into three sizes (through 8- on 15-, through 15- on 40-, and through 40-mesh), was added. Two Wetherill machines, one of each type, were used for making the separations.

Various working-tests were made on material from a great number of localities, and the results were verified by some 500 analyses made in the laboratory of the company.

Space will not permit of a detailed account and discussion of these results; it is merely intended here to present a general idea of what was accomplished. The previous magnetization-experiments had been made entirely on the richer soft ores, such as are now being used directly in the furnace, and of the composition given above.

Concentration-tests on this material by the Wetherill process gave the following results (calculated on a basis of 100 tons of raw ore) :

	Insoluble, percent	Iron, percent
Original ore	48.03	25.20
57 tons of heads, containing	57.10	13.10
28 tons of middlings, containing	46.20	25.40
15 tons of tails, containing	10.00	70.80

The above results compare most favorably with those previously obtained by the magnetizing-roasting process, particularly in the proportional amount of heads that were produced and the comparatively small percentage of iron carried in the tails.

Tests were also made on the so-called "hard ore," which represents that portion of the ore-bed from which the lime has not been leached. The raw ore of this character, as used at the furnaces, averages: Iron, 35.50; insoluble, 17.50; lime, 16 percent. From this were obtained from 50 to 60 percent in weight of heads, containing: Iron, 48; insoluble, 10.50; lime, 10 percent.

Regarding magnetic concentration Smith (2), in 1897, said:

The term "soft ore" is applied to the leached ore or to the ore with no lime, or very little lime in it, and the term "hard ore" to the unleached or limy ore with 15 percent to 20 percent of lime in it. The "soft ore" carries from 45 percent to 50 percent of metallic iron and from 18 percent to 20 percent of silica, and the "hard ore" from 35 percent to 40 percent of metallic iron and from 8 percent to 10 percent of silica. The two ores have about the same amount of phosphorus, about 0.35 percent. The "soft ore" however can be concentrated by magnetic separators until it carries, as reported, 60 percent of metallic iron and less than 12 percent of silica."

LOG WASHING

Again quoting Burchard (5):

With the gradual depletion of the highest grade of iron-ore reserves in all countries increasing attention is being paid to the possibilities of utilizing lower grades of ore. Beneficiation of iron ore in the Lake Superior district has been accomplished by means of extensive plants for washing, concentrating, roasting, nodulizing, and briquetting ores at various places in Minnesota, Michigan, and Wisconsin.

In the South the best-known process of beneficiating iron ores has been applied to brown ore and consists of crushing the ore, washing it in a log washer, screening the washed material, and picking the oversize on a picking-belt. Why some similar methods of treatment are not more generally applied to the betterment of the shaly grades of red iron is difficult to understand, in view of the success that has attended certain efforts in this direction in Tennessee and Alabama within the last 3 years. In mining a 4-foot bed of "Rockwood" ore with thin shale partings aggregating only 4 inches in thickness, over 8 percent by volume of shale is shot down with the ore, to which must be added more or less roof shale. In many places the total percentage of shale is probably not less than 20 percent, and it is difficult underground to separate this broken shale from the ore; consequently most of it is hauled to the surface, and if not separated at the tipple goes on to the blast furnace. At the mines of the Brown Mining Co., in the Rockwood-Cardiff area, picking tables have been given a practical trial extending over a period of 3 years. The results as gaged by analyses of the picked ores compared with analyses of the ores delivered prior to the installation of the

picking tables are reported to have shown a marked improvement in the ores, although inspection was necessary in order to maintain the improved grade. Gains in the average percentages of metallic iron and decreases in the average percentages of silica are apparent. Similar results are reported from operations at the Crudup mine, in northeastern Alabama, where the ore contains a considerable percentage of shale in the form of irregular seams and nodules.

Interesting experiments have been made recently in a private laboratory at Wilmington, Del., in the concentration of iron ores by the Moxham and du Pont haloid process. In this process the ore is ground to pass 100-mesh screens and is fed into troughs containing haloid solutions of high specific gravity in which the tailings float. Separations of the lighter siliceous impurities from the heavier iron oxide concentrates is thus effected. Three tests were made of siliceous Alabama ore carrying 34.32 percent of iron and 44.80 percent of insoluble matter, and therefore not of workable grade. On treatment this ore yielded concentrates ranging from Fe, 41.10, insoluble, 34.50, to Fe, 53.32, insoluble, 16.90. The richer the concentrates the smaller their quantity. Of the poorer concentrates the yield was 81.2 percent and of the richer, 48.8 percent; but the efficiency in the latter is 97.2 percent, compared to 75.7 percent in the former. It is reported that this process can be carried on economically; and if so, there would appear to be a great opportunity for its application in conserving large quantities of siliceous and shaly red iron ores in the Southern States. * * *

Although none of these special processes have yet been put into commercial application, it is possible that they will some time be commercially successful. It is of interest to know what would be the result of treating ore containing a large proportion of shale interlaminated with the hematite, as ore of this type is common in the vicinity of Chattanooga on both sides of the State line.

Singewald *(17)* credits A. J. Moxham with some of the earliest sink-float work on these ores:

Nearly 15 years ago A. J. Moxham experimented in the separation of silica from these ores by means of "float-and-sink" tests in heavy solutions. He ran three tests on the same ore with solutions of specific gravities 3.06, 3.34, and 3.64, respectively, after grinding the ore to pass through a 100-mesh screen. * * * These experiments are the first application of heavy solutions in the investigation of the problem and are of further interest because of his proposal to utilize them in the commercial concentration of the ores by the Moxham-du Pont haloid process.

SINK-FLOAT CONCENTRATIONS

Singewald *(6)*, in speaking of the importance of mineral dressing, cites an ore with a tenor of 35 to 36 percent iron, 12 percent lime, and 17 to 18 percent silica that is marginal without beneficiation.

For his fractionations of the comminuted material by sink-float he used the Thoulet solution with a specific gravity of 3.0. Some excerpts of his reports are given:

The possible value of the red hematite iron ore of the Southern Appalachian States, if some practicable method of concentrating it could be devised, has long been recognized. The ore is very low grade, and although in enormous quantities, particularly in the States of Alabama, Tennessee, and Georgia, is only at a few points rich enough to work, and even the best of it ranks among the lowest grade iron being mined in the United States. A cheap method of beneficiating this ore would make enormous quantities available and vastly increase the iron-ore resources of the country. However, this low-grade ore in effect constitutes a potential iron reserve, and as long as ample quantities of ore can be obtained elsewhere the question of its utilization does not demand immediate notice.

Where these ores are in part of workable grade, however, a problem of more immediate importance arises which, from considerations of economy and prevention of waste in their mining, ought to be solved or, at least, attacked as quickly as possible. Where these ores are being worked they are usually intimately associated with lower-grade material that in mining is left behind. The difficulty and expense of going into ground broken by old workings makes the recovery of this lower-grade ore practically impossible, and it is lost forever. Success in concentrating the red ores, therefore, would save material that is now being irretrievably lost. The successful large-scale concentration of low-

grade ore in the last few years on the Lake Superior iron ranges should encourage the attack of similar problems in the Birmingham district.

For the purpose of determining the possibility of removing silica and concentrating iron in these ores, 39 samples were taken from various mines lying along Red Mountain between the Ruffner mine on the northeast and the Fossil mines on the southwest; 2 of these samples were from the Ida seam, 3 from the Irondale, 24 represented the lower bench of the Big seam, and the other 10 were from the upper bench; 23 of the 24 lower bench samples were taken along Red Mountain southwest of Graces Gap; that is, in the mines where this ore is being lost. * * *

About a pound of each sample was broken in a small Blake crusher and then crushed in a mortar until it all passed through a 20-mesh screen. A small quantity of the crushed ore was taken as a crude ore sample and the rest was subjected to a screen analysis. * * *

As all the sample was finer than 20-mesh, there were eight sizes of material. The seven coarsest were subjected to concentration, but what was finer than 200-mesh was regarded as slimes and was not treated.

To separate the heavier particles, high in iron, from the lighter particles, low in iron and high in silica, a heavy solution, Thoulet solution of a specific gravity of 3.0, was used. Each size of material was treated separately. It was mixed with some of the liquid in a separating funnel; the heavy particles settled to the bottom and the light particles floated, the two portions being then drawn off separately. In this way each sample yielded seven sizes of concentrates and of tailings, besides the slime.

The bearing of the results of these experiments on probable results of concentration on a commercial scale by wet methods or with magnetic separation will now be considered. The author knows that the method of concentration he used is not feasible on a commercial scale, and its efficiency of separation is not obtainable in a commercial plant. The experiments were made to determine the degree to which the silica might be separated from the iron—that is, granted a perfect separation of the mineral particles, what is the result?

If this result indicates that concentration is feasible, the devising of a commercial method of separation would be encouraged. If, on the other hand, a perfect separation of the mineral particles does not yield a marketable product of sufficient value, there is no use working at separation on a commercial scale. The results obtained in the experiments have demonstrated that, so far as the experiments go, a solution of the problem is possible.

Results obtained in the application of the Moxham-du Pont haloid process to the concentration of low-grade iron ores open the possibility of the commercial application of the method used in this investigation and may render the results of these experiments directly applicable. But while this process is in the experimental stage, it will be better to consider the bearing of these results on separation by ordinary wet methods or by magnetic methods.

Wet methods of separation are based essentially on the differences in specific gravity of the mineral components of an ore. In the author's experiments the ores were crushed to pass 20-mesh, a fineness that is not at all unusual in ore mills. As the iron oxide in the ore tested is much softer than the silica that is to be removed, crushing causes an increase of iron in the finer sizes. So marked is this concentration of the iron that the crude material passing through 200-mesh has practically the same iron content as that obtained by concentrating the material coarser than 200-mesh. This fact is of great importance as it eliminates the slime problem in the use of wet methods. The material finer than 200-mesh may be separated by washing immediately after crushing, allowed to settle, and then added to the concentrate obtained from the rest of the material. The concentrates obtained with the same ratio of concentration will not be of as high grade as those made in the experiments because of less complete separation, but this difficulty could be met by increasing the ratio of concentration a little, thus raising the grade of the concentrate and lowering the iron recovery. As the concentrates from the tests of lower bench ores of the Big seam, for example, represented an iron recovery of 94.1 percent, it is evident that a little lowering of the iron recovery to meet the decreased efficiency of commercial work would still permit a good recovery and give a concentrate of high enough grade. Hence there seems to be no reason to doubt the possibility of obtaining equally good concentrates, although with a lower iron recovery, on a commercial scale by wet methods.

Coghill *(18)*, with the help of H. E. Messmore, undertook to promote the use of acetylene tetrabromide, methylene iodide, and thallium

malonate-formate for sink-float of comminuted ores. At that time (1925) these substances were little known in mineral dressing. The ores available at the time for the experimentation happened to be red ores from Birmingham, Ala. Soon the use of the liquids was mastered, and the investigation was switched to a detailed study of the ores. Hence the report was written on the nature of the ores rather than on the characteristics of the liquids used in the dissection. Some excerpts follow. As the title indicates, this paper treats of the degree of liberation of minerals after grinding. To obtain the supporting data a "tool" not then commonly used by metallurgists was employed. It is the "heavy solution."

Actual ores were employed in order to make the study concrete. The selection was somewhat incidental and happened to embrace the red iron ores of Alabama, so any reader interested in these ores will find some of their properties described. The red ores constituted the material examined and a somewhat new adaptation of heavy solutions was the tool of examination. If either the ores or the tool interest the reader these notes may be of some value. The writer's chief interest was in the rate of liberation of locked grains.

Ores are generally ground preparatory to concentration and as long as the concentrators make suitable products the effectiveness of the grinding is likely to be unquestioned. But when suitable products are difficult to make, an investigation becomes imperative. In such a case a dissection of the ore might be made to find the cause of the trouble. Such a condition of unsuitable products exists in the low-grade red iron ores of Alabama. They are difficult to concentrate. * * *

Concentration machinery has not taken a part in this investigation; the use of it would have been only machinery testing, and should never precede the ore testing which consists of such methods as herein outlined. When the properties of the ores have been gleaned the machinery may be used more effectively.

The red ores chosen for this study of effectiveness of grinding are too low in lime to meet the requirements of the blast furnace. If lime is added the burden would be too great. If silica could be removed an acceptable smelting product would result. * * *

Five representative samples were taken. The average content is 33 percent iron, 13 percent lime, and 23 percent insoluble. The average grade now being smelted is 36 percent iron, 17 percent lime, and 14 percent silica and alumina.

Screen analyses of the five respective ore samples were made. The same method of grinding was applied to each and they were made to pass 14-mesh. The Tyler scale was used and the screening extended to 325-mesh. * * *

Hematite is the softest constituent and an attempt to practice log washing would put the richest part of the ore into the overflow. If any part could be thrown away it would be the coarsest. The iron grades up in the finer sizes, lime does not vary materially, and "insoluble" grades down; the finest is the richest in iron and the leanest in insoluble. The five samples were very much alike in these respects. * * *

The method employed is what has been known as "float-and-sink," but it is better called "sink-and-float," to remove the expression as far as possible from confusion with "flotation."

The method depends almost entirely upon specific gravity of the grains; porosity might give misleading results. The shape of the grain which is important in gravity concentration also has to be determined by other means. Obviously after separating an ore into screened sizes and "sinking-and-floating" each of these to make various specific gravity increments, and having free access to chemical analyses and the microscope, one may determine at what size the ore is sufficiently crushed. * * *

Heavy solutions are often of great help to the chemist when quick and approximate results are required. When the material is not finer than about 150-mesh a separation can be made in a very few minutes. Finer material may be separated by allowing it to stand a few hours. For a long time geologists have used heavy solutions to isolate minerals for study and identification, but their solutions were of small volume because they were very expensive.

One of the most easily obtained heavy solutions for the metallurgist is acetylene tetrabromide with density of 2.95. It is miscible with carbon tetrachloride in all proportions so that a range of density from 2.95 to 1.6 may be had. Acetylene tetrabromide is not, as yet (1926), found in the general market but it may be bought from the Dow Chemical Co., Midland, Mich., for about $1.50 per pound.

For a solution of greater density than the above mentioned, stannic bromide may be used, specific gravity 3.34. It melts at 30° C. A chemist can prepare it at a cost of about 75 cents per lb. It can be diluted with acetylene tetrabromide for densities between 3.34 and 2.95. Antimony tribromide has the same general deportment and an even greater density, 3.7, and was used in the investigation herein described, but a more congenial reagent was desirable. Later on, thallium malonate-formate was used for a density of 4.2. It is a very acceptable liquid. * *

The heaviest grains are gray and the lightest grains with their veneer of hematite on a siliceous background are red; the siliceous grains are like a used "streak plate." This would be a guide to the operator if tables could be employed. From the beginning to the end the "insoluble" in the heaviest sinks is too high to permit a cheerful outlook on concentration. In another way the study lends encouragement because it indicates that a process acceptable for one of the ores would be acceptable for all. * * *

No evidence is available that grinding liberates the clay from the hematite. Fine grinding will, of course, remove some of the veneer of hematite from the old surface of the broken gangue. * * *

The nature of the residue remaining after dissolving the hematite out of the 4.20 sinks bears out the theory that clay permeates the iron grains; the residual grains are of about the same size as before lixiviation, and when broken yield a claylike dust of shells of sundry shapes. * * *

The iron content of the heaviest sinks, shows that if they could be segregated a concentrate of 60-percent grade is all that could be expected. The possible extraction will depend upon the amount of grinding. * * *

The inherent clay and silica amount to about 1 part to 8 parts of iron. * * *

The large portion of the iron contained in increments of intermediate densities makes concentration seem difficult. Only in the finest sizes is the iron abundantly segregated in the heavy and rich grains. * * *

High extraction and high grade of concentrate cannot be obtained without grinding to a size comparable with flotation or cyanide practice. If grade of concentrate be sacrificed the grinding need not be so severe.

In the same year (1927) that Coghill was making his investigation Singewald *(17)* said:

By means of the float-and-sink method it was demonstrated, therefore, that the degree of liberation of iron and silica in the siliceous red hematite, when crushed to pass 20-mesh, was such that with the low ratio of concentration of 1.32 and the high recovery of 94.1, a product could be made that was higher in grade than the average washed brown ore of the district. * * *

In the lack of direct and specific data, the question of cost of concentration may also be approached indirectly. The Mesabi Iron Co., with strong financial support and an officiary including names prominent in the mining industry, was formed to undertake the concentration of taconite of the Mesabi range by fine-grinding, magnetic separation and sintering with a ratio of concentration of three and an extraction of 85 percent. It must be recognized, however, that the conditions there are quite different from the Alabama situation. As an offset to the high ratio of concentration, the per ton cost of mining the Minnesota ore is much less, the ore mineral is magnetite and the concentrate is of much higher grade (63 percent Fe) and very low in phosphorus, and, therefore, of much greater value. (Notwithstanding these advantages, it has not yet been possible to run this plant on a commercial basis.)

Lee, Gandrud, and DeVaney *(19)* practiced magnetic roasting, followed by very fine grinding and wet magnetic log washing in a small continuous plant. Their combined grade and recovery were better than those of other experimenters, but they regarded their operating costs as too high. They write as follows:

The problem of developing a process that would make the high-silica red hematites of the Birmingham district available for use in blast furnaces was studied in anticipation of future need for concentration. The work was begun in July

1923, and ores from a number of points within the Birmingham district have since been studied. A method of magnetic concentration has been investigated which, it is believed, will successfully remove a large part of the gangue and incur only a small loss of iron. * * *

Shipments of iron ore from the Mesabi range, Minnesota, began in 1892; for the year 1901 they exceeded 13,000,000 tons. From about 1900 until a few years ago all active experimental work on Alabama red ores ceased because the output of high-grade Minnesota ores was so large. About ten years ago the Birmingham Ore & Mining Company built two small concentrators at the Helen Bess mine, but the plants burned down after short periods of operation. Attempts were made to concentrate the ore by jigging and tabling, with little success.

At various times individuals have reported new methods of concentration, but as far as can be determined no satisfactory process has been established. * * *

Samples of ore for the investigation were taken at eight points on Red Mountain, from the Ruffner mine northeast of the city of Birmingham to a point over 20 miles southwest. At three of the eight points three or more samples were taken, representing different beds. Chemical analyses of these samples show marked differences in composition; for example, the iron content ranges from 16.7 to 40.9 percent. Some samples represent leached and others unleached ores. The very low grade ores are in reality ferruginous sandstones.

A systematic series of tests of each of the ores was made to allow the results of parallel tests of different grades of ores to be compared. Each sample was crushed to one-fourth inch, and a screen analysis was made with a standard set of Tyler screens down to 100-mesh. * * *

Because concentrating finely crushed ores by gravity was difficult, a magnetic log-washer method was regarded as best-suited to the red ores. The latter are, however, hematites, and therefore not appreciably magnetic; for efficient magnetic concentration they must be converted to the magnetic form (magnetite) by artificial magnetization. For this reason roasting tests of different samples of ore were made under varying conditions of time, temperature, and size of ore. * * *

All the high-silica red ores had similar characteristics, in that they were hard and composed of rounded grains of silica, around which layers of iron oxide and kaolin had been deposited. Virtually no pure hematite can be recovered from such ore by mechanical means; the kaolin was deposited with the hematite, and the two are therefore very closely associated. * * *

Because very fine grinding is necessary for clean liberation of mineral from gangue, gravity concentration is not a suitable method of beneficiating the red iron ores. Magnetic concentration is an acceptable method if the iron mineral is in the magnetic form. As the red ores are in the form of hematite, they must be converted into magnetite by roasting in a reducing atmosphere. * * *

The data obtained from many tests are very good evidence of the fact that there is a limit to the grade of concentrate that can be produced by concentration. This limit is not as high as could be desired, although it is high enough to make a desirable blast-furnace feed. Many of the iron ores of this country can be concentrated into a product containing as high as 68 percent of iron, but the red ores of the Birmingham district have not yielded a concentrate much higher than a 56 percent concentrate, even after crushing to minus 300-mesh.

The impurities which remain in the concentrates and are so intimately mixed with the iron mineral as to be inseparable by any practicable mechanical process of concentration consist mainly of silica, alumina, and lime, the alumina being mainly in the form of the clay mineral kaolinite. Of these three impurities, the amount of alumina which is present in the concentrates produced from representative samples of ore is most persistent. A series of magnetic-tube concentrates produced from 11 representative samples of red ore showed an average of 4 percent of alumina. * * *

With very few exceptions, it seems to be a rule that the red ores of the Birmingham district contain a certain amount of inherent clay which stays with the iron mineral throughout any mechanical concentration process, therefore the concentrates contain as one of their impurities clay material amounting to about 10 units. * * *

Magnetic-tube tests of many samples of red ore showed only fair rejection of phosphorus in the tailing. * * * The percentage of phosphorus rejected ranges from 29.44 to 49.65. Ordinarily, however, 41 to 49 percent is rejected. * * *

Roasting coarse sizes of ore in a reducing atmosphere at 500° to 600° C. was found to be an efficient method of converting the hematite to magnetite.

The finely crushed ore in the form of wet pulp can ordinarily be concentrated in two stages of magnetic concentration. A third stage was necessary in one test. The average high-silica red ore containing 32 to 36 percent of iron can readily be concentrated into a product assaying 51 to 53 percent of iron before sintering, with a recovery of 95 percent of the iron. To accomplish this, 50 to 65 percent of the gangue material is removed in the tailing.

Lee *(20)* pioneered in the flotation of limestone from quartz. The incentive was to recover the lime that was lost in the magnetic log-washing of the lime-bearing iron ore. Since the magnetic roasting had been at a temperature below the calcining temperature of limestone, its floatability had not been impaired:

Certain experiments have demonstrated that the limestone may be floated from the magnetic log-washer tailing and incorporated in the iron concentrate.

If the loss of lime could be prevented, some ores though low in iron and high in insoluble contain enough lime to make their concentrates self-fluxing. Since this loss in the magnetic log-washer can not be avoided, the lime must be recovered from the tailing by some method and added to the magnetic iron concentrate before sintering. Then the resulting concentrate will be self-fluxing. * * *

The tailing from the magnetic log-washer is in excellent physical condition for flotation. The iron ore had been ground in a ball mill to 100-mesh for magnetic concentration. Hence the bond between the lime carbonate and the other constituents of the tailing was well-broken. * * *

The concentrates compare favorably with the limestones utilized at present as flux. The average analysis of the limestone now used in the Birmingham district is as follows: Insoluble 3.5 percent, iron 0.5 percent, and lime 53 percent. Dolomite is used extensively as a flux and averages 1.5 percent insoluble, 0.5 percent iron, 30.3 percent lime, and 20.7 percent MgO. The lime concentrate is somewhat higher in iron than the commercial stone, but the purpose of smelting is to recover iron. Hence this is not an undesirable feature. * * *

Inexpensive reagents are used, hence the quantity necessary for a good separation would not be prohibitive. The amount required as indicated by the tests would not exceed 1.5 pounds per ton of tailing treated. About equal quantities of oleic acid and cresol appear to give the most satisfactory results. At prevailing prices these two reagents cost 1 cent and 18 cents per pound, respectively. * * *

Not only is use made of the available lime for fluxing the gangue, but also the recovery of iron is increased and the sintered concentrate is rendered self-fluxing. Such a product would reduce very rapidly in a blast furnace, and thereby cause a pronounced increase in furnace capacity.

DeVaney *(21)* and associates practiced gravity concentration after rod milling with rods encased in garden hose. The separation was good, but other investigators complained of low grinding capacity. Although it is true that any scuffing or attrition scrubbing scheme has low capacity, here it must be borne in mind that the thick rubber hose had reduced the ultimate weight of the grinding medium so much that significant capacity should not have been expected. If the hose had been replaced by a very thin rubber coating permitting the use of larger rods, scuffing would have been obtained with twice the capacity. They say:

The removable siliceous gangue is in the form of oölitic grains which "shell" out like corn, leaving the hematite as rich high-grade "slime" and as clean granular grains. The "slime concentrate" may be recovered by well-known desliming processes. The oölitic grains and granular hematite are separated on tables.

The method of "shelling" is somewhat novel. The ordinary rod mill is used, but to prevent it from functioning as the usual grinding machine, the rods are covered with rubber: 1-inch rods are slipped into ordinary garden hose. The rubber covering makes a composite rod about 1½ inches in diameter and comparatively light in weight. Thus the innovation is in the use of a rod-mill sheller which liberates the iron and leaves the oölitic quartz grains unbroken. The oölitic grains are polished by the rubber-covered rods and may be rejected by tabling. Tabling will recover the clean hematite, which is too coarse to overflow the desliming devices. * * *

In practice, more than one-half of the iron recovered will be as slime. It is unfortunate that so much iron is slimed, but on the other hand it is very fortunate that the slime does not require subsequent treatment to bring it to grade. The objection to slime is partly compensated by its lime content; the slime contains more lime than the table concentrates.

The operators would be much more fortunate if all the insoluble content of their ores were in the form of oölitic grains. The cleanest possible iron concentrate still retains about 8 percent insoluble, which is so finely disseminated that it approaches the state of chemical combination. * * *

The ultimate concentrate made from the first ore contains 47.6 percent iron, 14.2 percent insoluble, 5.5 percent lime, and contains 87.8 percent of the total iron. The second ore yielded concentrates assaying 48.0 percent iron, 13.5 percent insoluble, 5.1 percent lime, and contained 85.5 percent of the total iron. Thus, useless products have been made marketable.

Gandrud *(22)* and associates went ahead with gravity concentration with one innovation; they trued up the minus 100-mesh slime by table concentration. The tabling of such fine slimy material would hardly be acceptable in a plant. A short review follows:

The ore was crushed dry to 4-mesh and then ground to 14-mesh in a laboratory rod mill, using regular steel rods. The minus 100-mesh material was screened out, dewatered, and set aside temporarily. The minus 14-plus 100-mesh fraction was classified into four products that were tabled on laboratory equipment. The resulting concentrate and tailing were finished products. The middlings from the four table runs were combined and reground in the rod mill to minus 35-mesh and the minus 100-mesh material removed by screening. The resulting minus 35-plus 100-mesh fraction was classified into four products and tabled to give only finished concentrate and tailing. The two minus 100-mesh portions then were tabled separately to give finished concentrate and tailing.

The method of tabling the minus 100-mesh product differed somewhat from the ordinary in that concentrates were removed from two different zones. A granular concentrate was removed, as usual, from the concentrate end of the table while a slime concentrate was carried across the table by the wash water and discharged between the head motion end and the tailing zone. The concentrates were about 48 percent iron, representing a recovery of 85 percent.

DIFFERENTIAL GRINDING

DeVaney *(23)* used small steel punchings in a ball mill for differential grinding and removed about one-half of the iron as a merchantable slime. The granular portion was tabled. Academically the scheme is fine, but the present authors have yet to find a ball mill that is not a reservoir for track spikes and tramp material of all sorts. If such a museum were permitted to accumulate the punchings would soon be pushed out at the mill discharge. He correctly states that the quartz grains will not be reduced. He discusses the differential grinding as follows:

Differential grinding is the grinding of the softer increment at a greater rate than the harder portion. With almost all forms of crushing and grinding there is some differential grinding, but under proper conditions it may be carried to the point where one mineral increment is effectively ground while the other remains virtually unaffected. * * *

A low-cost, efficient method of concentrating the low-grade, siliceous, oölitic iron ores of Alabama has long been sought. * * *

Oölitic iron ores have a peculiar structure consisting essentially of nuclei that are usually quartz surrounded by concentric layers of hematite, siliceous and argillaceous material, and occasionally layers of calcite. Hematite and calcite cement the oölites together. * * *

The fact that the hematite is soft and the quartz oölites are hard seemed to offer distinct possibilities for the use of differential grinding as an aid in the concentration of these ores. * * *

Ball-mill grinding tests have shown that there is a definite size of grinding medium best adapted for a given size and hardness. Larger balls are required to

grind efficiently a hard mineral than ore that is soft. As the size of grinding medium is decreased, the grinding rate for a hard mineral, such as quartz, becomes smaller and smaller until a size is reached at which the grinding rates become nil for all practical purposes. Media of this size may, however, be large enough to grind a soft mineral rather effectively. * * *

The ore chosen for laboratory work was from the Spaulding mine and was considered typical of a large tonnage of siliceous ore in the Birmingham district requiring concentration. The ore had the following analysis: Fe, 36.6 percent; insoluble, 27.7; and CaO, 7.4. The quartz oölites ranged from 10-mesh down; most of them were in the 20- to 48-mesh size, with few finer than 100-mesh. The desired grinding medium was one that would be large enough to grind particles of hematite but too small to grind the quartz oölites. Small steel punchings were given preference because they are cheap and have a better surface : weight ratio than balls. * * *

When ¼-inch punchings were used the slimes contained only 11.5 percent insoluble. This insoluble content is near the minimum that can be expected and indicates that practically none of the quartz oölites were ground to slimes. * * *

In this type of grinding the action of the punchings or small balls is distinctive. To describe the rubbing action, the term "scuffing" has been adopted. The media are large enough to break away the cementing material from the oölites but too small to crush the oölites. The action of the small media is to rub or scuff off the layers of iron oxide, leaving the polished quartz grains uncrushed. * * *

In the "scuffing" tests the ore was stage-crushed in rolls to pass 8-mesh, after which it was deslimed, classified, and tabled. Primary slimes produced in this manner are of concentrate grade, as very few of the quartz oölites are shattered at this size. The granular part of the ore was tabled after classification, and concentrates only were removed. The remainder was then sent to the ball mill and scuffed with ¼-inch punchings, after which it was classified and tabled, giving concentrates, middlings, and tailings. Middlings were returned to the scuffing mill. It is impossible to make low-grade tailings from the original ore, owing to the layers of iron oxide surrounding the quartz nuclei. This is why concentrates only were made in the primary circuit and tailings were not removed until after the material had been thoroughly scuffed. * * *

The classifier slimes were minus 100-mesh and of concentrate grade. The scuffing action of the punchings left the round quartz nuclei of the oölites as uncrushed polished grains from which the iron oxide had been rubbed off and which were readily rejected by tabling. * * * It may be noted that 82.5 percent of the iron was recovered in concentrates assaying 48.7 percent Fe, 12.5 percent insoluble, and 5.2 percent CaO. Of the concentrates, 42 percent was recovered by tables and the remainder as slimes. * * *

If classified ores crushed in the usual manner are tabled, the slimes will be high in silica and must be concentrated. If the scuffing principle is applied, the ore cannot be overground, because the quartz grains will not be reduced after the iron oxides have been scuffed from the nuclei. This means that the grinding circuit does not require careful balance and regulation. If the grinding media are of the proper size, the insoluble content of the slimes will be sufficiently low no matter how severe the grind.

Coghill and Delano (24) did differential grinding with a very small volume of rods in the mill. The performance lay between rod milling and self-grinding. This "between" condition had a defect that was not discovered immediately. Under a slight change in feed rate the deportment in the mill would "flop" from self-grinding to rod-mill grinding and vice versa. This was watched through a large opening in discharge. At one moment the charge would be tumbling with no rods visible (self-grinding), and at another moment rods came to the surface and cascaded in regular rod-mill fashion. When the rods ceased to cascade and the mill "slumped" into self-grinding performance the capacity was impaired. Possibly some operators working with a small volume of grinding medium have had this experience but failed to diagnose the cause because the design of the mill did not permit internal inspection. Because of the softness of the hematite and the selection of a screen with openings that would pass the coarsest .

quartz grains, the critical size was almost negligible. The report shows that this condition made the screening easy:

The work was done on two samples from the Birmingham district. One was an oölitic red ore containing about 35 percent iron; the other was a ferruginous sandstone containing about 25 percent iron, particularly attractive because of its low phosphorus content. In both samples rounded quartz (sand) grains were embedded in an oölitic, "onion skin" matrix of hematite. * * *

The particle size of the sand in the two samples is different. The predominating size of the sand in the red ore is 35-mesh, whereas the predominating size in sandstone is 100-mesh. About half of the ferruginous sandstone is rounded quartz sand. * * *

The hematite that is cemented to the sand grains and persists as an iron stain is beyond recovery by any known physical separation. Hence, such a loss would appear to be proportional to the amount of surface of the sand. Assuming that this theory is correct, a unit weight of the ferruginous sandstone would have about five times as much sand-hematite interface as an equal amount of red iron ore. If the assumption is carried further to consider percentage recovery, it may be shown that for each unit of iron in the ferruginous sandstone there is seven times as much sand surface as in the red ore. On this basis the red ore would be expected to give a higher percentage recovery. However, this assumption does not hold true by tests—the ferruginous sandstone yields as good a recovery as the red iron ore. Hence, the conclusion is reached that the cementation is not so severe in the ferruginous sandstones. They yield a lower-grade tailing in spite of the preponderance of sand surface; the "shelling" of the ferruginous sandstones releases cleaner quartz grains than the red iron ore. * * *

If all the hematite yielded a slime in differential grinding, the recovery could be made to approach closely to 100 percent, but its nature does not permit such an ideal recovery. Part of the hematite is hard, high-grade, and resistant, and for maximum recoveries gravity concentration is necessary. Furthermore, in the red ores oölites high in lime induce a loss by their resistance to treatment. A red ore with a low lime content is expected to be most benefitted by the differential grinding. * * *

The rod milling was done in a laboratory mill 19 inches in diameter and 36 inches long, with rods 1½ inches in diameter; the discharge opening was 8 inches in diameter. The practice was a special type, not the ordinary rod milling with a full load of rods nor was it "self-grinding" with the grinding medium omitted. A full load of rods would have minimized differential grinding, and self-grinding would have yielded too small an output. Hence, a rod volume of only 13 percent (300 pounds) instead of the customary 45 percent was selected.

Ordinarily a rod mill is half full of rods, leaving space for ore only between the interstices of the rods and in the pool, but here the small rod volume leaves space for a great amount of ore, some of the particles of which will be subjected to attrition by tumbling upon each other. The speed of the mill must not be high enough to cataract the rods against the breast of the mill, as such cataracting would crush some of the quartz grains.

The red ore was ground in closed circuit with an 8-mesh screen and the ferruginous sandstone in closed circuit with a 30-mesh screen, these sizes being the top sizes of the sand grains in the respective ores.

Doubtless the first impulse of the operator will be one of alarm about screening an iron ore through 30-mesh, and hence it must be explained that the product is unlike the discharge of ordinary grinding machines. Ordinarily the critical size is in large amounts, but here it is very small; the 20- to 28-mesh size is only 1.0 percent of the whole. The reason for this "anomaly" is that neither coarse hematite nor coarse quartz is present to yield critical fractures. The same observation can be made about the red ore at 8-mesh. Furthermore the sharp and angular grains met usually in ball milling are absent.

By this method much high-grade slime may be removed while leaving a granular product for table concentration.

Patents on milling processes include:

Aldrich *(25)*, who claims a patent for a process for the separation by attrition of mineral aggregates into components differing in hardness, specific gravity, and coefficient of wear:

The herein described step in the process of separating by attrition mineral components having different coefficients of wear, which consists in subjecting the

mineral aggregate in the form of a pulp bath of small particles to a tumbling action, causing the tumbling action to project particles with free flight beyond the axis of the tumbling motion, and so regulating the length of the free fall of the particles into the pulp bath to their weight as substantially to erode only the desired softer component.

As the attrition proceeds the harder particles or grains of lime or silica embedded in the soft iron matrix gradually protrude until they are shelled from the iron ore matrix and thus become free and separate particles which thereafter, due to their hardness or resistance to wear, will not be appreciably reduced by attrition in the mill so as to increase the insoluble matter in the slimes produced and therefore my process produces slimes of maximum richness in iron content.

Schiffman *(26)* and associates have patented a plan for simultaneous grinding and magnetic roasting. They claim as follows:

This invention relates to the concentration of low grade iron ores, such as low grade oxide and carbonate ores, etc., and comprises an improved method of treating such low grade iron ores by a combined roasting and grinding treatment, and particularly a combined reducing and grinding treatment, followed by separation of the treated ore, and particularly magnetic separation, to produce a high grade iron ore or concentrate.   *   *   *

Apparently we secure a very desirable type of breakage and grinding with the roasting, or roasting and reducing, treatment. It is advantageous, in some cases, to subject the ore alternately to reduction and to oxidation while at high temperature.   *   *   *

Preheating may be carried out in separate equipment or in equipment combined with the roasting-grinding equipment. The temperature attained in the preheating treatment may approach that used in the roaster-grinder or may even exceed that in the roaster-grinder. The preheating can advantageously be accomplished with the use of the hot gases escaping from the roaster-grinder, for example, by combustion of the reducing gases escaping from the reducing-grinding operation with additional fuel if required, for example, using additional fuel such as coal, coke, fuel oil, producer gas, coke-oven gas, natural gas, or other similar combustible material.   *   *   *

The temperature to which the ore is heated during the combined roasting-grinding treatment will vary somewhat with the ore and its composition. In general, the temperature will be above 500° and not above 2,000° F. and more advantageously within the range of 900° to 1,500° F. with low-grade iron ores of the character referred to. It is desirable to hold the temperature below that at which sintering takes place, which temperature will vary with different ores and with the character of the gangue present. With the oölitic hematite ores we have investigated, no appreciable sintering has occurred below 1,500° F., and usually temperatures above 2,000° F. were required to show any indication of sintering. * * *

In order to avoid reoxidizing of the ground and reduced ore, this ore should be protected from the atmosphere while hot. Cooling of the ore can readily be accomplished by a limited quenching with water in just sufficient quantity to quickly reduce the temperature to somewhat above atmospheric without saturating with water. If the treated ore is to be subjected to wet separation, the ore from the combined roasting-grinding treatment can be discharged into water at this point. The hot ore can also be cooled, without quenching, in apparatus in which it is protected from atmospheric oxidation.   *   *   *

It is one of the advantages of the process of the present invention that fine grinding to a uniform small size is not necessary, and larger size particles can be effectively separated than where the grinding operation follows the roasting and cooling of the ore. In general it will be desirable to remove from the product particles larger than 20-mesh and to return them to the grinder for reworking; but we have secured a high grade concentrate with good iron recovery from the product of the combined reducing and grinding treatment by subjecting "run-of-mill" material to magnetic separation with no sizing or reworking of the material.   *   *   *

We have found that by sizing the material before separation, a more efficient separation can be obtained on the sized material, and better efficiency secured from each separator by presenting to it a more uniformly sized product.   *   *   *

The hydraulic classifier shown may be of any suitable type which will separate the material into a number of sizes, using either "free settling" or "hindered settling" classifications. The ores may thus be separated, e.g., into a coarser size

which is minus 20 and plus 40, a medium size which is minus 40 and plus 60, and a finer size which is minus 60. * * *

The ore treated contained 34.5 percent iron and 27 percent insoluble. It contained 9.5 percent calcium oxide, 0.34 percent phosphorus, and 0.17 percent manganese. It was crushed to minus ¼ inch, preheated to about 1,050° F. and subjected to a combined reducing and grinding treatment in a ball mill type of apparatus with increase in temperature to about 1,200° F. and using coke oven gas as the reducing atmosphere, the time of treatment being about 25 minutes. The material lost about 7 percent in weight and showed 37.0 percent iron, 29 percent insoluble. This material was cooled to atmospheric temperature in a reducing atmosphere and was then separated by screening using 10-, 20-, 40-, 60-, and 80-mesh screens.

The plus 10 and plus 20 sizes were subjected to dry magnetic separation and the middlings obtained were reground to minus 60-mesh and subjected to wet magnetic separation. The finer sizes were subjected separately to wet magnetic separation. The final concentrate obtained represented 62.4 percent of the roasted ore and contained 53.1 percent iron, 9.7 percent calcium oxide, 0.31 percent phosphorus, 0.08 percent manganese, and 13.6 percent insoluble. The tails obtained represented 34.7 percent of the weight of the roasted ore and contained 11.7 percent iron and 48.3 percent insoluble.

Lee *(27)* describes the concentrator that was the first to be a factor in the production in Birmingham. He says:

In 1936, Republic Steel Corp. started an independent investigation of red iron-ore concentration. Careful consideration was given to roasting and magnetic separation as well as to gravity methods. In the fall of 1940, a 1,000-ton pilot plant was completed and placed in operation at the Spaulding mine, and after 2 years of operation this was expanded into a 3,000-ton plant in the spring of 1943. The expansion is a Defense Plant Corporation project. Daily output is 1,800 tons of concentrate.

A gravity method of concentration is employed. This involves crushing to ⅜-in., grinding to — 8-mesh, desliming, hydraulic classification, tabling, filtering, and sintering. * * *

The question that immediately arises in one's mind is that of cost. How can one afford to mine iron ore and throw away 40 percent of its weight in the concentrating process in order to raise the grade from 34 percent iron in the crude ore, to 47 percent in the concentrate, and to 52 percent in the sinter? The answer is in the saving in freight from the sintering plant to the blast furnaces, the increased pig iron production due to a richer burden, the lower coke consumption per ton of pig iron, the lower slag volumes, and the savings resulting from a more uniform burden of sintered material with its faster reducing rate.

JIGGING

Coe and Coghill *(28)* had met repeated annoyance by the choking of spigots in the hydraulic classifiers, so they adopted the oversize spigot to discharge such miscellaneous material as nuts, wires, and washers.

To retard and control the rate of flow through the oversize spigot they developed the periodic pincer to actuate the opening and closing of a collapsible rubber tubing terminating the orifice. This arrangement permitted the use of a 1½-inch spigot where otherwise a 9/16-inch orifice would have been required.

Coe and Coghill *(29)* believe that they have gone farther with precise jigging than anyone else. This work was prompted by an exigency, not a fancy. Word had been received of a death from poisoning by heavy liquids used to fractionate ores. A substitute was required. The jig proved to be a wonderful fractionater. For instance, coal sized 14- to 20-mesh and iron ore sized 35- to 48-mesh was fractionated with the jig, and the results obtained were more significant than would have been possible by using heavy liquids.

TABLING

Feld and associates *(30)* had an unusual experience in adapting specific gravity to a quick method for finding the iron content of a mill product. They knew that if the gangue minerals had about the same specific gravity and were much lighter than the economic mineral a specific-gravity method should work. They met a snag because they followed the customary practice of drying in the first step. The porosity of some of the ore particles caused trouble. When the drying was left until the last step no more trouble was met.

Workmen in the plant learned how to use this method for random samples when the chemical laboratory was engulfed in routine work. Results were obtained in 15 minutes.

Coghill and associates *(31)* found that the Big seam ore contains about 20 percent of sea sand, the coarsest of which is at 10-mesh. Beneficiation consists of removing this sand. The gravity method now in favor is reduction to 10-mesh and tabling of hydraulically classified increments. In the commercial iron-ore concentrator 88 tables are now in operation.

The irregular shape and preponderance of coarse high-iron-bearing mineral and the near-spherical shape of the sand grains are characteristics that are detrimental in the usual scheme of concentration. This paper deals with a system of gravity concentration which, to a considerable extent, circumvents the detrimental effects. The system has proved its merit in plant operation. It consists of supplementing the present plant flow sheet with concentration (hydraulic) classifiers. A miniature, newly developed nonchoking classifier set at the concentrate-middling corner of the table recovered, from the 18-inch middling zone adjacent to the concentrate, nearly as much concentrate as the table yielded. The concentrate from the classifier was of the same grade and coarser than the table concentrate. A concentration classifier set at the side of the table (at about the dividing line between tailing and middling zones) is effective for truing-up the table tailing. In adopting the small classifiers the tables should be mounted on a base that is a few inches higher than is the common practice, and the middling launder should be adjacent to the tailing launder. By this plan the first tables of a series can be made to produce more and coarser concentrates, and tailings of a lower grade.

CLASSIFICATION

Coghill and associates *(32)* have pointed out that, although the principles of hydraulic classification for preparing feeds for table concentration have always been recognized as irrefutably sound, the literature shows a neglect of research as a guide in the design of classifiers. Their paper gives the results of research.

They found that the manner of withdrawing the spigot products was faulty, that teeter columns were too broad, and that they were too shallow.

Research on classification in an iron-ore concentrator of Birmingham, Ala., disclosed that although the constriction plate, with which the usual hydraulic classifiers are equipped, provides uniform water distribution at the entry to the teeter column it does not assure stable

teeter. The degree of stability depends more upon diameter of the teeter column than upon mode of entry of the hydraulic water.

The essential condition for good classification is stable teeter, which is particularly hard to maintain when classifying coarse, high-gravity particles. In addition to the defects of large diameters the sluicing action of the spigot particles through the constricting orifice militates against stable teeter.

A new classifier was developed that provides the advantages of the small-diameter teeter column and at the same time has a high capacity because of (1) a multiplicity of teeter columns all delivering to the same spigot-product reservoir, (2) deep teeter column, and (3) a two-stage classification treatment. Two of these classifiers installed to run as concentration classifiers, in the plant already referred to, recovered 4 tons per hour of concentrate from about 8 tons per hour of "coarse" table middlings. The overflows from the classifiers were of near-tailing grade, and the classifier efficiency was about 80 percent. An important property of these classifiers was the unusually low water requirement. Results such as these were made possible, in addition to the above-mentioned features, by the use of open-bottom teeter columns subtended by a reservoir from which the accumulated quiescent bed of spigot material was discharged by means of a periodic pincer.

SUMMARY OF MILLING RESEARCH

Briefly, then, the milling research on the red ore has included the following:
(a) Both wet and dry magnetic concentration with and without prior roasting.
(b) Log washing.
(c) Heavy-liquid (sink-float) concentration.
(d) Differential grinding.
(e) Jigging.
(f) Tabling.
(g) Classification.

In addition to tests of the suitability of the ore for concentration by these methods, the research has further demonstrated the importance of certain characteristics of the ore. These are:

1. *Porosity.*—Because of the porosity of the hematite grains there is less difference in the apparent specific gravity of the hematite and gangue than is indicated by measurement of their true specific gravities.

2. *Particle shape.*—The quartz grains are near-spherical, whereas the hematite grains are pock-marked and irregular. This difference in shape tends to make the settling rate of a certain-size grain of hematite more nearly the same as that of the same-size quartz grain than would be expected from the specific-gravity difference. The particle shape and porosity characteristics of the ore combine to make sink-float an unsuitable means of appraising the ore for gravity concentration. Precision batch jigging has been suggested as being more suitable than sink-float testing.

3. *Particle size.*—Unquestionably some of the hematite breaks down more readily than the associated sand grains. As a consequence of this difference in grinding resistance, the very fine size particles (about minus 200-mesh) that result from primary and secondary crushing and from mild grinding are so impoverished of sand as to be of concentrate grade.

4. *Lime.*—The geologists seem to be unanimously of the opinion that as much as possible of the lime contained in the ore should be recovered in the hematite concentrate. Some operators hold this same opinion, whereas one operator verbally stated that he and his associates considered the lime present in the hematite objectionable.

SECTION III. STATUS OF BIRMINGHAM IRON INDUSTRY

Of all the methods of concentration that have been investigated, only gravity concentration by classification and tabling and magnetic concentration following reduction roasting are now in use on a commercial scale in the Birmingham district. No data have been published regarding the reduction roasting-magnetic concentration plant recently put in commercial operation. The gravity-concentration system now practiced commercially is well-described by Lee *(27)*.

Although the ore beneficiated by these two systems is at present a small proportion of the total ore mined, the importance and potentialities of red-ore concentration may be gleaned by considering the largeness of the present Birmingham iron industry.

Birmingham, situated 600 feet above sea level and supporting a population of nearly one-half million people, derives its lifeblood from Red Mountain—wherein are located iron ore and limestone (flux). The third essential material for the production of iron—that is, coal—is abundant on both sides of the mountain. Not only are all of the essential raw materials available in large quantities within a few miles of Birmingham, but the mechanical facilities for converting the ore to metal and the metal to useful shapes also exist, to a large extent, within this same area. For example, the four most active mining companies (Tennessee Coal, Iron & Railroad Co., Woodward Iron Co., Sloss Sheffield Steel & Iron Co., and Republic Steel Corp.) operate a total of 21 blast furnaces, 26 open-hearth furnaces, 3 Bessemer converters, and 864 coke ovens, in addition to numerous forming mills. Alabama's iron making and forming facilities are now (April 1945) essentially the same as recorded in U. S. Steel News, August 1937 (vol. 2, No. 8) and Steel, January 3, 1944 (vol. 114, No. 1, p. 208). That is:

Alabama iron- and steel-making facilities

TENNESSEE COAL, IRON & RAILROAD CO.

Fairfield

3 blast furnaces
9 open-hearth furnaces
1 45-inch blooming mill
1 21-inch billet mill
1 110-inch plate mill
1 24-inch structural mill
1 11-inch merchant mill
1 12-inch cotton tie and hoop mill
1 wire mill
1 cold-reduced tin-plate mill
2 bolt machines
4 nut machines
1 wire works
3 batteries coke ovens (357 ovens)

PRINCIPAL PRODUCTIVE COAL AREAS OF ALABAMA

INCLUDING KEY TO BUREAU OF MINES COAL INVESTIGATIONS MADE AT THE SOUTHERN EXPERIMENT STATION

WARRIOR FIELD

CAHABA FIELD
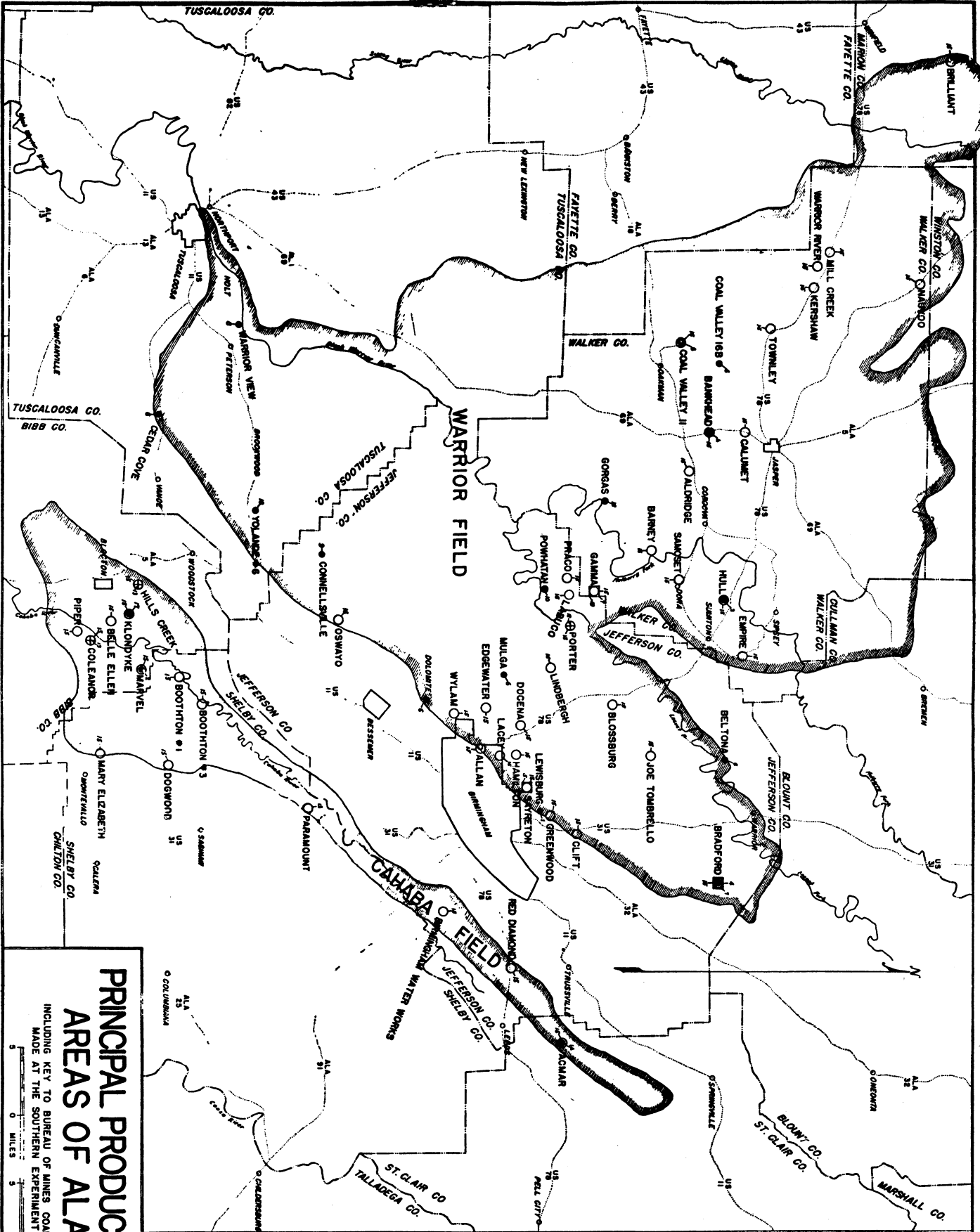
- SYMBOLS -

●	WASHABILITY STUDY
+	JIG OR TABLE TESTS
□	RETREATMENT TEST
○	GRINDABILITY AND FRIABILITY TESTS

BUREAU OF MINES PUBLICATION REFERENCES

REFERENCE NO.	PUBLICATION	DATE
1	R.I. 3012	1930
2	R.I. 3043	1930
3	R.I. 3067	1931
4	R.I. 3083	1931
5	R.I. 3101	1931
6	R.I. 3187	1932
7	R.I. 3185	1932
8	R.I. 3204	1933
9	R.I. 3200	1933
10	R.I. 3170	1932
11	R.I. 3254	1934
12	R.I. 3264	1934
13	R.I. 3262	1934
14	R.I. 3318	1936
15	R.I.3345 & R.I.3364	1936
16	R.I.	1936
17	R.I.	1936
18	R.I.	1936

MILES

Figure 3

Ensley

6 blast furnaces
3 Bessemer converters }
9 basic open hearth   } Duplex process
1 44-inch blooming mill
1 34-inch billet mill

Lipscomb

Blending plant
3 Dwight-Lloyd furnaces

Bessemer

1 rolling mill
1 16-inch bar mill
1 8-inch merchant bar mill

Holt

1 blast furnace

REPUBLIC STEEL CORP.

Alabama City

Steel mill:
  8 stationary open-hearth furnaces, 715,000-ton capacity
Rolling mill:
  1 blooming rolls, 40-inch
  1 bar rolls, 20-inch and 13-inch
  1 bar rolls, 18-inch, 12-inch, 12-inch, and 9-inch
  1 Universal and sheared-plate rolls, 22-inch x 90-inch
  1 jobbing rolls, 30-inch x 60-inch to 84-inch
  1 sheet rolls, 30-inch x 48-inch, 30-inch x 38-inch to 58-inch,
      30-inch x 38-inch to 44-inch
  1 sheet-flattening rolls, 26-inch x 48-inch to 64-inch
Strip furnaces
Rod and wire mills

Thomas

2 blast furnaces
1 Dwight-Lloyd furnace
1 battery coke ovens (57 ovens)

Spaulding

1 concentrator (88 tables)
1 Dwight-Lloyd furnace

Gadsden

2 blast furnaces
2 batteries coke ovens (102 ovens)

WOODWARD IRON CO.

Woodward

3 blast furnaces
2 batteries coke ovens (228 ovens)

SLOSS SHEFFIELD STEEL & IRON CO.

Birmingham

2 blast furnaces
1 battery coke ovens (120 ovens)
2 blast furnaces

North Birmingham

Brown-ore washers at Russellville
                     Ironton
                     LaGrange

It is not enough merely to mention coal and flux in the trinity of raw materials. Coal, in particular, and the coal fields deserve at least a paragraph in a treatise devoted to the iron ores.

The area of the coal fields is enormous as compared with the area of the iron "fields." The accompanying map (fig. 3) shows the Warrior and Cahaba coal fields. Port Birmingham on the Black Warrior

River, 25 miles from Birmingham, affords barge service to Mobile, Ala., and other Gulf of Mexico ports. The best yardstick for magnitude of the area is that Tuscaloosa, in the lower left, is 50 miles from Birmingham, in the middle. By contrast the iron-ore area is small, reaching only from Bessemer to Birmingham and in a narrow strip.

Of course, area is only a part of the story. The iron mines extend much deeper than the coal mines; but the Warrior coal field has many seams of coal, most of which are of coking quality. The most important seams are Pratt, Mary Lee, Black Creek, American, and Brookwood. Others could be added.

Statistical reviews credit Birmingham with producing annually about 15,000,000 tons of coal, one-fourth of which is used for making coke in nearly 1,000 coke ovens. Wet washing is a step preparatory to coking. The washing is done at about 50 plants with many jigs and 100 tables. Alabama leads in the percentage of its coal that is washed (80 percent, approximately). The preparation of 3,000 tons per 8-hour shift is probably the top record.

Dolomite, in the presence of an abundance of limestone, is the popular blast-furnace flux; and well it might be, because it is reputed to contain much less than 1 percent silica, whereas the limestone may have as much as 3 percent. The use of dolomite, however, causes some anxiety lest there might not be enough calcium to carry sulfur into the slag.

The interested reader is referred to the articles by C. H. Johnson (33) and C. E. Abbott (34) for detailed information regarding the Birmingham limestone deposits.

SECTION IV. BENEFICIATION INVESTIGATIONS OF RED MOUNTAIN IRON ORES

The foregoing, due to its generalized nature, may have failed properly to emphasize the fact that of the six recognized iron-bearing strata of Red Mountain the Big seam is of preeminent commercial importance today. Although not of commercial importance at present, some of the other iron-bearing strata have from time to time provoked enough interest to warrant the Southern Experiment Station of the Bureau of Mines appraising them as a possible economic source of iron. There follow chemical and physical data on not only the Big seam ore but also the Upper Ferruginous sandstone, Ida seam, Middle Ferruginous sandstone, and Irondale seam. The beds are shown in panorama in figure 2.

SITE AND DESCRIPTION OF SAMPLES

The sampling (begun in 1941) was under the direction of Richard W. Smith. He prepared a sample map (fig. 1), with key numbers, 03 at the south end to 16 at the north end. Some assistance was rendered by Prof. J. R. Cudworth of the University of Alabama and G. D. Coe of the Bureau of Mines. The key numbers are supplemented by reference to many topographical features known as gaps. Red Gap is the most important from the standpoint of commercial geography because through it communication between Birmingham and the rest of the world is made easy. It lies between Irondale and Gate City and is the most important, since it is the gateway to Birmingham from the

northeast. Through it pass the tracks of all four of the railroads entering Birmingham from that direction.

The gaps that "key" this report are, from south to north, Sparks Gap (03), Readers Gap (01), Tanyard Gap (1), Graces Gap (4½), Walker Gap (5), 20th Street Gap at Vulcan Park (6), Red Mountain Gap (6½), Red Gap (8), Sadler Gap (11), and Morrow Gap (12). Last is Red Gap (13), the name of which infringes upon the one mentioned above as (8). The northwest-southeast highways (and railroads) crossing Red Mountain are the best guides in finding the gaps on the map.

The field record lists 129 character samples in tables 2 and 3. Metallurgical samples (1 to 5 tons) were taken for some of the beneficiation tests. The field record gives significant information, including location, formation, type of ore, and thickness of bed along a 25-mile strip of Red Mountain. All samples were taken with beneficiation in mind and hence are not intended to be used for deductions about ore reserves.

The field record (table 2) is arranged by seams in the same order as table 3. In these tables the geologic sequence has been violated for reasons to be given later.

This report goes far beyond examination of the 129 samples. It covers many hundred samples, and fractions therefrom, examined throughout the work.

TABLE 2—*Field record*

Sample No.	Description
	BIG SEAM
	Potter No. 1, Tennessee Coal, Iron, & Railroad Co.
02½–A	5 ft. badly weathered ore at old slope mouth.
	Raimund No. 1, Republic Steel Co.
02–C	Hard ore, run-of-mine sample taken at tipple from several skiploads. Thickness mined, about 6 ft.
	Sloss No. 2, Sloss-Sheffield Steel & Iron Co.
¾–02–A	9 ft. 8 in. hard ore from hoisting-Y, 500 ft. from slope in 59th right heading, 3,500 ft. down slope under 1,050 to 1,075 ft. cover.
	Ishkooda No. 11, Tennessee Coal, Iron & Railroad Co.
3–E	19 ft. "soft" oölitic ore from road outcrop.
3–F	Upper bench: 8 ft. 5 in. "soft" ore at slope mouth, collected by G. C. Miller, January 1934.
3–G	Lower bench: 12 ft. 2 in. "soft" ore from same place as above.
3–01–A	Upper bench: 7 ft. 10 in. hard ore from 56th heading, about 3,700 ft. down slope under 900 ft. cover, collected by G. C. Miller, January 1934.
3–01–B	Lower bench: 11 ft. 1 in. hard ore from same place as above.
3–02–A	Upper bench: 5 ft. 3 in. hard ore from main haulage at rotary dump, 5,600 ft. from outcrop under 1,200 ft. cover, collected by A. L. Fairley, jr., Mar. 25, 1941.
3–02–B	Lower bench: 10 ft. 3 in. hard ore from same place as above.
	Ishkooda No. 13, Tennessee Coal, Iron & Railroad Co.
4–01–A	Upper bench: 4 ft. 5 in. hard ore from just above Ishkooda fault, 1,500 ft. from outcrop under 650 ft. cover, collected by G. C. Miller, January 1934.
4–01–B	Lower bench: 12 ft. 10 in. hard ore from same place as above.
4–02–A	Upper bench: 4 ft. 5 in. hard, limy, fossiliferous ore from right manway at 18th heading, collected by G. C. Miller, February 1941.
4–02–B	Lower bench: Top 5 ft. hard, fossiliferous ore from same place.
4–02–C	Lower bench: Next 5 ft., hard, oölitic ore from same place.
4–02–D	Lower bench: Bottom 4 ft., hard, oölitic ore from same place.

Ishkooda No. 14, Tennessee Coal, Iron & Railroad Co.

4¼–A	Lower bench: 15 ft. 1 in. pebbly and oölitic ore from outcrop about 200 ft. NE. of slope mouth, collected by G. C. Miller, January 1934.
4¼–01–A	Upper bench: 5 ft. hard, massive ore, from old slope at 14th heading, collected by G. C. Miller, February 1941. (No parting between upper and lower benches.)
4¼–01–B	Lower bench: Top 5 ft. hard, massive ore from same place.
4¼–01–C	Lower bench: Next 5 ft. hard, massive ore from same place.
4¼–01–D	Lower bench: Next 5 ft. hard ore with 3½-in. shale partings omitted, from same place.
4¼–01–E	Lower bench: Bottom 3 ft. 8 in. hard ore, omitting 2½-in. shale partings, from same place.
4¼–02–A	Upper bench: 3 ft. 10 in. hard, limy, fossiliferous, ore, from 27th left heading about 600 ft. from slope, collected by G. C. Miller, February 1941. (No parting between upper and lower benches.)
4¼–02–B	Lower bench: Top 5 ft. hard, fossiliferous, oölitic ore, from same place as above.
4¼–02–C	Lower bench: Next 5 ft. hard, oölitic ore, from same place.
4¼–02–D	Lower bench: Next 5 ft. hard, oölitic ore, from same place.
4¼–02–E	Lower bench: Bottom 3 ft. 6 in. hard, oölitic ore, with few thin shale lenses, from same place.

Spaulding No. 2, Graces Gap, Republic Steel Co.

4½–A	18 ft. hard, oölitic ore, from manway at 34th right heading, 75 feet from slope.
1095–B	Run-of-mine hard, oölitic ore, collected by Republic Steel Co. February 1, 1940.
4½–C	18 ft. 6 in. "soft" ore from outcrop along company road back of new mill.
4½–02–A	Upper bench: 5 ft. hard, limy, coarse, pebbly to oölitic ore, from 31st left heading, 500 ft. from slope.
4½–02–B	Lower bench: Top 5 ft. hard, coarse, oölitic ore, from same place (no parting between upper and lower benches).
4½–02–C	Lower bench: Next 5 ft. hard, oölitic ore, from same place.
4½–02–D	Lower bench: Bottom 4 ft. 6 in. hard, oölitic ore from same place.
4½–03–A	Upper bench: 5 ft. hard, coarse, pebbly to oölitic ore, from 32d right heading, 600 ft. from slope. (No parting between upper and lower benches.)
4½–03–B	Lower bench: Top 5 ft. hard, oölitic ore, from same place.
4½–03–C	Lower bench: Next 5 ft. hard, oölitic ore, from same place.
4½–03–D	Lower bench: Bottom 3 ft. hard, oölitic ore with some shale in bottom foot, from same place.
4½–04–A	Upper bench: 5 ft. hard, oölitic ore, with some limy streaks, from 34th right heading, 200 ft. from slope. No parting between upper and lower benches.
4½–04–B	Lower bench: Top 5 ft. hard, oölitic ore, from same place.
4½–04–C	Lower bench: Bottom 3 ft. hard, oölitic ore, from same place.

Spaulding No. 3, Republic Steel Co.

4¾–A	Top 10 ft. oölitic ore from outcrop at mouth of old slope.

Walker Gap, Cut on Green Springs Road, Republic Steel Co.

5–B	17 ft. 6 in. massive pebbly siliceous ore, from road cut.

Green Springs, Tennessee Coal, Iron & Railroad Co.

5¼–C	Upper bench: 10 ft. 3 in. oölitic ore in old open-cut.
5¼–B	Lower bench: 11 ft. oölitic ore in old open-cut.

Old Mine, 1,200 ft. SW. of Valley View No. 2 Mine, Republic Steel Co.

5⅜–A	Upper bench: 10 ft. oölitic ore from junction of 3 old underground openings.

Valley View No. 2 or Robinson, Republic Steel Co.

5½–B   Upper bench: 11 ft. 2 in. oölitic ore from mouth of old slope.

5½–C   Bottom bench: 3 ft. 8 in. oölitic ore from mouth of old slope.

5½–D   Bottom bench: Bottom 6 ft. 6 in. oölitic ore from cut of tramway.

5½–01–A   Upper bench: Top 7 ft. "semihard" ore from 3d left heading, 200 ft. from slope mouth.

5½–02–A   Upper bench: Bottom 6 ft. "semihard" ore from same place.

Valley View No. 1, Central Iron & Coal Co.

5¾–A   Upper bench: 10 ft. oölitic ore from face at right of slope mouth.

5¾–01–A   Upper bench: 11 ft. 10 in. "semihard," oölitic ore from right rib about 100 ft. from mouth of slope.

5¾–02–A   Upper bench: 10 ft. 6 in. "semihard," oölitic ore from 1st right heading, 40 ft. from slope and about 175 ft. from mouth of slope.

Vulcan Park, 20th Street Gap, Birmingham

6–C   20 ft. ore, top 1 ft. 2 in. fossiliferous, rest oölitic, from highway cut.

Helen Bess, Alabama By-Products Co.

7–B   18 ft. 7 in. siliceous oölitic ore with 1-in. shale parting 7 ft. from bottom, from surface outcrop.

Hammond, Sloss-Sheffield Steel & Iron Co.

8–B   14 ft. 3 in. massive siliceous ore from old tunnel.

IDA SEAM

Raimund No. 3, Republic Steel Co.

02¼–A   3 ft. massive coarse, oölitic ore under 6 in. fine-grained, ferruginous sandstone, total 3 ft. 6 in.

Raimund No. 1, Republic Steel Co.

02–B   3 to 5 ft. coarse-grained, siliceous ore.

Woodward No. 1, Tanyard Gap, Woodward Iron Co.

1–B   3 ft. siliceous, oölitic ore.

Powderly-Oxmoor Road, ⅛ mi. N. of Wenonah No. 9, Tennessee Coal, Iron & Railroad Co.

2–B   2 ft. siliceous, oölitic ore.

Ishkooda No. 11, Tennessee Coal, Iron & Railroad Co.

3–D   1½ ft. coarse, pebbly, siliceous ore.

Walker Gap, Cut of Green Springs Road, Republic Steel Co.

5–D   2 ft. 6 in. siliceous, oölitic ore.

Valley View No. 1, Central Iron & Coal Co.

5¾–C   7 ft. siliceous oölitic ore (Ida) and ferruginous sandstone (Middle).

Henrietta Road and 22d Street, Birmingham

6½–A   1 ft. 10 in. siliceous, oölitic ore, overlying 1 ft. 2 in. ferruginous sandstone, total 3 ft.

Helen Bess, Alabama By-Products Co.

7–D   4 ft. siliceous, oölitic ore.

Hammond, Sloss-Sheffield Steel & Iron Co.

8–C   3 ft. massive, fine-grained, oölitic ore.

Ruffner No. 3, Sloss-Sheffield Steel & Iron Co.

9–A   4 ft. 4 in. siliceous, oölitic ore.

Ruffner No. 1, Sloss-Sheffield Steel & Iron Co.

9½–C   4 ft. 5 in. siliceous, oölitic ore.

MIDDLE FERRUGINOUS SANDSTONE

Woodward No. 1, Tanyard Gap, Woodward Iron Co.

1–C   6 ft. ferruginous sandstone.

Powderly-Oxmoor Road, ⅛ mi. N. of Wenonah No. 9, Tennessee Coal, Iron & Railroad Co.

2–C   1 ft. 6 in. siliceous, oölitic ore (Pentamerous bed), overlying 3 ft. ferruginous sandstone. Total, 4 ft. 6 in.

Ishkooda No. 11, Tennessee Coal, Iron & Railroad Co.

3–C   6 ft. fine-grained, ferruginous sandstone, overlying 1 ft. 6 in. siliceous, oölitic ore (Pentamerous bed). Total, 7 ft. 6 in.

3–B   10 ft. ferruginous sandstone, separated from overlying 3-C (above) by about 15 ft. nonferruginous sandstone and shale.

Walker Gap, Cut of Green Springs Road, Republic Steel Co.

5–E   6 ft. ferruginous sandstone, including Pentamerous bed at top.

Green Springs, Tennessee Coal, Iron & Railroad Co.

5¼–D   21 ft. 2 in. ferruginous sandstone and shale exposed in side of old open-cut, omitting shales over 2 in. thick.

Valley View No. 2 or Robinson, Republic Steel Co.

5½–A   11 ft. 9 in. massive to thin-bedded ferruginous sandstone, omitting several shale beds. Taken above mouth of old slope.

Valley View No. 1, Central Iron & Coal Co.

5¾–B   15 ft. ferruginous sandstone, omitting some interbedded thin shales, from face at right of old slope mouth. Overlies Big seam (5¾-A) and is separated from Ida seam (5¾-C) by covered gap of 12 ft.

5¾–E   23 ft. 8½ in. ferruginous sandstone, not including total of 9 ft. 11½ in. interbedded shales from above Big seam above caved drift on left side of road from tipple to Burgess's house.

Vulcan Park, 20th Street Gap, Birmingham

6–B   23 ft. 6 in. medium-grained to coarse, pebbly, ferruginous sandstone, in road cut.

Henrietta Road and 22d Street, Birmingham

6½–F   1 ft. siliceous ore (Pentamerous bed) overlying 29 ft. 4 in. ferruginous sandstone and shale; total, 30 ft. 4 in. Shale layers over 2 in. thick omitted from sample.

Helen Bess, Birmingham, Alabama By-Products Co.

7–A   15 ft. 9 in. ferruginous sandstone with Pentamerous bed at top, overlain by 4 ft. siliceous ore (Ida seam—see 7–D), total of 19 ft. 9 in. Shale partings omitted, and sample does not include 6-in. shale bed 7 ft. from bottom.

Hammond, Sloss-Sheffield Steel & Iron Co.

8–A   30 ft. coarse-grained, pebbly, ferruginous sandstone, top 5 ft. from mouth of tunnel, rest from in tunnel and very soft.

Ruffner No. 3, Sloss-Sheffield Steel & Iron Co.

9–B   19 ft. coarse- to fine-grained ferruginous sandstone, upper 3 ft. 8 in. containing limy pebbles, at mouth of slope just under Ida seam (9–A).

9–C   Retaking of 9–B, leaving out the upper 3 ft. 8 in. of limy pebbles and including 2 ft. 2 in. below that previously sampled. Total, 17 ft. 6 in.

Ruffner No. 1 Sloss-Sheffield Steel & Iron Co.

9½–D   10 ft. 7 in. fine-grained to pebbly ferruginous sandstone with some limy pebbles, omitting 2-in. shale bed.

9½–B   5 ft. fine-grained siliceous oölitic ore, just under 9½-D.

Shook & Fletcher Mine, Red Gap, 3 mi. NW. of Trussville

13–D   15 ft. ferruginous sandstone with some shale, fairly massive and cross-bedded at top, pebbly at bottom.

UPPER FERRUGINOUS SANDSTONE

Sparks Gap, 4 mi. S. of Bessemer

03–A   8 ft. 8 in. massive, fine-grained, ferruginous sandstone from old prospect opening.

Raimund No. 1, Republic Steel Co.

02-A   14 ft. fine-grained massive to thin-bedded ferruginous sandstone, omitting several shale beds totaling 2 ft. (Beds disturbed by faulting and correlation doubtful.)

Readers Gap, Public Road S. of Muscoda No. 4, Tennessee Coal, Iron & Railroad Co.

01-A   12 ft. thin-bedded somewhat ferruginous sandstone, overlying 3 ft. massive, fossiliferous, ferruginous sandstone weathering soft; total, 15 ft.

01-B   36 ft. massive to thin-bedded ferruginous sandstone, overlying 9 ft. dark, soft, ferruginous sandstone with a few hard beds, overlying 10 ft. massive ferruginous sandstone, overlying 5 ft. soft, black, ferruginous sandstone; total, 60 ft. Is separated from the overlying 01-A by 60 ft. shale and non-ferruginous sandstone.

01-C   12 ft. massive to thin-bedded ferruginous sandstone, directly underlying 01-B.

01-D   11 ft. massive to thin-bedded ferruginous sandstone and thin, interbedded shale. Separated from overlying 01-C by 4 ft. soft, ferruginous sandstone not sampled.

01-1-A   35 ft. "hard," ferruginous sandstone from underground at Muscoda No. 4 mine, material raise at left manway, 6,000 ft. from outcrop and under 1,200 to 1,400 ft. cover; taken by A. L. Fairley, jr.

Ferruginous Sandstone Quarry No. 2, Muscoda No. 5, Tennessee Coal, Iron & Railroad Co.

Q(1025E)   Fine-grained, ferruginous sandstone from bottom 5 ft. of 25-ft. quarry.

Q(1025F)   25 ft. fine-grained ferruginous sandstone from whole thickness in quarry.

Sloss No. 1, Road Across Red Mountain, Sloss-Sheffield Steel & Iron Co.

½-A   21 ft. massive, fine-grained, ferruginous sandstone.

½-B   4 ft. fine-grained, clayey, ferruginous sandstone softer than that above and weathering rough and cobbly. Directly under ½-A.

Woodward No. 1, Tanyard Gap, Woodward Iron Co.

1-A   32 ft. massive, fine-grained, ferruginous sandstone, omitting about 20 in. shale from thin beds near the top and bottom.

Wenonah No. 7, Old Road Over Red Mountain, Tennessee Coal, Iron & Railroad Co.

1 ½-A   30 ft. massive, fine-grained, ferruginous sandstone, rather poorly exposed in old road. May be some shale beds concealed.

Powderly-Oxmoor Road, ⅛ mi. N. of Wenonah No. 9, Tennessee Coal, Iron & Railroad Co.

2-A   21 ft. massive to thin-bedded and cross-bedded ferruginous sandstone, omitting a total of about 18 in. shale in thin partings.

Wenonah No. 10, Tennessee Coal, Iron & Railroad Co.

2 ¼-A   Underground sample of 33 ft. 3 in. hard, fine-grained, ferruginous sandstone, the upper 2 or 3 ft. of the bed not being exposed, taken by A. L. Fairley, jr., from the right manway, just below heading No. 48, 5,600 ft. from the outcrop and under about 1,100 ft. of cover, 83-lb. sample.

Ishkooda No. 11, Tennessee Coal, Iron & Railroad Co.

3-A   35 ft. fine-grained, ferruginous sandstone, some beds not well-exposed, from outcrop along company road.

Ishkooda No. 13, Tennessee Coal, Iron & Railroad Co.

4-A   25 ft. massive to thin-bedded and cross-bedded ferruginous sandstone, from old sandstone quarry east of tipple

Walker Gap, Cut of Green Springs Road, Republic Steel Co.

5-A   24 ft. 6 in. massive to thin-bedded and cross-bedded ferruginous sandstone from road cut, omitting a 2-ft. shaly sandstone 1 ft. from top, a 1½-ft. shale bed 3 ft. from bottom, and several shale partings.

Vulcan Park, 20th Street Gap, Birmingham

6-A   6 ft. weathered, ferruginous sandstone from road cut.

Hammond, Sloss-Sheffield Steel & Iron Co.

Q-D   1 ft. 6 in. fine-grained, ferruginous sandstone from outcrop.

IRONDALE SEAM

Sloss No. 2, Sloss-Sheffield Steel & Iron Co.

¾-01-B   7 ft. hard, ferruginous sandstone and thin seams of limy ore with 6 in. limy pebbles at top ("kidney" bed), from "rock tunnel" on main slope near the 77th left heading, about 6,000 ft. from outcrop.

¾-01-C   1 ft. 8 in. hard, coarse-grained, siliceous, oölitic ore with a few limy streaks, immediately under ¾-01-B at the same place.

Ishkooda No. 11, Tennessee Coal, Iron & Railroad Co.

3-B   7 ft. 7 in. oölitic ore and fine-grained, ferruginous sandstone from outcrop at slope mouth, collected by G. C. Miller, January 1934.

3-01-C   Underground sample of 8 ft. 2 in. hard ore from immediately under 3-01B (Big seam) from 56th heading about 3,700 ft. down slope under about 900 ft. cover, collected by G. C. Miller, January 1934.

3-02-C   Underground sample of 9 ft. hard, limy, ferruginous sandstone with a few shale layers, collected by A. L. Fairley, Jr., from main haulage at rotary dump, 5,600 ft. from outcrop under 1,200 ft. cover.

Ishkooda No. 13, Tennessee Coal, Iron & Railroad Co.

4-01-C   Underground sample of 8 ft. 10 in. hard ore and ferruginous sandstone with 1 ft. 2 in. shale omitted at 6 ft. 6 in. from top, collected by G. C. Miller, January 1934, from just above Ishkooda fault, about 1,500 ft. from outcrop under 650 ft. cover.

4-02-E   Underground sample of 6 ft. hard, ferruginous sandstone and limy ore, not including 2 shale beds 4 in. and 1 ft. 2 in. thick, collected by G. C. Miller from right manway at 18th heading.

Ishkooda No. 14, Tennessee Coal, Iron & Railroad Co.

4¼-01-F   Underground sample of 5 ft. 4 in. hard, ferruginous sandstone and lean, limy ore, not including 1 ft. 7 in. shaly sandstone and shale in middle, collected by G. C. Miller from old slope at 14th heading.

Spaulding No. 2, Graces Gap, Republic Steel Co.

4½-B   Underground sample of 6 ft. 11 in. hard, ferruginous sandstone from slope 2,495 ft. from outcrop, directly under Big seam.

Walker Gap, Cut of Green Springs Road, Republic Steel Co.

5-C   6 ft. 5 in. fine- to coarse-grained limy ore and ferruginous sandstone with 1 in. "kidney" bed at top, omitting 4 in. shale and 4 in. limy sandstone, from road outcrop.

Green Springs, Tennessee Coal, Iron & Railroad Co.

5¼-A   6 ft. 6 in. siliceous ore, including "kidney" beds at top, and 3 ft. ferruginous sandstone above "kidney" bed, total of 9 ft. 6 in., from tram cut to old open-cut mine.

5¼-A-1   7 ft. 6 in. siliceous ore, resampling of above from under "kidney" bed down to a shale parting.

Valley View No. 2 or Robinson, Republic Steel Co.

5½-E   7 ft. 9 in. siliceous ore and thin-bedded, ferruginous sandstone, from tram cut from old slope mouth to tipple.

Valley View No. 1, Central Iron & Coal Co.

5¾-D   6 ft. 9 in. shaly ore and ferruginous sandstone, from below Big seam in caved drift on left side of road from hoist house to Burgess's house.

Vulcan Park, 20th Street Gap, Birmingham

6–D   4 ft. 9 in. soft, siliceous, fine-grained ore, from road cut.

Helen Bess, Alabama By-Products Co.

7–C   5 ft. oölitic ore with 5-in. shale parting in middle not sampled. Has been mined by open-cut and drift.

Ruffner No. 2, Sloss-Sheffield Steel & Iron Co.

10–A   4 ft. "soft" oölitic ore from tunnel at left rib of Roebuck slope.
10–B   4 ft. "hard" oölitic ore from right rib of main slope at 14th right heading. This was under water from April 1928 until mine was reopened in April 1940.

Sadlers Gap, U. S. Highway No. 11

11–A   4 ft. 6 in. soft, weathered, oölitic ore from railroad cut about 450 ft. west of highway, 4 in. badly weathered ore at bottom not exposed.

Smith Brothers, ¼ mi. S. of Morrow Gap

12–A   5 ft. soft, oölitic ore, mined in open-cut.

Shook & Fletcher, Red Gap, 3 mi. NW. of Trussville

13–B   1 ft. 5 in. very soft, fossiliferous ore, mined as "upper bench" in open-cut.
13–A   2 ft. 3 in. harder "soft" ore immediately under 13-B, mined as "lower bench" in open-cut.

PROXIMATE CHEMICAL AND PHYSICAL DATA

The coal industry has employed the terms "proximate" and "ulti-mate" to describe the constituents for which a sample has been ana-lyzed. The authors herein borrow the term "proximate" and use it loosely and arbitrarily to denote certain essential tests to which the iron ores have been subjected. The proximate chemical analysis includes Fe, CaO, insoluble, and phosphorus. The proximate physical tests include crumbling rate and, most important, size analysis of the sea sand. More comprehensive chemical and physical data will be presented later, but to list the essential differences of the ore sample in the briefest form possible the proximate data are useful. These are contained in table 3.

The chemical analyses for Fe, CaO, insoluble, and phosphorus fol-lowed the standard methods of the steel companies and are reported on the dry, moisture-free basis. The insoluble represents material insol-uble in hydrochloric acid and includes sand and part of the inherent clay.[5]

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[5] The amount of clay that may remain undissolved is indicated by the following data, which apply to a combined gravity concentration-flotation pilot-plant test of Big seam ore (given in percent):

	Fe	Insol.	SiO₂	Al₂O₃ in insol.	Total, Al₂O₃
Table concentrate	51.6	11.0	7.82	1.84	3.99
Hydro separator slimes	46.3	12.2	8.88	1.90	3.81
Table tailings	9.9	65.6	62.96	.80	.74
Flotation concentrate	40.0	13.0	10.14	1.30	3.23
Flotation SiO₂ rejects	6.0	73.4	72.84	.40	.45

According to these results, it appears that about one-third of the alumina is contained in a compound insoluble in concentrated hydrochloric acid, and approximately two-thirds of the alumina is insoluble. As the tailings and "rejects" contain large quantities of quartz, the above rule does not apply to them.

TABLE 3.—*Proximate chemical and physical data on the iron ores of Red Mountain in the Birmingham district of Alabama*

BIG SEAM

Sample No.	Wt., per cent	8/10	10/14	14/20	20/28	28/35	35/48	48/65	65/100	100/150	150/200	-200	Mean size, mesh	Fe	CaO	Insol.	Phos.	s or U[1]	Crumbling rate	Sample No.
02-1/2-A	13	1.7	8.1	7.4	6.3	6.6	4.3	3.2	11.4	13.4	8.4	22.2	80	53.1	Tr.	22.3	0.07	s	215	02-1/2-A
02-C	13	1.6	7.1	10.6	8.1	7.3	6.0	4.6	7.5	5.6	5.4	36.2	80	32.7	18.3	17.8	.25	U	77	02-C
3/4-02-A	6		10.0	9.9	11.0	9.9	9.6	5.5	5.3	3.2	1.0	34.6	65	35.1	19.1	11.2	.28	U	52	3/4-02-A
3-E	16		3.5	12.1	13.3	23.2	12.7	6.3	3.4	2.0	2.7	12.9	43	33.9	8.6	23.5	.32	s	165	3-E
3-F	7		1.0	8.5	13.3	15.8	17.0	6.4	5.4	3.0	5.1	24.5	65	39.7	15.1	23.6	.33	s	--	3-F
3-G	20		4.5	12.6	14.7	21.1	11.9	5.0	4.8	2.3	2.1	12.3	42	39.4	6.3	27.7	.39	s	--	3-G
3-01-A	7		5.2	12.2	12.6	13.1	12.5	5.2	4.8	3.3	4.2	26.8	65	37.6	16.9	21.1	.31	U	--	3-01-A
3-01-B	21		3.1	17.0	23.8	19.8	13.5	5.6	4.3	2.2	2.5	8.2	43	35.2	11.3	15.6	.36	U	--	3-01-B
3-02-A	10	1.5	5.7	10.6	13.3	13.2	11.7	5.6	8.1	5.3	1.9	23.2	60	40.1	15.9	15.3	.32	U	104	3-02-A
3-02-B	20		2.9	14.2	19.8	17.8	14.1	7.0	5.9	2.9	2.9	11.3	35	36.1	10.1	25.3	.31	U	75	3-02-B
4-01-A	9	1.0	5.7	18.1	18.1	13.5	10.5	7.0	5.5	3.0	3.5	12.4	43	35.1	18.7	14.9	.31	U	--	4-01-A
4-01-B	15		3.8	12.3	18.1	18.6	14.4	7.2	5.9	1.8	3.6	20.4	37	39.2	9.7	22.9	.37	U	--	4-01-B
4-02-A	8	1.1	7.0	14.5	16.1	15.4	11.4	5.3	4.9	3.0	3.8	18.1	43	32.6	20.4	12.8	.23	U	79	4-02-A
4-02-B	15	.7	8.4	14.1	18.6	15.8	13.0	6.1	5.9	2.7	4.2	18.1	43	39.2	8.7	20.6	.37	U	95	4-02-B
4-02-C	23		4.1	14.8	24.2	22.9	12.6	6.1	4.9	2.2	2.0	6.5	42	40.7	20.4	28.0	.30	U	193	4-02-C
4-02-D	27		3.2	15.0	24.1	23.2	12.0	6.1	4.8	2.2	2.0	10.8	42	34.1	10.4	28.0	.28	U	107	4-02-D
4-1/4-A	25	5.4	9.0	19.1	24.5	20.0	10.3	4.3	4.1	1.7	1.7	6.2	35	32.5	9.4	32.4	.35	s	--	4-1/4-A
4-1/4-01-A	8		4.8	14.7	18.8	13.6	11.6	4.1	3.9	2.7	3.3	22.6	48	35.3	18.1	13.0	.28	U	139	4-1/4-01-A
4-1/4-01-B	13		4.9	14.7	25.4	19.3	13.2	7.0	5.9	2.7	2.7	11.5	43	39.9	9.8	19.8	.38	U	133	4-1/4-01-B
4-1/4-01-C	21		4.2	13.8	23.1	24.8	14.0	6.2	4.9	2.3	2.1	7.6	42	36.5	9.1	26.8	.32	U	183	4-1/4-01-C
4-1/4-01-D	25		3.2	13.8	21.1	25.5	13.2	4.7	3.6	1.7	1.7	10.0	43	33.8	9.3	28.8	.28	U	177	4-1/4-01-D
4-1/4-01-E	24		.5	10.7	8.5	20.9	14.7	5.3	4.5	2.6	2.0	12.9	65	30.3	11.6	32.2	.29	U	162	4-1/4-01-E
4-1/4-02-A	6		4.1	14.9	15.3	15.2	20.9	5.8	6.7	4.4	4.8	19.5	65	34.1	21.8	16.0	.26	U	109	4-1/4-02-A
4-1/4-02-B	10	2.7	4.9	15.0	15.3	18.2	11.3	5.8	4.9	2.6	3.9	19.2	43	37.2	15.0	16.0	.33	U	105	4-1/4-02-B
4-1/4-02-C	19		5.0	9.9	19.2	19.8	14.4	6.8	5.4	2.8	2.6	8.8	43	34.6	11.4	26.8	.32	U	126	4-1/4-02-C
4-1/4-02-D	22		2.7	9.4	19.2	22.5	15.8	7.6	6.4	3.0	2.7	8.8	43	32.8	9.2	30.7	.32	U	158	4-1/4-02-D
4-1/4-02-E	24		2.3	11.3	21.0	24.3	14.6	7.6	4.8	3.0	2.4	10.9	43	32.8	10.7	30.1	.30	U	110	4-1/4-02-E
4-1/2-A	20	.7	9.3	16.7	20.4	21.8	12.1	6.4	4.5	1.7	1.7	9.3	35	34.7	10.3	25.4	.34	s	135	4-1/2-A
1085-B	19	.7	4.4	14.3	20.8	21.8	14.0	4.7	3.7	2.3	1.8	8.9	42	37.1	10.3	28.9	.35	s	207	1085-B
4-1/2-C	9	.5	4.5	15.0	15.3	17.3	12.9	5.6	4.7	2.8	2.8	9.2	43	45.4	15.4	31.2	.17	s	177	4-1/2-C
4-1/2-02-A	23	1.4	5.7	13.1	15.3	19.6	13.1	4.5	4.5	2.7	2.8	16.5	43	38.2	9.2	15.5	.32	U	195	4-1/2-02-A
4-1/2-02-B	25		4.9	17.9	26.0	19.6	14.7	7.6	5.8	3.6	2.8	6.7	42	32.7	10.2	25.5	.33	U	173	4-1/2-02-B
4-1/2-02-C	25		3.2	14.3	24.5	22.1	13.6	7.3	3.1	1.8	1.5	5.6	35	31.9	10.9	31.9	.30	U	189	4-1/2-02-C
4-1/2-02-D	13	1.4	4.6	13.6	19.6	24.7	13.0	4.6	3.3	1.8	1.5	7.7	42	31.3	10.9	32.0	.30	U	137	4-1/2-02-D
4-1/2-03-A	22		4.2	14.1	21.5	16.3	13.6	6.5	5.4	3.4	3.1	11.0	42	39.0	11.3	20.7	.39	U	98	4-1/2-03-A
4-1/2-03-B	22	.6	1.6	12.6	20.6	24.6	16.6	6.3	5.0	2.6	2.8	8.0	42	35.6	9.8	27.8	.33	U	141	4-1/2-03-B
4-1/2-03-C	27		3.4	11.5	20.6	24.6	16.6	4.5	4.8	2.1	1.8	8.5	42	33.6	9.3	30.6	.30	U	146	4-1/2-03-C
4-1/2-03-D	19	.6	4.9	18.2	23.0	23.4	11.9	4.5	6.3	1.7	1.7	6.7	42	29.5	9.2	35.8	.28	U	110	4-1/2-03-D
4-1/2-04-A	19	.8	3.4	11.5	20.4	18.5	16.0	8.2	6.3	2.8	3.1	8.5	42	34.8	10.2	28.8	.38	U	148	4-1/2-04-A
4-1/2-04-B	20	.8	3.6	11.4	19.0	22.1	14.8	6.5	5.2	3.0	2.6	11.0	43	34.2	10.2	29.0	.33	U	132	4-1/2-04-B

Sample No.	Sample¹																No.
4-1/2-04-C	109	U	.27	32.1	14.6	26.4	42	7.8	2.0	2.3	4.1	5.1	12.7	24.4	28.1	15.4	3.1
4-3/4-A	266	S	.08	31.5	Tr.	45.8	35	5.7	1.6	1.7	3.1	5.3	12.6	20.8	21.7	19.4	7.5
5-B	262	S	.33	29.6	9.9	35.5	35	5.9	1.4	1.5	2.8	4.5	12.0	21.4	22.6	20.8	6.6
5-1/4-C	182	S	.33	35.7	13.7	25.4	42	4.9	1.6	2.1	4.2	4.3	13.0	22.3	22.0	15.1	6.6
5-1/4-B	227	S	.33	34.7	6.6	34.6	35	5.8	1.6	2.1	4.1	5.1	16.4	23.4	22.0	15.1	4.3
5-3/8-A	188	S	.34	27.3	6.1	38.5	42	6.5	2.2	2.8	4.5	6.9	14.6	20.4	20.9	18.5	7.8
5-1/2-B	223	S	.29	33.3	1.8	41.1	35	4.9	1.7	2.0	5.3	5.4	13.9	21.4	20.7	18.1	7.6
5-1/2-C	143	S	.22	35.3	9.	41.1	35	4.9	1.9	2.0	4.0	5.4	13.0	21.4	21.2	16.1	5.7
5-1/2-D	222	U	.35	43.4	.4	36.1	35	7.4	1.9	1.8	3.9	4.7	12.8	24.1	21.2	16.3	6.8
5-1/2-01-A	157	S	.32	27.9	9.5	36.4	35	7.0	1.9	3.1	4.2	6.7	13.3	20.1	19.3	12.2	6.8
5-1/2-02-A	262	S	.35	30.1	8.5	35.1	35	7.0	2.3	2.0	5.1	4.6	15.2	20.3	18.0	19.9	8.5
5-3/4-A	193	U	.32	32.0	6.5	44.2	35	3.7	2.4	2.5	4.4	5.8	14.0	23.4	20.3	21.3	8.4
5-3/4-01-A	214	S	.35	29.2	7.4	37.6	35	4.7	1.5	2.3	3.6	4.5	13.5	18.6	20.3	15.9	10.7
5-3/4-02-A	193	U	.12	28.1	5.2	37.1	42	6.2	2.0	1.7	2.9	5.2	14.2	19.2	18.7	16.7	7.9
6-C	325	S	.32	39.0	.3	41.1	35	4.0	2.8	3.3	5.3	4.9	13.5	21.7	18.7	16.7	5.9
7-B	294	S	.17	41.9		32.4		10.4	2.8	3.3	5.3	4.9	12.8	19.7			8.0
8-B	278	S		49.0		33.8		5.6	2.9	2.3	3.8	4.9	12.8	20.6	21.0	17.5	

¹ S, Surface; U, underground.

IDA SEAM

Sample No.	Sample¹																No.
02-1/4-A	240	S	0.030	45.4	0.1	36.9	48	6.0	3.8	3.8	8.1	20.3	8.6	15.2	17.5	9.7	6.0
02-B	315	S	.031	42.7	Tr.	39.2	42	4.6	4.4	1.9	7.4	20.0	9.4	13.6	18.0	13.5	8.9
1-B	194	S	.088	34.0		43.7	60	8.2	2.4	2.6	7.9	32.3	4.2	8.3	10.2	18.4	3.8
2-B	209	S	.037	38.0	Tr.	41.3	48	10.2	2.4	1.6	4.4	25.0	16.0	6.6	9.6	13.1	7.8
3-D	246	S	.192	38.6	.4	41.3	48	12.1	2.0	1.8	5.1	22.3	21.0	6.9	9.9	14.8	7.0
5-D	236	S	.100	34.7	.	43.6	42	14.1	1.4	1.2	3.1	19.3	19.8	4.6	14.1	10.3	8.2
5-3/4-C	250	S	.068	39.3	.1	41.6	35	3.9	1.1	2.2	7.5	31.8	23.9	7.4	6.1	10.7	8.1
6-1/2-A		S	.041	38.5	.	41.3	43	6.6	1.8	2.3	5.9	25.2	26.3	9.9	11.2	6.5	3.1
7-D	134	S	.110	38.0	Tr.	41.3	48	6.9	1.5	2.7	3.7	19.1	31.5	15.2	12.9	15.1	2.6
8-C	157	S	.155	31.8	.2	44.7	6u	19.4	1.2	2.7	4.0	7.0	35.9	28.6	22.4	6.5	5.7
9-A	292	S	.120	51.4	Tr.	32.5	35	10.1	1.0	1.2	2.4	2.9	11.7	24.5	22.4	15.1	.9
9-1/2-C	297	S	.150	51.0	.3	31.8	35	7.6	.9	.9	2.6	4.6	10.1	25.4	22.2	16.3	8.4

MIDDLE FERRUGINOUS SEAM

Sample No.	Sample¹																No.
1-C	237	S	0.058	53.7	Tr.	30.0	100	6.0	4.2	15.9	36.6	22.5	10.9	2.8	0.7	0.3	0.1
2-C	337	S	.031	57.1	Tr.	27.5	115	31.3	4.7	3.7	16.1	37.9	4.9	.	22.8	.1	.3
3-C	261	S	.086	56.1	0.2	22.5	100	15.9	4.8	7.3	22.1	35.4	8.1	.2	2.8	1.1	1.5
3-B	365	S	.088	61.4	1.3	22.5	65	3.0	2.0	5.5	13.7	27.7	25.2	12.3	2.8	2.5	.6
3-E	341	S	.099	49.6	.2	21.4	80	.7	5.5	8.1	29.2	37.8	9.0	2.3	6.1	1.6	
5-1/4-D	410	S	.035	66.8		21.1	60	.	2.2	8.4	19.5	22.5	21.4	9.5	4.4	4.2	1.4
5-1/2-A	375	S	.040	69.1		19.6	48	.4	2.1	3.6	19.1	16.4	34.3	20.1	3.1	3.1	1.4
5-3/4-B	387	S	.032	68.8		20.1	60	3.8	1.7	1.0	8.1	7.3	33.5	36.8	3.6	3.6	1.1
5-3/4-E	503	S	.030	61.3	Tr.	20.7	48	4.0	1.7	6.0	12.8	16.3	28.6	18.9	6.2	5.2	1.9
6-B	619	S	.037	63.8	Tr.	25.7	48	3.9	1.8	6.5	8.3	16.7	32.4	14.2	6.3	3.0	2.3
6-1/2-B	311	S	.037	52.3	Tr.	23.8	42	3.3	2.8	4.5	14.3	16.7	26.9	17.6	6.3	3.3	2.9
7-A	256	S	.078	72.2	1.3	29.6	80	6.1	2.4	11.6	31.4	18.2	18.4	5.4	2.6	14.5	7.2
8-A	331	S	.048	43.5	.8	18.7	32	5.9	2.5	1.7	2.4	3.1	11.8	31.0	22.8	2.3	9.
9-B	242	S	.36	45.2	.2	36.0	65	2.9	2.5	7.9	21.0	29.3	19.6	5.3	1.8	1.2	.7
9-C	287	S	.180	55.2	.2	28.8	80	5.5	1.8	5.4	16.7	23.4	26.2	5.6	2.8	2.7	1.3
9-1/2-D	322	S	.100	55.2	.1	28.8	80	4.4	2.5	12.9	30.1	22.8	13.3	6.1	3.4	2.5	.4
13-D	451	S	.029	73.4		16.8	48	4.1	1.8	2.8	10.5	17.8	20.6	25.4	13.8		

IRONDALE SEAM

Sample No.	Wt. per cent	8/10	10/14	14/20	20/28	28/35	35/48	48/65	65/100	100/150	150/200	-200	Mean size, mesh	Fe	CaO	Insol.	Phos.	s or U¹	Crumbling rate	Sample No.
3/4-01-B	29	0.6	6.1	10.7	7.8	7.9	9.9	13.4	28.2	6.8	3.4	5.2	60	20.8	15.6	35.9	0.16	U	77	3/4-01-B
3/4-01-C	26	2.3	12.3	13.4	10.1	7.7	6.5	7.8	17.4	9.5	5.8	7.2	48	22.7	15.4	34.3	.16	U	72	3/4-01-C
3-H	32	.6	5.2	11.9	17.7	7.7	6.8	7.3	6.4	7.8	6.2	17.1	60	33.7	11.7	44.5	.17	s		3-H
3-01-C	23		.9	1.8	1.9	4.0	5.4	8.1	34.8	16.1	7.7	17.5	115	28.8	11.8	34.0	.22	U	83	3-01-C
3-02-C	19			1.8	6.2	4.4	5.0	8.6	35.9	15.1	6.7	11.8	100	24.2	8.5	27.1	.14	U		3-02-C
4-01-C	23		1.1	1.9	3.9	4.4	5.2	8.4	37.8	13.7	6.0	17.6	100	27.4	11.8	35.8	.21	U		4-01-C
4-02-E	19			1.7	3.1	2.0	3.4	9.2	44.3	17.7	6.7	12.5	100	31.5	12.1	26.9	.21	U	85	4-02-E
4-1/4-01-F	21		.8	1.2	1.7	1.2	2.4	8.4	51.0	17.9	5.5	9.2	100	32.5	11.5	29.3	.19	U	151	4-1/4-01-F
4-1/2-B	25		.6		1.7	4.0	5.2	8.4	43.3	19.3	7.0	9.2	100	28.1	14.1	30.2	.17	U	105	4-1/2-B
5-C	26			.8		1.9	3.3	8.8	48.8	21.0	4.8	8.8	115	28.3	13.4	33.7	.19	U	121	5-C
5-1/4-A	25		4.6	12.8	15.1	13.3	4.2	8.2	18.6	20.4	4.6	7.3	60	26.1	14.0	31.9	.18	s	166	5-1/4-A
5-1/4-A-1	21	.6				1.6	6.0	6.1	47.6	20.6	4.8	9.2	115	28.4	13.6	28.4	.16	s	178	5-1/4-A-1
5-1/2-E	28		2.2	7.4	9.6	10.7	6.7	7.5	28.8	12.0	5.8	10.1	60	39.6	.1	38.5	.17	s	170	5-1/2-E
5-3/4-D	23		.6	1.0	7.4	8.4	8.0	8.1	27.8	19.6	7.2	10.1	65	42.5	.2	33.7	.11	s	218	5-3/4-D
6-D	16		3.8	6.7	4.1	7.0	7.7	8.1	21.7	23.1	8.1	10.1	80	48.5		26.9	.12	s	231	6-D
7-C	10	1.2	5.3	14.6	5.3	15.1	19.9	9.1	6.8	11.4	2.8	20.4	100	40.6	10.7	20.2	.11	s	131	7-C
10-A	12	1.4	4.2	14.6	13.0	14.1	23.0	7.9	6.9	3.5	2.8	9.1	80	43.6	8.0	19.9	.27	U	190	10-A
10-B	18		9.8	14.9	11.3	15.2	12.9	6.8	6.4	3.7	2.2	11.7	48	36.0	11.6	23.8	.15	s	179	10-B
11-A	18	1.8	6.7	14.9	16.3	22.1	12.9	7.3	5.9	2.6	2.3	7.3	48	39.4		24.5	.33	s	290	11-A
12-A	18		6.8	14.9	18.7	9.7	12.9	7.4	8.7	2.6	2.3	3.9	42	49.1		26.3	.06	s	235	12-A
13-B	9	1.4	6.7	7.7	6.3	9.7	23.9	7.4	8.7	3.5	2.3	31.3	42	54.4	.1	17.8	.05	s	221	13-B
13-A	26	.7	5.1	8.7	10.4	21.4	20.2	11.7	9.7	3.5	6.9	4.7	80	41.2	.2	36.6	.06	s	234	13-A

UPPER FERRUGINOUS

Sample No.	Wt. per cent	8/10	10/14	14/20	20/28	28/35	35/48	48/65	65/100	100/150	150/200	-200	Mean size, mesh	Fe	CaO	Insol.	Phos.	s or U¹	Crumbling rate	Sample No.
03-A	46			0.1	0.1	0.2	3.2	25.6	46.8	13.1	4.7	6.8	100	31.5	Tr.	53.3	0.052	s	101	03-A
02-A	53				.3	.3	11.4	35.6	34.2	6.7	3.1	9.1	100	26.9	Tr.	60.1	.029	s	286	02-A
01-A	49				.2	.2	1.3	5.8	18.9	41.9	23.2	14.5	170	27.4	Tr.	57.3	.069	s	232	01-A
01-B	55			.3	.5		4.9	8.8	47.1	26.4	7.4	11.2	115	27.0	Tr.	63.7	.022	s	394	01-B
01-C	59						.7	24.7	32.7	8.5	7.1	21.1	115	24.1	.1	67.6	.025	s	323	01-C
01-D	50							12.2	43.2	8.7	7.1	8.2	115	27.9	.1	57.0	.025	s		01-D
01-1-A	53	.5		.3	.5	2.0		6.2	31.3	18.8	3.3	8.3	100	25.6	1.0	59.4	.096	U	190	01-1-A
Q (1025E)	49					.4	1.1	20.8	39.3	18.8	4.0	8.3	115	29.7	.1	55.7	.038	s	184	Q (1025E)
Q (1025F)	51					.3	1.1	14.9	47.0	21.7	4.1	7.9	115	27.3	Tr.	58.4	.036	s	249	Q (1025F)
1/2-A	52						2.6	20.6	49.6	17.2	4.4	6.8	115	26.1	.1	59.4	.035	s	297	1/2-A
1/2-B	58			.1	.1	.1	1.6	17.5	48.2	6.9	5.3	38.4	150	15.6	Tr.	73.5	.034	s	338	1/2-B
1-A	57			.1	.1	.7	.9	19.5	54.7	12.0	3.3	8.4	100	23.6	Tr.	64.8	.043	s	253	1-A
1-1/2-A	53		.5	.6	.2		1.8	22.7	56.0	6.9	3.0	6.0	100	26.0	Tr.	59.5	.018	U	23	1-1/2-A
2-A	54		.3	.1	.4	.2	6.5	28.9	52.1	10.0	2.1	7.0	100	25.9	2.2	61.5	.021	s	410	2-A
2-1/4-A	52		.8	1.4	1.0	8.8	5.4	28.9	25.6	4.3	3.6	9.2	80	23.7	Tr.	59.4	.090	s	481	2-1/4-A
3-A	54		.5	.4	1.1	1.7	6.5	29.1	28.9	5.1	2.4	12.5	100	24.5	.1	63.6	.025	s	230	3-A
4-A	48		.3	1.1	.6	8.7	5.4	50.9	29.1	4.2	2.2	5.9	80	29.0	Tr.	54	.032	s	305	4-A
5-A	51	1.0	.8	4.4	5.9	8.8	23.9	36.6	8.3	1.8	2.2	7.9	65	28.0	Tr.	57.9	.044	s	312	5-A
6-A	51	.2	6.2	2.5	11.1	12.3	23.3	16.8	3.8	1.8	2.2	12.0	80	27.1	.1	59.2	.038	s	317	6-A
8-D	48		.7	4.3	10.0	17.4	18.3	13.7	14.6	9.4	4.9	6.5	60	26.7	Tr.	53.0	.610	s	194	8-D

Sizing analyses — Assay, percent

A summation of the analyses (table 3) shows that the constituents as determined account for only 92 to 98 percent of the ore (the higher figure will apply to concentrates), and the differences could not be explained by errors in determinations. Hence complete analyses of many samples were made to find out about the "missing" constituents and learn more about the nature of the ores. These analyses are given in table 6.

Near the right of table 3, "S" or "U" signifies surface or underground samples. Next to this column is "Crumbling rate." These numbers indicate softness, the highest numbers applying to the softest ores. This evaluation was made by a rule-of-thumb method that need not be described. It is difficult, because of the broad range of sizes of sand that was expected, to escape degradation in the test.

Summation of the results of the crumbling tests shows that the Irondale samples of high lime content were the hardest, followed by the Big seam, Ida, Upper Ferruginous, and finally the Middle Ferruginous samples, which were the softest. In the Big seam samples, those that had 20 percent lime were harder than those with one-half that amount.

The relation between CaO and crumbling rate of unweathered samples is shown more fully by the following:

Average CaO, percent	Average crumbling rate
9.1 to 9.4	156
9.4 to 10.4	133
10.4 to 12.1	130
12.1 to 21.8	107

The column for "Phos." shows that the phosphorus content is as high as 0.38 percent in the Big seam samples and is about the same for the samples from the Irondale seam, which lies immediately below the Big seam. The samples from the other three seams have a lower phosphorus content.

These assays are significant because present-day concentration methods fail to produce a concentrate appreciably lower in phosphorus than the raw ore.

Now the main features of table 3—the amount and particle size of the sea sand—may be considered. Sea sand is of important consideration from the standpoint of concentration (31). To obtain the sand in a natural and discrete condition, the sample was crushed only to 3/16-inch and digested by a modification of the sulfuric acid method standard for dissolution of clays. Substantially the only solids remaining were grains of sand. But this method is not inerrant; some of the sand grains had cracks containing hematite. When the hematite was dissolved, the grains fell to pieces and gave more fines than were present in the natural ore. Petrographic slides of the ore (figs. 4 and 5) show the shattered condition. The sand grains exhibit the characteristics of sands found on the sea shores today. Quartz is the principal mineral, but there is a small amount of feldspar with traces of mica and other minerals. Due to the marine origin there should be only a very small amount of the minus 200-mesh size. So, after what has been said about the "shattered condition," it would be well to discount the amount in the finest sizes shown in the size analysis.

Five seams or beds are shown in table 3, but they are not in geologic sequence. They are arranged in the order of progressive fineness of
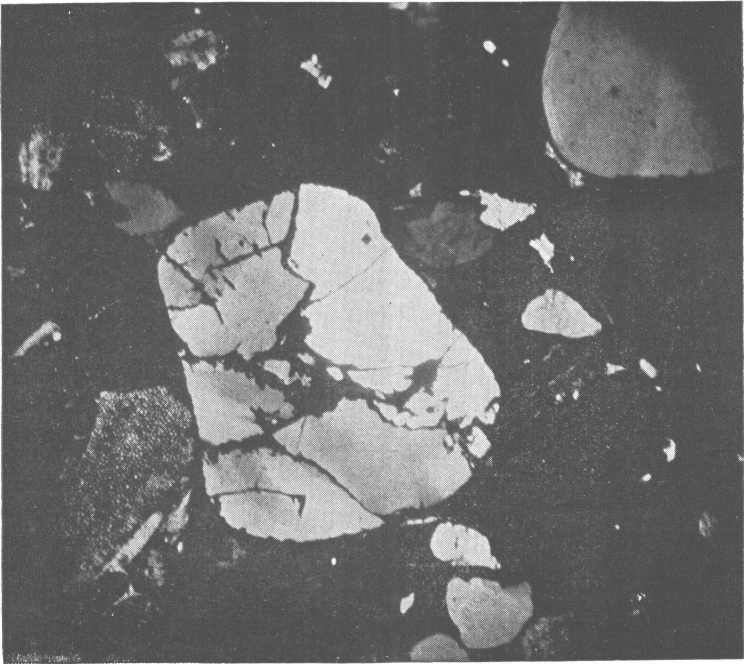
FIGURE 4.—Fractured quartz in iron ore, X 27.
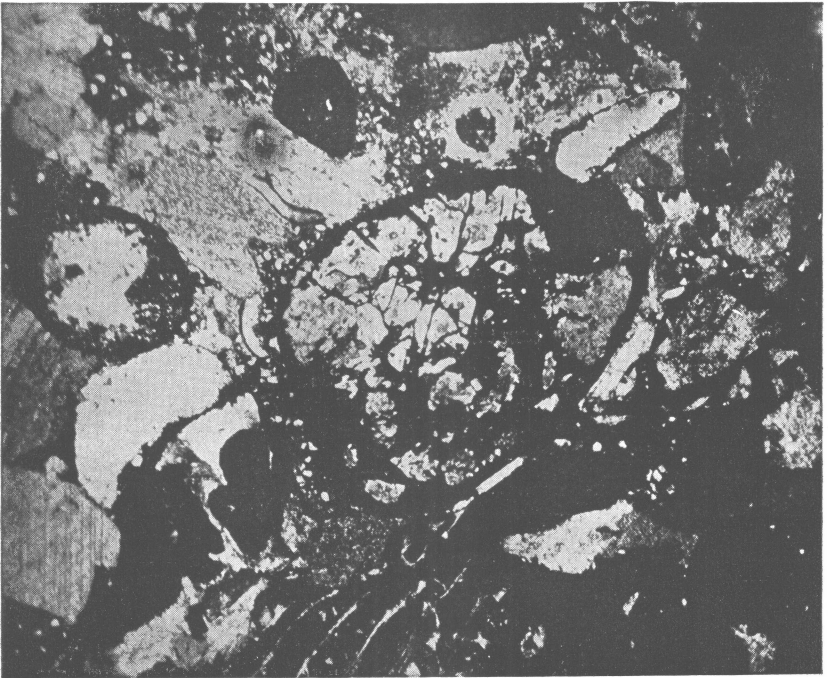
FIGURE 5.—Fractured "sugary" quartz in iron ore, X 27.

the sea sand. This plan, as shown by the column "mean size, mesh" places the seams in the following order: Big, Ida, Middle Ferruginous, Irondale, and Upper Ferruginous. (Phillips *(7)* places the mean mesh of the sand in the Irondale between the Ida and Big seam.) The Big seam has a mean size at about 42-mesh, and the weight, percent, reaches a peak at about 28-mesh, whereas the Upper Ferruginous has a mean size at about 80-mesh and the weight, percent, reaches a peak at about 80-mesh.

Throughout the records of the Big seam, the particle size of sand runs very uniform. The same may be said about the Ida; but in the remaining three seams there is a striking tendency for the sand to be coarser at the north end of the range. This regular variation of sizes of sand no doubt fits in with what is already known about the geology, but the only interest here is in the amenability to removal in mineral dressing. The size-distribution in a given product follows, to only a moderate degree, the distribution that would be expected in crushed homogeneous material.

The high sand content at the north end of the Big seam is known to be particularly objectionable. The operators say that the north end of the seam is too acid. Table 4 (a recompilation of only the underground samples included in table 3) shows that the increased acidity at the north end is due to increased sand in the upper bench. From south to north the amount of sand in the upper bench is at about 8 percent until Graces Gap is reached. There it begins to increase and reaches 25 percent at a point near 20th Street Gap. The amount of sand in the lower benches is about 22 percent throughout.

TABLE 4.—*Quartz sand in Big seam benches (underground)*

Sample No.	Quartz sand, percent		Sample No.	Quartz sand, percent		Sample No.	Quartz sand, percent	
	Upper bench	Lower bench		Upper bench	Lower bench		Upper bench	Lower bench
02-C	13		4 1/4-01-B		13	4 1/2-02-D		25
3-01-A	7		4 1/4-01-C		21	4 1/2-03-A	13	
3-01-B		21	4 1/4-01-D		25	4 1/2-03-B		22
3-02-A	10		4 1/4-01-E		24	4 1/2-03-C		22
3-02-B		20	4 1/4-02-A	6		4 1/2-03-D		27
4-01-A	9		4 1/4-02-B		10	4 1/2-04-A	19	
4-01-B		15	4 1/4-02-C		19	4 1/2-04-B		20
4-02-A	8		4 1/4-02-D		22	4 1/2-04-C		26
4-02-B		15	4 1/4-02-E		24	5 1/2-01-A	23	
4-02-C		23	4 1/2-02-A	9		5 1/2-02-A		23
4-02-D		27	4 1/2-02-B		18	5 3/4-01-A	25	
4-1/4-01-A	8		4 1/2-02-C		25	5 3/4-02-A	22	

Photomicrographs of some of the Big seam sand grains prepared by the leaching treatment were taken by L. H. Berkelhamer of the Bureau of Mines. They are shown in figures 6 to 14 in all sizes from 10- to 200-mesh, and the magnification is adjusted so that all appear in the same dimensions. The first three sizes, to and including the 20- to 28-mesh (figs. 6, 7, and 8), are roundish and of the shape that would be expected of water-worn grains, but at 28- to 35-mesh (fig. 9) the appearance changes; shattering becomes evident in forms like thin flakes. These look like gypsum, but petrographic examination shows that they are quartz. The amount of shattered grains increases in the fine sizes, and sugary looking grains make their appearance and are

most abundant in the finest sizes. The petrographic section, figure 5, gives the best illustration of the sugary nature. The particles are friable because they consist of very fine grains cemented together closely. These shattered grains have contributed too much to the fine sizes in table 3.
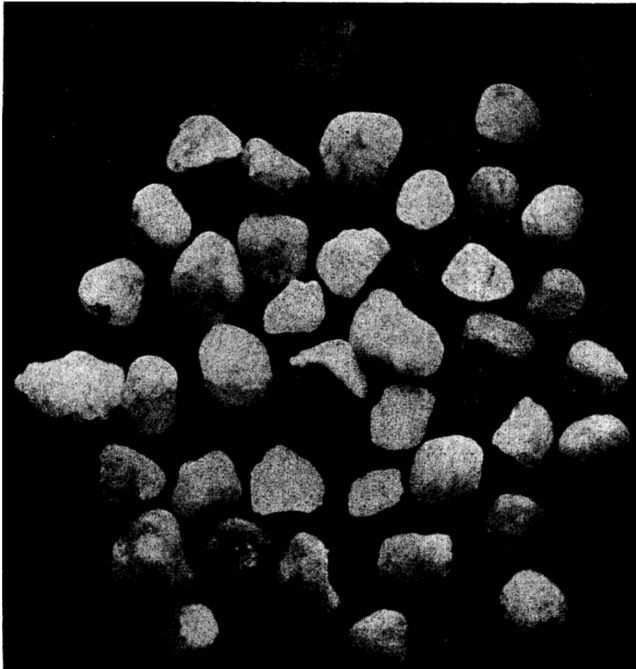
FIGURE 6.—Acid-cleaned sand from Big seam sample 7B, minus 10- plus 14-mesh, X 5.
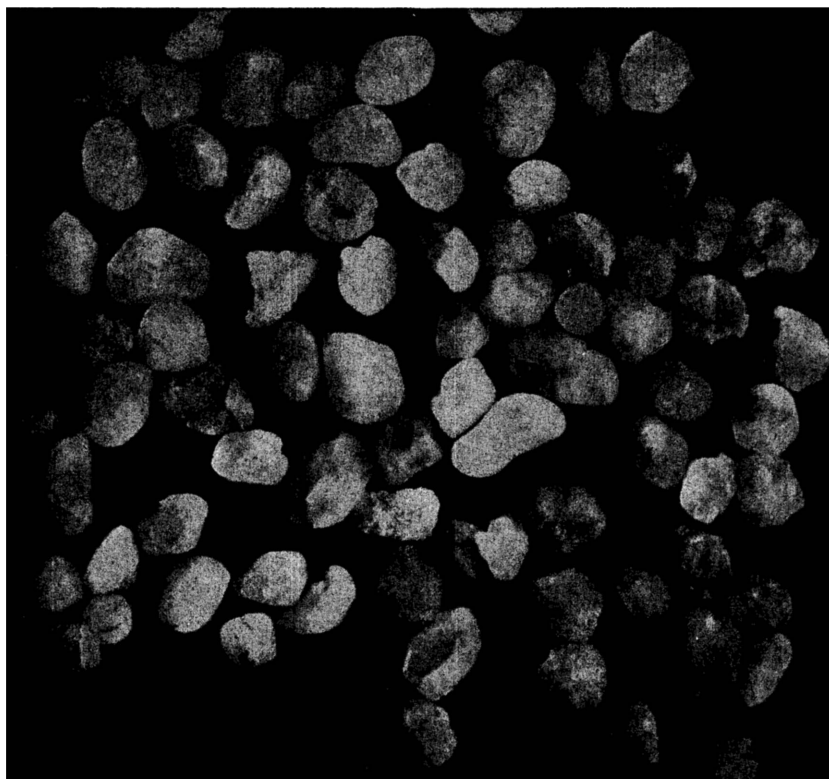
FIGURE 7.—Acid-cleaned sand from Big seam sample 7B, minus 14- plus 20-mesh, X 7.1.
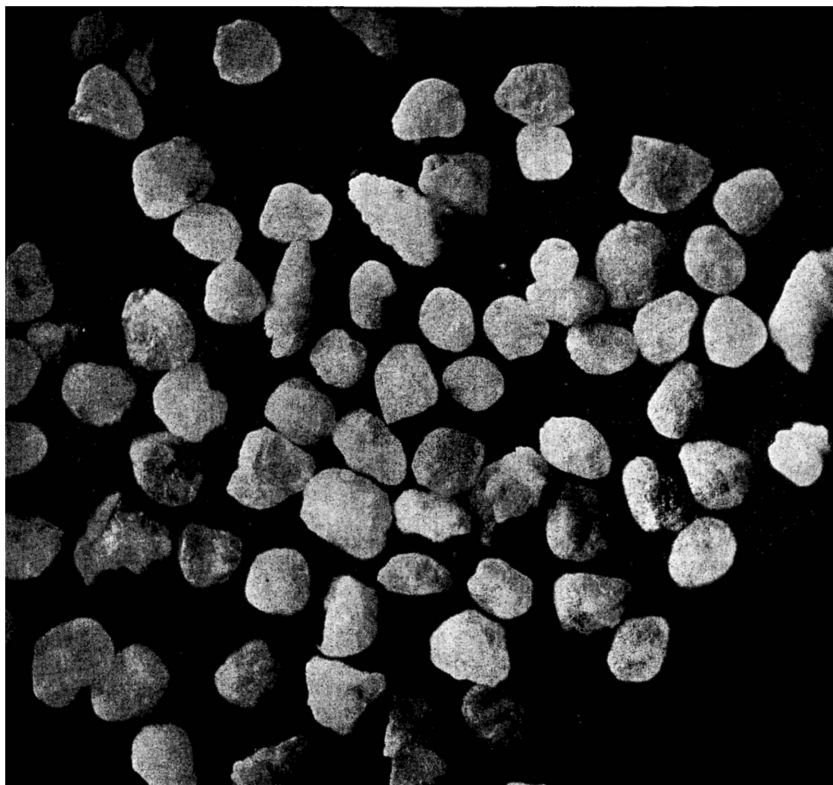
FIGURE 8.—Acid-cleaned sand from Big seam sample 7B, minus 20- plus 28-mesh, X 10.
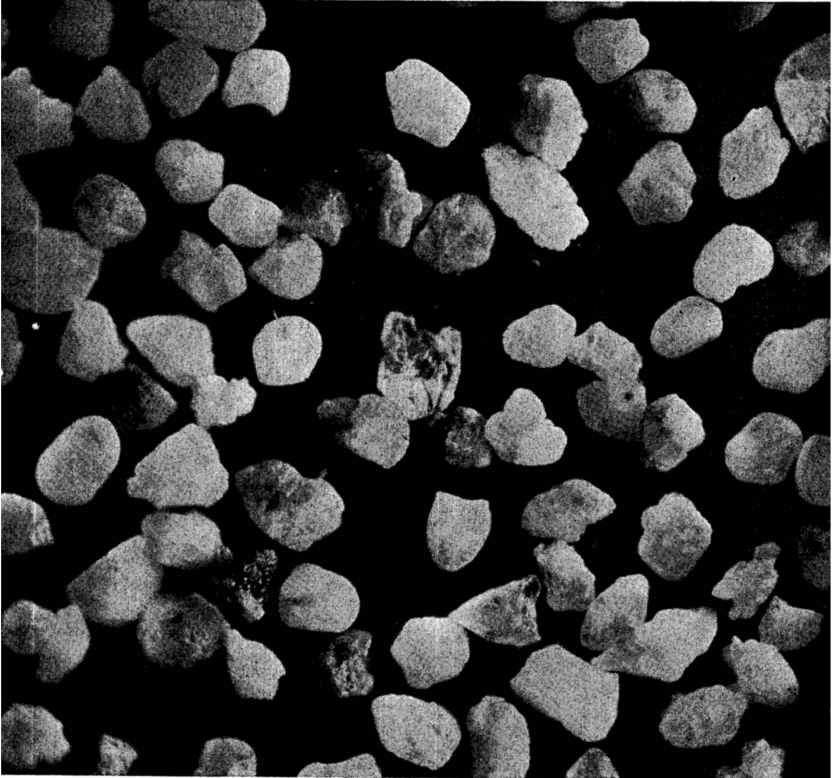
FIGURE 9.—Acid-cleaned sand from Big seam sample 7B, minus 28- plus 35-mesh, X 13.9.
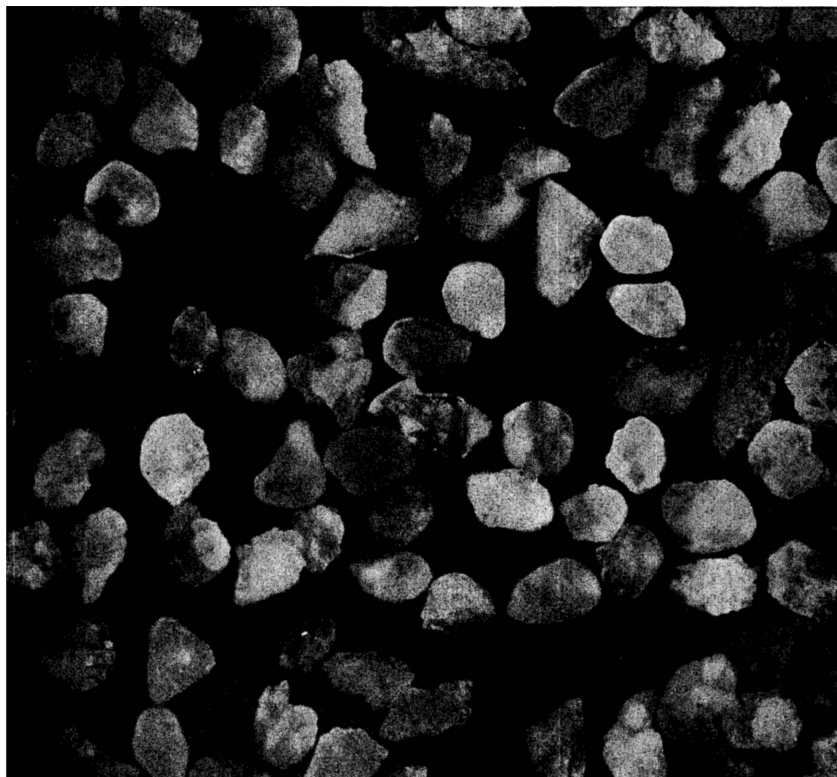
FIGURE 10.—Acid-cleaned sand from Big seam sample 7B, minus 35- plus 48-mesh, X 19.7.
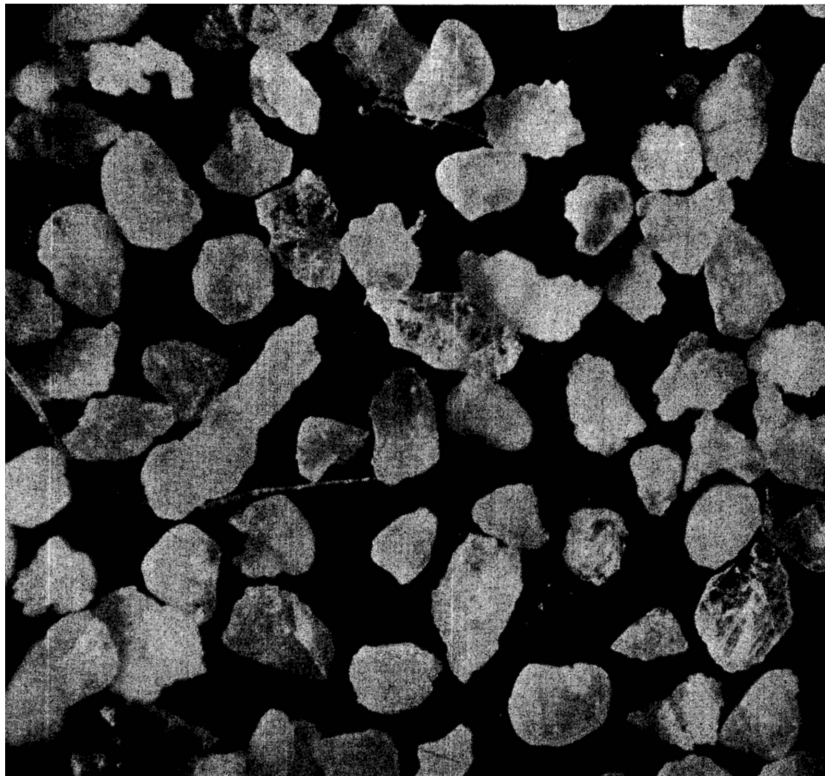
FIGURE 11.—Acid-cleaned sand from Big seam sample 7B, minus 48- plus 65-mesh, X 28.4.
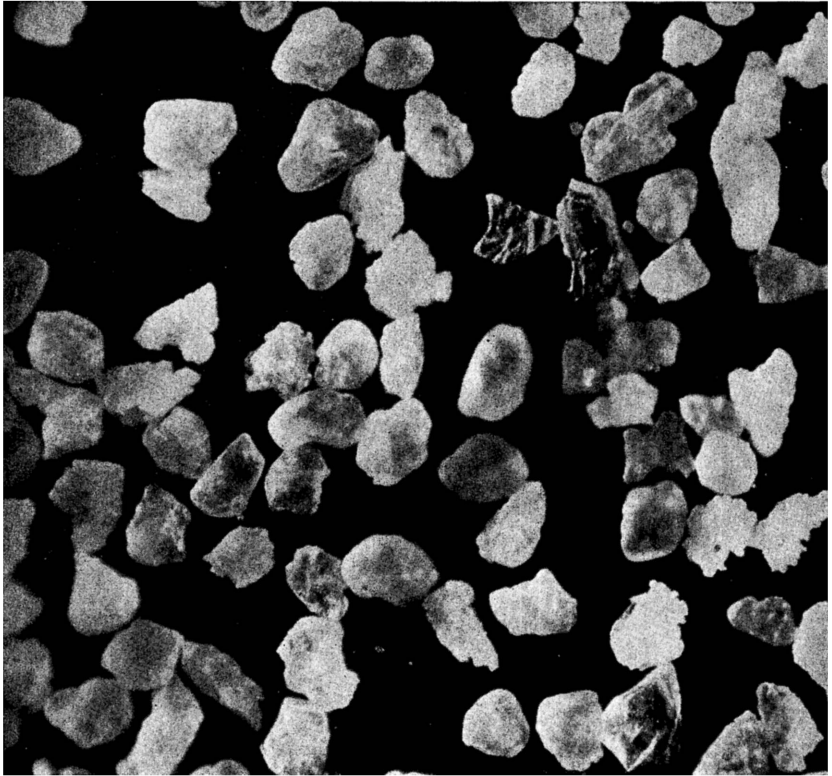
FIGURE 12.—Acid-cleaned sand from Big seam sample 7B, minus 65- plus 100-mesh, X 39.4.
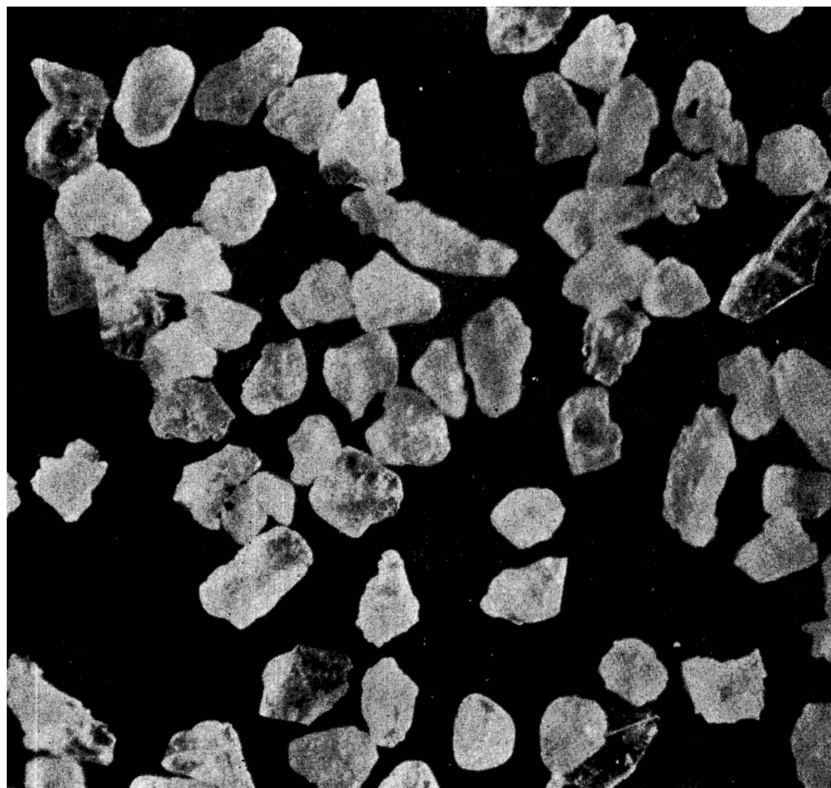
FIGURE 13.—Acid-cleaned sand from Big seam sample 7B, minus 100- plus 150-mesh, X 54.6.
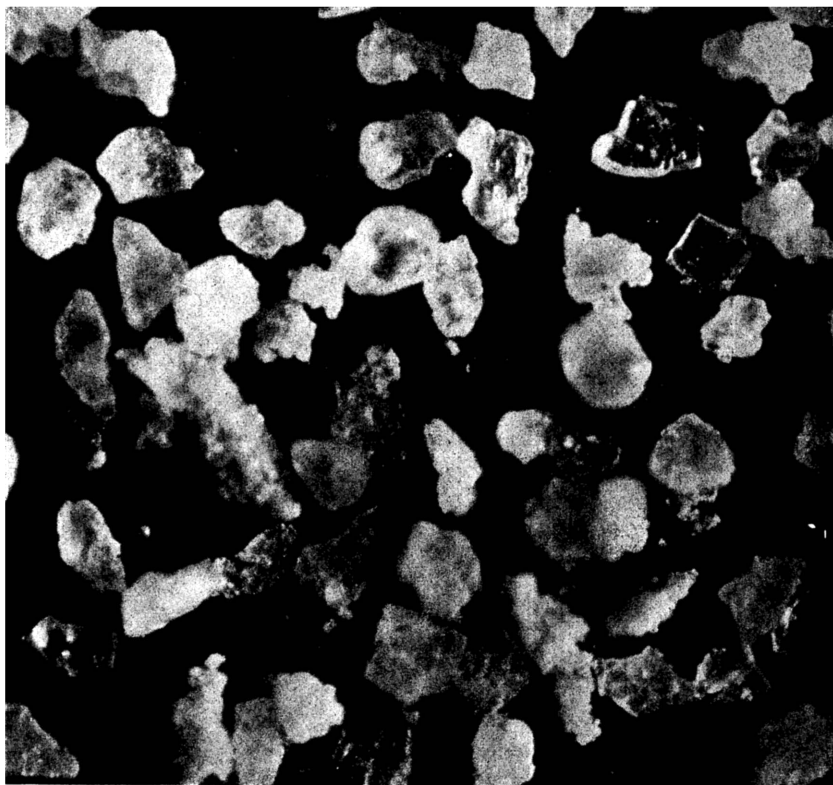
FIGURE 14.—Acid-cleaned sand from Big seam sample 7B, minus 150- plus 200-mesh, X 79.

In spite of vigorous acid treatment, some of the coarse grains retain a thin veneer of hematite, but this may be removed by a gentle treatment in hydrofluoric acid, after which the clear quartz of this particular suite is revealed. Samples from other places might have a different appearance.

TABLE 5.—*Average size analyses of the sea sand in the several seams*

Sieve size, mesh	Big seam	Ida seam	Middle Ferruginous seam	Irondale seam	Upper Ferruginous seam
8 to 10	1.0	0.9	0.8	1.2	0.6
10 to 14	5.3	6.1	1.5	3.8	1.3
14 to 20	14.3	11.7	3.1	7.0	1.5
20 to 28	19.4	13.3	5.7	7.7	2.0
28 to 35	19.3	11.9	12.6	9.2	2.9
35 to 48	13.0	19.5	20.3	8.7	6.7
48 to 65	5.6	19.0	22.6	8.0	22.9
65 to 100	4.6	5.2	17.4	25.4	34.5
100 to 150	3.5	2.0	6.5	11.8	12.4
150 to 200	2.6	1.7	1.6	5.6	4.8
Minus 200	11.4	8.7	6.9	11.6	10.4
Total	100.0	100.0	100.0	100.0	100.0
Mean mesh	42	42	60	60	80

The average sea-sand analyses, by seams, are given in table 5. Here again the reader is reminded that the sand, in place, is not as fine as indicated by the amount through 200-mesh. Because of this fact, all mean mesh calculations in tables 3 and 5 were made in accordance with the arithmetical rather than the geometrical "surface factors" advocated by the authors in a recent article *(35)*. The arithmetical "surface factors" method of calculating mean mesh tends to weight the finer sizes less and, for these particular data, gives a truer picture of the sand grains before the ore was digested in acid.

COMPREHENSIVE ANALYSES

Comprehensive chemical analyses were made of many of the ores to evaluate the elements missing in table 3. The results are given in table 6. Small amounts of manganese, titanium, magnesium, alkalies, acid-insoluble iron and lime, and acid-soluble silica and alumina were found to explain the low totals of the proximate analyses. No direct method for determining alumina was entirely satisfactory, and the values reported were calculated as the difference between the total mixed oxides and the sum of the oxides determind. The silica content computed from the complete analysis is 2 to 8 percent higher than that determined by the sulfuric acid method mentioned on page 39. The difference is due largely to silicate minerals soluble in sulfuric acid. 0.1 to 0.4 percent iron and up to 0.1 percent lime is insoluble in acid and hence goes unreported in the customary chemical analysis of these ores. That, however, is relatively unimportant, since all analyses are on a comparable basis as to accuracy, and more exact work is impractical in plant operation.

Arsenic assays are not included in table 6. Tests of sundry samples, however, showed that it was present. Three samples from the Big seam averaged 0.018 percent, two from the Upper Ferruginous were 0.009 percent, two from the Middle Ferruginous were 0.005 percent, two from the Ida were 0.006 percent, and two from the Irondale were 0.010 percent arsenic.

TABLE 6.—*Comprehensive analyses*

UPPER FERRUGINOUS SANDSTONE

Sample No.	Location	Loss by ign.	SiO₂	R₂O₃	CaO	MgO	Na₂O	K₂O	Total	P₂O₅	Fe₂O₃	MnₐO₄	TiO₂	H₂O	Quartz sand	Al₂O₃ by diff.
									Weight, percent, analyzed as received							
03-A	Sparks Gap	1.67	49.92	49.03	Tr.	0.33	0.13	0.53	93.71	0.12	44.51	Tr.	0.22	0.77	48.2	3.23
02-A	Raimund No. 1	1.43	56.34	40.93	Tr.	.22	.13	.46	93.56	.07	38.05	0.02	.17	.77	56.1	2.62
01-A	Realers Gap	4.94	51.14	43.03	Tr.	.13	.14	.15	93.33	.16	37.76	.17	.33	3.45	48.9	4.61
01-B	do	2.42	53.54	38.02	Tr.	.25	.13	.39	93.75	.05	33.80	.03	.38	1.24	56.9	3.76
01-C	do	1.86	62.56	34.46	0.12	.20	.21	.52	99.93	.06	30.25	.02	.36	.82	61.4	3.77
01-D	do	1.79	53.94	43.62	.03	.14	.19	.42	99.93	.06	30.59	.04	.30	.78	52.3	3.63
01-1A	do	1.89	54.52	40.90	.94	.48	.22	.53	103.18	.20	36.29	.01	.25	.43	53.1	4.15
Q(1025E)	Muscoda No. 5	1.64	50.44	46.74	Tr.	.27	.16	.47	99.53	.09	42.05	.02	.24	.63	48.1	4.34
Q(1025F)	do	1.60	54.02	42.94	.14	.25	.16	.52	99.72	.08	38.69	.02	.29	.64	52.5	3.86
1-A	Tanyard Gap	1.34	59.22	37.90	.02	.31	.27	.70	99.63	.08	33.51	.02	.29	.49	58.6	3.98
2-A	Wenonah No. 9	1.43	56.96	40.38	Tr.	.22	.11	.60	99.76	.10	36.79	.02	.24	.93	56.1	3.28
3-A	Ishkooda No. 11	1.97	57.34	38.92	Tr.	.29	.07	.96	99.45	.05	34.56	.04	.24	.93	56.7	4.02
4-A	Ishkooda No. 13	1.45	52.36	45.40	Tr.	.14	.03	.56	99.94	.07	42.38	.05	.22	.51	51.5	2.68
5-A	Walker Gap	1.43	53.60	43.44	.02	.30	.06	.63	99.48	.10	39.66	.04	.21	.52	52.5	3.43
6-A	20th St. Gap	1.99	54.53	41.93	.14	.17	.03	.60	99.54	.09	33.29	.03	.14	.78	53.6	3.43
						IDA SEAM										
02-B	Raimund No. 1	2.25	37.98	59.20	0.02	0.20	0.06	0.54	100.25	0.07	55.22	0.02	0.21	1.83	36.3	3.68
02-1/4-A	Raimund No. 3	1.61	41.92	55.54	.06	.12	.05	.59	99.89	.07	52.30	.01	.29	.77	41.1	2.87
6-1/2-A	22nd Street	1.72	35.92	61.16	.14	.22	.06	.42	99.64	.09	58.22	.01	.10	.89	34.8	2.74
9-A	Ruffner No. 3	2.06	47.68	49.26	Tr.	.13	.09	.39	99.61	.27	45.76	.03	.14	1.10	46.5	3.06
9-1/2-O&D	Ruffner No. 1	3.31	50.20	45.22	Tr.	.37	Tr.	.73	99.83	.25	40.74	.14	.12	2.39	49.2	3.97

MIDDLE FERRUGINOUS SANDSTONE

No.	Mine														
6-B	20th St. Gap	1.39	58.08	30.48	0.02	0.11	0.16	0.44	99.86	0.09	36.43	0.15	0.12	0.78	57.4	2.87
6-1/2-B	22nd Street	2.39	59.08	37.48	Tr.	.11	.10	.46	99.62	.08	33.41	.10	.10	1.36	57.6	3.13
7-A	Helen Bess	2.70	49.84	46.02	Tr.	.20	.14	.56	99.46	.18	41.51	.08	.12	1.54	43.5	4.13
8-A	Old Hammond	1.62	68.70	28.38	Tr.	.12	.12	.31	99.45	.11	26.29	.13	.09	.98	69.1	1.96
9-B	Ruffner No. 3	3.60	38.48	55.26	1.30	.31	.15	.56	99.96	.82	50.60	.15	.16	1.76	37.6	3.53

BIG SEAM

No.	Mine														
02-1/2-A	Potter No. 1	2.53	18.04	78.32	Tr.	0.08	0.16	0.62	99.75	0.16	73.13	0.01	0.24	1.58	15.5	4.78
02-C	Raimund No. 1	15.89	13.70	50.50	18.12	.72	.17	.59	99.43	.57	46.20	.11	.10	.69	12.5	3.52
4-1/2-A	Spaulding No. 1	8.90	23.08	56.06	10.60	.43	.21	.38	99.67	.73	52.42	.29	.09	.76	22.7	2.48
4-1/2-C	...do...	2.23	29.63	67.18	.14	.15	.05	.41	93.14	.39	63.83	.20	.15	1.37	28.8	2.56
1095-B	Spaulding No. 3	8.82	26.36	53.52	10.28	.49	.24	.34	101.05	.80	49.36	.14	.16	.50	25.2	3.06
5-B	Walker Gap	2.05	28.52	68.74	.02	.11	.17	.57	101.13	.19	65.42	.10	.19	1.03	27.5	2.84
4-3/4-A		8.32	26.61	54.29	9.82	.30	.13	.37	99.34	.76	50.27	.15	.10	.71	25.1	3.01
5-B	20th St. Gap	1.71	35.34	61.98	5.14	.07	.03	.50	99.65	.27	58.02	.16	.15	.93	33.6	3.38
6-C	Helen Bess	5.45	38.13	61.98	.02	.14	.23	.47	100.01	.73	45.75	.19	.19	1.22	37.1	3.56
7-B	Old Hammond	2.11	45.64	50.40	.30	.11	.09	.52	99.57	.39	47.43	.12	.17	1.32	45.4	2.69
8-B	Ruffner No. 1	11.27	29.06	46.04	12.43	.13	.24	.52	99.74	.37	42.81	.12	.15	1.10	29.3	2.59

IRONDALE SEAM

No.	Mine														
4-1/2-B	Spaulding No. 1	12.89	27.14	44.15	13.91	0.85	0.21	0.37	99.52	0.39	39.58	0.24	0.15	0.92	25.3	3.79
5-D	Walker Gap	11.35	29.42	44.72	13.28	.47	.32	.56	100.12	.41	40.23	.19	.18	.74	28.6	3.71
6-D	20th St. Gap	2.72	22.20	73.62	.08	.16	.06	.69	99.53	.25	68.04	.35	.21	1.40	19.5	4.77
7-C	Helen Bess	9.77	15.62	62.30	10.60	.47	.31	.60	99.67	.62	57.29	.19	.31	1.01	14.7	3.89
10-A	Ruffner No. 2	7.88	16.92	65.82	7.87	.36	.21	.55	99.61	.34	61.04	.12	.13	1.63	15.1	4.19
10-B	...do...	11.25	20.20	56.12	11.62	.44	.15	.45	100.23	.76	51.43	.11	.14	.28	18.0	3.68
11-A	Sadlers Gap	4.20	21.06	73.87	.28	.14	.21	.32	101.08	.14	63.99	.08	.23	2.52	18.4	4.43
12-A	Smith Cut	4.22	22.30	72.44	.08	.13	.19	.33	99.69	.12	67.94	.05	.23	3.03	20.2	4.10

RESULTS OF SAMPLING ARRANGED BY OWNERSHIP

A selected portion of the information in table 3 is shown in table 7, wherein the results of the sampling are arranged by ownership from south to north so that the owner can make immediate reference to samples taken on his property. Table 7 features, in addition, the name of the operating company, name of the seam, name of the mine, description of the point where the sample was taken, and thickness of the cross section.

TABLE 7.—*Sampling in order of ownership, from south to north*

Tennessee Coal, Iron & Railroad Co.
Sparks Gap, 4 miles south of Bessemer
UPPER FERRUGINOUS

| | Sample No. | Location | Crumbling rate | Analyses, dry basis, percent | | | | Quartz, sand | | |
				Fe	CaO	In-sol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
S[1]	03–A	Old prospect, above Southern Ry., 9-foot face.	101	31.5	Tr.	53.3	0.052	46	100	6.8

Potter Division
BIG SEAM

	Sample No.	Location	Crumbling rate	Fe	CaO	In-sol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
S	02–1/2–A	Potter No. 1.	215	53.1	Tr.	22.3	0.070	13	80	29.2

Republic Steel Corp.
Raimund Mines
IDA SEAM

	Sample No.	Location	Crumbling rate	Fe	CaO	In-sol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
S	02–1/4–A	Raimund No. 3, 5 ft. 6 in.	240	36.9	0.1	45.4	0.030	36	48	6.0

UPPER FERRUGINOUS

	Sample No.	Location	Crumbling rate	Fe	CaO	In-sol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
S	02–A	Raimund No. 1, 12 ft	296	26.9	Tr.	60.8	0.029	53	100	8.1

IDA SEAM

	Sample No.	Location	Crumbling rate	Fe	CaO	In-sol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
S	02–B	Raimund No. 1, 4 ft	315	39.2	Tr.	42.7	0.031	36	42	4.6

BIG SEAM

	Sample No.	Location	Crumbling rate	Fe	CaO	In-sol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
U[2]	02–C	Raimund No. 1, 6 ft	77	32.7	18.3	17.8	0.25	13	80	36.2

Tennessee Coal, Iron & Railroad Co.
Muscoda Division
UPPER FERRUGINOUS

	Sample No.	Location	Crumbling rate	Fe	CaO	In-sol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
S	01A	Muscoda No. 4, Readers Gap, 15 ft.	232	27.4	Tr.	57.3	0.069	49	170	14.5
S	01B	Muscoda No. 4, Readers Gap, 60 ft.	394	24.0	Tr.	63.7	.022	55	115	11.2
S	01C	Muscoda No. 4, Readers Gap, 12 ft.	323	21.4	0.1	67.6	.025	59	115	21.1
S	01D	Muscoda No. 4, Readers Gap, 11 ft.		27.9	.1	57.0	.025	50	115	8.0
U	01–1–A	Muscoda No. 4, Readers Gap, 35 ft.	190	25.6	1.0	59.4	.086	53	100	9.2
S	Q(1025E)	Muscoda No. 5, bottom of quarry.	184	29.7	Tr.	55.7	.038	49	115	8.3
S	Q(1025F)	Muscoda No. 5, run-of-quarry.	249	27.3	.1	58.9	.036	51	115	7.9

[1] S—Surface.
[2] U—Underground.

Sloss-Sheffield Steel & Iron Co.
UPPER FERRUGINOUS

	Sample No.	Location	Crumb-ling rate	Analyses, dry basis, percent				Quartz,sand		
				Fe	CaO	In-sol.	P	Wt., percent	Mean size, mesh	−200 mesh.. wt.. percent
S	1/2-A_____	Sloss No. 1, top 21 ft_____	287	26.4	Tr.	59.4	0.035	52	115	6.8
S	1/2-B_____	Sloss No. 1, bottom 4 ft__	338	15.6	0.1	73.5	.034	58	150	38.2

BIG SEAM

| U | 3/4-02-A___ | Sloss No. 2, 9 ft. 8 in____ | 52 | 35.1 | 19.1 | 11.2 | 0.28 | 6 | 65 | 34.6 |

IRONDALE SEAM

| U | 3/4-01-B____ | Sloss No. 2, top 7 ft._____ | 77 | 20.8 | 15.6 | 35.9 | 0.16 | 29 | 60 | 5.2 |
| U | 3/4-01-C____ | Sloss No. 2, bottom 1 ft. 8 in. | 72 | 22.7 | 15.4 | 34.3 | .16 | 26 | 48 | 7.2 |

Woodward Iron Co.
Woodward Mine
UPPER FERRUGINOUS

| S | 1-A_____ | Tanyard Gap, 32 ft_____ | 253 | 23.6 | Tr. | 64.8 | 0.043 | 57 | 100 | 8.4 |

IDA SEAM

| S | 1-B_____ | Tanyard Gap, 3 ft_____ | 194 | 43.7 | 0.10 | 34.8 | 0.088 | 26 | 60 | 8.2 |

MIDDLE FERRUGINOUS

| S | 1-C_____ | Tanyard Gap, 6 ft_____ | 237 | 30.0 | Tr. | 53.7 | 0.058 | 44 | 100 | 6.0 |

Tennessee Coal, Iron & Railroad Co.
Wenonah Division
UPPER FERRUGINOUS

| S | 1-1/2-A_____ | Wenonah No. 7, 30 ft____ | 410 | 26.0 | Tr. | 59.5 | 0.018 | 53 | 100 | 6.0 |
| S | 2-A_____ | Wenonah No. 9, 21 ft____ | 481 | 25.9 | Tr. | 61.5 | .021 | 54 | 100 | 7.0 |

IDA SEAM

| S | 2-B_____ | Wenonah No. 9, 2 ft_____ | 209 | 41.3 | Tr. | 38.0 | 0.037 | 28 | 48 | 10.2 |

MIDDLE FERRUGINOUS

| S | 2-C_____ | Wenonah No. 9, 9½ ft___ | 337 | 27.5 | Tr. | 57.1 | 0.031 | 44 | 115 | 31.3 |

UPPER FERRUGINOUS

| U | 2-1/4-A_____ | Wenonah No. 10, 33 ft 3 in. | 280 | 23.7 | 2.2 | 59.4 | 0.090 | 52 | 50 | 9.2 |

Ishkooda Division No. 11
UPPER FERRUGINOUS

| S | 3-A_____ | Outcrop along company road, 35 ft. | 305 | 24.5 | Tr. | 63.0 | 0.025 | 54 | 100 | 12.5 |

IDA SEAM

| S | 3-D_____ | Coarse, pebbly, 1½ ft___ | 246 | 41.3 | 0.4 | 36.6 | 0.192 | 28 | 48 | 12.1 |

MIDDLE FERRUGINOUS

	Sample No.	Location	Crumbling rate	Analyses, dry basis, percent				Quartz, sand		
				Fe	CaO	Insol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
S	3–B_____	10 ft_____	365	22.5	1.3	61.4	0.088	57	65	3.3
S	3–C_____	7½ ft_____	261	27.8	.2	56.1	.086	48	100	15.9

BIG SEAM

S	3–E_____	Road outcrop, 15 ft_____	165	38.9	8.6	23.5	0.32	16	48	12.9
U	3–02–A_____	At rotary dump, upper bench, 5 ft. 3 in.	104	40.1	12.3	15.9	.36	10	60	23.2
U	3–02–B_____	Lower bench, 11 ft. 1 in.	75	36.1	10.1	25.3	.32	20	35	11.3

IRONDALE SEAM

U	3–02–C_____	At rotary dump, 9 ft____	83	24.2	18.8	27.1	0.14	19	100	11.8

BIG SEAM

S	3–F_____	Upper bench, 3 ft. 5 in__	_____	39.7	15.1	13.6	0.33	7	65	24.5
S	3–G_____	Lower bench, 12 ft. 2 in_	_____	39.4	6.3	27.7	.39	20	42	12.3
U	3–01–A_____	Upper bench, 7 ft. 10 in_	_____	37.6	16.9	11.1	.31	7	65	26.8
U	3–01–B_____	Lower bench, 11 ft. 1 in_	_____	35.2	11.3	25.6	.36	21	42	8.2

IRONDALE SEAM

S	3–H_____	Lower bench, 7 ft. 7 in___	_____	33.7	0.5	44.5	0.17	32	60	17.1
U	3–01–C_____	Lower bench, 8 ft. 7 in__	_____	28.8	11.7	34.0	.22	23	115	17.5

Ishkooda Division No. 13
UPPER FERRUGINOUS

S	4–A_____	Quarry, 25 ft_____	360	29.8	Tr.	54.8	0.032	48	80	5.9

BIG SEAM

U	4–01–A_____	Upper bench, 4 ft. 5 in__	_____	35.1	18.3	14.3	0.31	9	48	20.6
U	4–01–B_____	Lower bench, 12 ft. 10 in_	_____	39.2	9.7	22.9	.37	15	48	12.4
U	4–02–A_____	Upper bench, 4½ ft_____	79	32.6	20.4	12.8	.28	8	48	18.1
U	4–02–B_____	Lower bench, 5 ft_____	95	40.7	9.4	20.6	.37	15	42	11.1
U	4–02–C_____	Lower bench, 5 ft_____	193	34.1	10.4	28.0	.30	23	42	6.5
U	4–02–D_____	Lower bench, 4 ft_____	107	32.5	.4	32.4	.28	27	42	10.8

IRONDALE SEAM

U	4–02–E_____	Right manway at 18th heading, 6 ft.	85	31.5	12.1	26.9	0.21	19	100	12.3
U	4–01–C_____	Above Ishkooda fault, 8 ft. 10 in.	_____	27.4	11.5	35.8	.21	23	100	17.6

Ishkooda Division No. 14
BIG SEAM

S	4–1/4–A____	Lower bench, 15 ft. 1 in__	_____	42.8	0.9	32.5	0.35	25	35	6.2
U	4–1/4–01–A__	Upper bench, 5 ft_____	139	35.3	18.1	13.0	.28	8	48	22.6
U	4–1/4–01–B__	Lower bench, 5 ft_____	133	39.9	9.8	19.8	.38	13	48	11.5
U	4–1/4–01–C__	Lower bench, 5 ft_____	183	36.5	9.1	26.8	.32	21	42	7.6
U	4–1/4–01–D__	Lower bench, 5 ft_____	177	33.8	9.3	29.8	.28	25	42	10.0

IRONDALE SEAM

U	4–1/4–01–F__	Old slope at 14th heading	151	32.2	11.5	29.3	0.19	21	100	9.5

BIG SEAM

	Sample No.	Location	Crumbling rate	Analyses, dry basis, percent				Quartz, sand		
				Fe	CaO	In-sol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
U	4-1/4-02-A__	Upper bench, 3 ft. 10 in__	109	34. 1	21. 8	10. 0	0.26	6	65	19. 5
U	4-1/4-02-B__	Lower bench, 5 ft _____	105	37. 2	15. 0	16. 0	.33	10	48	19. 2
U	4-1/4-02-C__	Lower bench, 5 ft _____	126	34. 6	11. 4	26. 2	.32	19	42	8. 8
U	4-1/4-02-D__	Lower bench, 5 ft _____	158	33. 8	9. 2	30. 7	.31	22	48	10. 2
U	4-1/4-02-E__	Lower bench, 3 ft. 6 in__	110	32. 8	10. 4	30. 1	.30	24	48	10. 9

Republic Steel Corp.
Spaulding No. 2
BIG SEAM

	Sample No.	Location	Crumbling rate	Fe	CaO	In-sol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
U	4-1/2-A_____	Manway at 34th right heading, 18 ft.	135	37. 1	10. 7	25. 4	0.34	20	35	9. 3
U	1095B_____	Run-of-mine___ _____	207	34. 7	10. 3	28. 9	.35	19	42,	8. 9

IRONDALE SEAM

	Sample No.	Location	Crumbling rate	Fe	CaO	In-sol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
U	4-1/2-B_____	In slope, 6 ft. 11 in_____	105	28. 1	14. 1	30. 2	0.17	25	100	9. 2

BIG SEAM

	Sample No.	Location	Crumbling rate	Fe	CaO	In-sol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
S	4-1/2-C_____	Outcrop, 18 ft. 6 in _____	177	45. 4	0. 1	31. 2	0.17	23	42	9. 2
U	4-1/2-02-A__	31st left heading, top 5 ft.	195	37. 8	15. 4	15. 8	.32	9	48	16. 5
U	4-1/2-02-B__	31st left heading, next 5 ft	173	38. 2	9. 2	25. 5	33	18	42	6. 7
U	4-1/2-02-C__	31st left heading, next 5 ft	189	32. 7	10. 2	31. 9	.30	25	35	5. 6
U	4-1/2-02-D__	31st left heading, bottom 4½ ft.	137	31. 3	10. 9	32. 0	.30	25	42	7. 7
U	4-1/2-03-A__	32d right heading, top 5 ft.	98	39. 0	11. 3	20. 7	.39	13	42	11. 0
U	4-1/2-03-B__	32d right heading, next 5 ft.	141	35. 6	9. 8	27. 8	.33	33	42	7. 7
U	4-1/2-03-C__	32d right heading, next 5 ft.	146	33. 6	9. 3	30. 6	.29	22	42	8. 5
U	4-1/2-03-D__	32d right heading, bottom 3 ft.	110	29. 5	9. 2	35. 8	.28	27	42	8. 5
U	4-1/2-04-A__	34th right heading, top 5 ft.	148	34. 8	10. 2	28. 8	.38	19	42	6. 7
U	4-1/2-04-B__	34th right heading, next 5 ft.	132	34. 2	10. 2	29. 0	.33	20	48	11. 0
U	4-1/2-04-C__	34th right heading, bottom 3 ft.	109	26. 4	14. 6	32. 1	.27	26	42	7. 8

Spaulding No. 3
BIG SEAM

	Sample No.	Location	Crumbling rate	Fe	CaO	In-sol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
S	4-3/4-A_____	Mouth of old slope, top 10 ft.	266	45. 8	Tr.	31. 5	0.08	26	35	5. 7

UPPER FERRUGINOUS

	Sample No.	Location	Crumbling rate	Fe	CaO	In-sol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
S	5-A_____	Walker Gap, 25 ft_____	312	28. 0	Tr.	57. 9	0.044	51	65	7. 9

IDA SEAM

	Sample No.	Location	Crumbling rate	Fe	CaO	In-sol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
S	5-D_____	Walker Gap, 2 ft. 6 in___	236	43. 6	0. 1	34. 7	0.100	28	42	9. 4

MIDDLE FERRUGINOUS

	Sample No.	Location	Crumbling rate	Fe	CaO	In-sol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
S	5-E_____	Walker Gap, 6 ft_____	341	32. 4	0. 2	49. 6	0.099	41	80	7. 7

BIG SEAM

	Sample No.	Location	Crumbling rate	Fe	CaO	In-sol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
S	5-B_____	Walker Gap, 12 ft. 6 in__	262	35. 5	9. 9	29. 6	0.33	23	35	5. 9

IRONDALE SEAM

	Sample No.	Location	Crumbling rate	Analyses, dry basis, percent				Quartz, sand		
				Fe	CaO	Insol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
S	5–C_____	Walker Gap, 6 ft_____	121	28.3	13.4	33.7	0.18	26	115	7.7

Tennessee Coal, Iron & Railroad Co.
Old Green Springs mine
MIDDLE FERRUGINOUS

S	5–1/4–D_____	Open-cut, 21 ft. 2 in_____	410	21.4	0.1	66.8	0.035	61	60	4.2

BIG SEAM

S	5–1/4–C_____	Top bench, 10 ft. 3 in_____	182	25.4	13.7	35.7	0.23	30	35	4.9
S	5–1/4–B_____	Bottom bench, 11 ft_____	227	34.6	6.6	34.7	.33	27	42	5.8

IRONDALE SEAM

S	5–1/4–A_____	Open-cut, 9 ft. 6 in_____	166	26.1	14.0	31.9	0.16	25	60	7.3
S	5–1/4–A–1___	Open-cut, 7 ft. 6 in_____	178	28.4	13.6	28.4	.17	21	115	9.2

Republic Steel Corp.
Valley View No. 2
BIG SEAM

S	5–3/8–A_____	Junction of old openings, top bench, 10 ft.	188	38.5	6.1	27.3	0.36	20	42	6.5

MIDDLE FERRUGINOUS

S	5–1/2–A_____	Robinson, 11 ft. 9 in_____	375	19.6	0.1	69.1	0.04	62	48	3.8

BIG SEAM

S	5–1/2–B_____	Robinson, top bench, 11 ft. 2 in.	223	41.1	1.8	33.2	0.34	28	35	4.9
S	5–1/–2–C_____	Robinson, bottom bench, 3 ft. 8 in.	148	30.1	9.3	35.3	.29	30	35	7.3
S	5–1/2–D_____	Robinson, bottom bench, 6 ft. 6 in.	222	36.4	.4	43.4	.22	34	35	6.4

IRONDALE SEAM

S	5–1/2–E_____	Robinson, 7 ft. 9 in_____	170	39.6	0.1	38.8	0.11	28	65	10.1

BIG SEAM

U	5–1/2–01–A__	Upper bench, top 7 ft___	157	35.1	3.8	27.9	0.35	23	35	7.0
U	5–1/2–02–A__	Upper bench, next 6 ft__	262	35.0	9.5	30.1	.32	23	42	6.9

Central Iron & Coal Co.
Valley View No. 1
IDA SEAM

S	5–3/4–C_____	7 ft_____	250	41.6	0.1	39.3	0.068	32	35	3.9

MIDDLE FERRUGINOUS

S	5–3/4–B_____	Right of old slope, 15 ft__	387	20.1	0.1	68.7	0.032	67	48	4.0
S	5–3/4–E_____	On road, 23 ft. 8 in_____	503	20.3	.1	68.8	.030	61	60	3.9

BIG SEAM

	Sample No.	Location	Crumb-ling rate	Analyses, dry basis, percent				Quartz, sand		
				Fe	CaO	In-sol.	P	Wt., percent	Mean size, mesh	−200 mesh, wt., percent
S	5-3/4-A_____	Top bench, 10 ft_____	198	44.2	0.4	32.0	0.32	24	35	3.7

IRONDALE SEAM

S	5-3/4-D_____	In caved drift, 6 ft. 9 in__	218	42.5	0.2	33.7	0.12	23	80	10.2

BIG SEAM

U	5-3/4-01-/ __	Top bench, 14 ft. 10 in__	234	37.5	6.5	29.2	0.35	25	35	4.7
U	5-3/4-02-A__	Top bench, 10 ft. 6 in____	198	37.6	7.4	28.1	.35	.22	35	6.2

Vulcan Park
20th Street Gap
UPPER FERRUGINOUS

S	6-A_____	Cut at road, 6 ft_____	317	27.1	0.1	59.2	0.038	51	48	12.0

MIDDLE FERRUGINOUS

S	6-B_____	Cut in road, 23 ft. 6 in___	619	25.7	Tr.	61.3	0.039	54	48	3.3

BIG SEAM

S	6-C_____	Highway cut, 20 ft__ ____	325	41.1	Tr.	39.0	0.12	32	35	40.0

IRONDALE SEAM

S	6-D	Cut in road, 4 ft. 9 in____	231	48.5	0.1	26.9	0.11	16	100	10.1

Henrietta Road and 22d Street
IDA SEAM

S	6-1/2-A_____	3 ft_____	_____	4.2	0.1	38.5	0.041	30	48	6.6

MIDDLE FERRUGINOUS

S	6-1/2-B_____	30 ft. 4 in_____	311	23.8	Tr.	63.8	0.037	56	42	6.1

Alabama By-Product Corp.
Old Helen Bess
MIDDLE FERRUGINOUS

S	7-A_____	5 ft. 9¼ in_____	256	29.6	Tr.	53.2	0.078	47	80	5.9

IDA SEAM

S	7-D_____	4 ft_____	134	41.3	Tr.	38.0	0.110	?1	48	6.9

BIG SEAM

S	7-B_____	18 ft. 7 in_____	294	32.4	5.2	41.9	0.32	32	42	10.4

IRONDALE SEAM

S	7-C_____	Open-cut and drift, 5 ft__	131	40.6	10.7	20.2	0.27	10	80	20.4

Sloss-Sheffield Steel & Iron Co
Ruffner District
UPPER FERRUGINOUS

	Sample No.	Location	Crumb-ling rate	Analyses, dry basis, percent				Quartz, sand		
				Fe	CaO	In-sol.	P	Wt. percent	Mean size, mesh	—200 mesh, wt., percent
S	8–D_____	Old Hammond, 1 ft. 6 in_	194	26.7	Tr.	53.0	0.61	48	60	6.5

IDA SEAM

S	8–C__ _____	Old Hammond, 3 ft_____	157	44.7	0.2	31.8	0.155	22	60	19.4

MIDDLE FERRUGINOUS

S	8–A__ _____	Old Hammond, 30 ft____	331	18.7	Tr.	72.2	0.048	66	32	2.9

BIG SEAM

S	8–B_____	Old Hammond, 14 ft. 3 in.	278	33.8	0.3	49.0	0.17	42	35	5.6

IDA SEAM

S	9–A_____	Ruffner No. 3, 4 ft. 4 in__	292	32.5	Tr.	51.4	0.120	45	35	10.1

MIDDLE FERRUGINOUS

S	9–B_____	Ruffner No. 3, 19 ft_____	242	36.0	1.3	43.5	0.36	36	65	5.5
S	9–C_____	Ruffner No. 3, 7 ft. 6 in__	287	34.0	.8	45.2	.180	36	65	5.6

IDA SEAM

S	9–1/2–C_____	Ruffner No. 1, 4 ft. 5 in__	297	31.8	0.3	51.0	0.15	42	35	7.6

MIDDLE FERRUGINOUS

S	9–1/2–D_____	Ruffner No. 1, 10 ft_____	322	28.8	0.2	55.2	0.100	47	80	4.4

IRONDALE SEAM

S	10–A_____	Ruffner No. 2, 4 ft_____	190	43.6	8.0	19.9	0.15	12	48	9.1
U	10–B_____	Ruffner No. 2, 4 ft_____	179	36.0	11.6	23.8	.33	18	48	11.7

Sadler Gap
IRONDALE SEAM

S	11–A_____	West of highway 11, in railway cut, 4 ft. 6 in.	290	49.4	0.3	21.5	0.06	18	42	8.3

Smith Bros.
Morrow Gap mine, ¼ mile south of Morrow Gap
IRONDALE SEAM

S	12–A_____	Smith Cut, 5 ft_____	235	49.1	0.1	26.3	0.05	18	42	7.9

Shook & Fletcher Supply Co.
Trussville mine, Red Gap
MIDDLE FERRUGINOUS

S	13–D_____	3 miles northwest of Trussville, 15 ft.	451	16.8	0.1	73.4	0.029	65	48	4.4

IRONDALE SEAM

S	13–B_____	Upper bench, 1 ft. 5 in__	221	54.4	0.1	17.8	0.06	9	80	31.3
S	13–A_____	Lower bench, 2 ft. 3 in___	234	41.2	.2	36.6	.06	26	48	4.7

TESTS FOR PHYSICAL NATURE

"TOOLS"

Many tools are available for testing the physical nature of ores. In fact, they are so numerous—and so frequently unsuited to a particular ore—that considerable research is necessary to isolate the useful from the useless. For example, the literature shows that subsieve sizing may be accomplished by any of the following methods:

Dry work: Haultain Infrasizer.[6]
             Roller analyzer.[7]
             Carman air permeability.[8]
             Ocular photocell.[9]
             Coercimeter—magnetic remanence.[10]
             van der Waal's adsorption.[11]
             Statistical methods.[12]

Wet work: Bouyoucos hydrometer.[13]
             Andreasen pipette.[14]
             Wagner turbidimeter.[15]
             Cooke short-column elutriator.[16]
             Sedimentation balance.[17]
             Solubility rate (quartz).[18]
             Flow meter.[19]

Each of these methods has its merits—and limitations. For instance, the coercimeter is not adaptable to nonmagnetic powders, van der Waal's adsorption method is not suitable for sizes coarser than a few microns, and the Wagner turbidimeter is extremely limited in its field of usefulness.

In this work, therefore, the authors have used only those tools that will yield reliable, pertinent information on these particular iron ores. The most useful tools have been found to be:

(*a*) *Particle-size apparatus.* Although woven-wire (square-mesh) sieves are indispensable in studying the characteristics of an ore they are so familiar as to require no comment other than that the sieves used in this work constituted a series wherein the area of the opening varied as $\sqrt{2}$ (Tyler sieve scale). In less common usage are the rectangular-mesh sieves, used for determining the amount of flaky particles (*36*). Data on the flakiness of the Red Mountain iron deposits

[6] Haultain, H.E.T., Splitting Minus 200-Mesh with Superpanner and Infrasizer: Canadian Min. and Met. Bull. N. 301, May 1937 (Trans.), pp. 229–240.

[7] Roller, P. S., Measurement of Particle Size with an Analyzer: Proc. Am. Soc. Test. Mat., vol. 32, part 2, 1932, p. 607.

[8] Carman, P. C., Fluid Flow Through Granular Beds: Trans. Inst. Chem. Eng., vol. 15, 1937, p. 150.

[9] Faust, Geo. T., and Cooke, S. R. B., Ocular Photocell for Rapid Determination of Projected Area of Opaque Particles: Bureau of Mines Rept. of Investigations 3460, 1939, 14 pp.

[10] DeVaney, Fred. D., and Goghill, Will H., Coercimeter in Grinding Tests: Trans. Am. Inst. Min. and Met. Eng., vol. 134, 1939, pp. 282–295.

[11] Gaudin, A. M., and Bowdish, F. W., Surface Measurement by van der Waal's Adsorption: Am. Inst. Min. and Met. Eng., Min. Technol., vol. 8, No. 3, May 1944, pp. 1–6.

[12] Rammler, E., (Determination of the Specific Surface of Milled (Ground) Materials): Ztschr. Ver. deut. Ing. Verfahrungstechnik, 1940. No. 3, pp. 150–180.

[13] Norton, F. H., and Speil, S., Measurement of Particle Size in Clays: Jour. Am. Ceram. Soc., vol. 21, 1938, pp. 89–97.

[14] Loomis, G. A., Grain Size of Whiteware Clays as Determined by the Andreasen Pipette: Jour. Am. Ceram. Soc., vol. 21, 1938, pp. 393–399.

[15] Wagner, L. A., A Rapid Method for the Determination of Specific Surface of Portland Cement: Proc. Am. Soc. Test. Mat., vol. 33, part 2, 1933, p. 553.

[16] Cooke, S. R. B., Short-Column Hydraulic Elutriator for Subsieve Sizes: Bureau of Mines Rept. of Investigations 3333, 1937, pp. 35–51.

[17] Weinig, A. J., A Functional Size Analysis of Ore Grinds: Colorado Sch. Mines Quart., vol. 28, No. 3, July 1933, pp. 53–68.

[18] Gross, J., and Zimmerley, S. R., Surface Measurement of Quartz: Trans. Am. Inst. Min. and Met. Eng., vol. 87, 1930, pp. 7–50.

[19] Bell, John W., Method for the Measurement of Surface of Finely Dividend Material: Canadian Min. and Met. Bull., November 1944, pp. 424–436.

will not be found in this report—not because the tests were not made but because the percentage of flaky particles was so insignificant as to present no especial problem in beneficiation.

Subsieve data were obtained by the Bouyoucous hydrometer,[20] short-column elutriator, [21] and Andreasen pipette[22] methods.

(b) *Optical microscope.* Evidence of the value of the microscope in studying the ore has already been given (figs. 4 and 5).

(c) *Sink-float.* Sink-float data are very important for appraising liberation. As a guide to gravity concentration of these ores sink-float is, in the opinion of the authors, inferior to the recently developed method of precision batch jigging.

(d) *Precision batch jig (29).* The precision batch jig for segregating particles according to settling rate is valuable for predicting the amenability of an ore to concentration by those so-called "gravity" processes that, in reality, effect a separation because of difference in settling rate.

(e) *Attrition scrubber (37).* The attrition scrubber is an effective laboratory tool for appraising the differential-grinding propensities of an ore.

(f) *Magnetic concentrator.* Its use is obvious.

(g) *Iron–specific-gravity relationship (30).* The iron–specific-gravity relationship finds its most useful application in the commercial ore concentrator. Plant tests were greatly facilitated by its use. The method is outlined on page 76.

<div align="center">SIZE ANALYSES</div>

Size analyses of the Birmingham iron ore cannot be rationalized without complete understanding and appreciation of their heterogeneity. The factor contributing most to the failure of the comminuted material to meet the "laws of distribution" of homogeneous products is, in part, the presence of particles of sea sand. When their size-distribution is understood, the irregularities in screen analyses will be accepted and not reflect criticism upon the laboratory. Familiarity with the sizing of the sea sand (table 3) will help in the understanding of the irregularities of table 8 (p. 63) on the screen analysis of the crushed ores. The data of table 8 are intended to show the general contrast in size analysis of the Big seam *type* of material with the ferruginous sandstone *type*.

The Big seam sample (table 8) was crushed through 2 inches in a gyratory. Unlike some other ores, except for easily identified extraneous material such as slate, any particle coarser than 6-mesh is likely to be of about the same grade as a carload of the ore. In the finer sizes the liberated particles of quartz appear as a diluent. In the Big seam they come in at about 8-mesh and reach maximum proportions between 10- and 20-mesh. The sample, having been crushed through 2 inches and screened dry, would have given a regular downward-sloping "weight-percent" curve below 4-mesh if the ore had been homogeneous. Here, due to liberated sea sand, there is a bulge in the distribution curve at 10- to 20-mesh, and the diluting effect of the sand is shown by the iron content, which drops from 34.8 percent at 8-mesh

[20] See footnote 13.
[21] See footnote 16.
[22] See footnote 14.

to 32.3 percent at 20-mesh. At 65-mesh the sand is less abundant and because of the vanishing quantity of sea sand the tenor of the iron reaches 40.2 percent, increasing regularly in the fine sizes. Because of the differential crushing of the hematite and the absence of sand in the finest size, the minus 200-mesh portion was graded up to 45.7 percent iron, the average tenor of the whole sample being 34.6 percent iron. The semieconomic value of lime in all of the Big seam ores has already been discussed (p. 18). If not semieconomic, it is at least an inoffensive diluent. In the absence of free limestone the variation of CaO from size to size is not enough to be significant.

The ferruginous sandstone (table 8) does not contain a significant amount of lime as a friendly diluent. Sea sand is the main diluting agent, but inherent clay must not be overlooked. The sand has a top limiting size (table 3) at about 20-mesh, and free particles predominate at 48- to 100-mesh. When this observation is applied in table 8 the high percentage weight and low iron content of the ferruginous sandstone in the sizes from 48 to 150 will be understood.

TABLE 8.—*Screen analyses of crushed products from Big seam and ferruginous sandstone, in percent*

| Size, mesh | Big seam, all through 2 inches | | Ferruginous sandstone | | | | |
| | Weight | Assay Fe | Weight | Assay | | | |
				Fe	CaO	MgO	Combined alkalies
Plus 4	72.2	34.6					
4 to 6		34.7	9.9	24.2			
6 to 8	3.0	34.8	9.8	24.2			
8 to 10	3.5	33.9	9.1	24.2			
10 to 14	4.9	32.8	6.3	24.0			
14 to 20	5.0	32.3	5.3	23.6			
20 to 28	3.9	33.0	4.3	23.2			
28 to 35	3.0	35.1	4.6	23.4			
35 to 48	1.7	37.5	4.8	23.1			
48 to 65	1.1	40.2	8.6	20.1			
65 to 100	.7	42.0	17.2	14.1			
100 to 150	.4	43.4	8.0	15.9			
150 to 200	.3	44.1	3.1	33.5			
Minus 200	.3	45.7					
200 to 270			2.0	39.7			
270 to 400			1.9	42.2			
Minus 400 to 20 λ			2.8	43.5	1.10	0.40	0.90
Minus 20 λ			2.3	37.3	.60	.40	1.20
	100.0	34.6	100.0	23.1			

From 200-mesh and down below the lower limits of the screens the grade gradually improves until the minus 20-micron size is reached where the grade in some samples has deteriorated because of clay.

At the bottom of table 8 will be found supplementary notes showing that CaO, MgO, and combined alkalies, respectively, are scarcely above 1 percent in the finest sizes of the ferruginous sandstone.

In these ores, as well as others, the coarsest size of gangue particles determines the size at which beneficiation may begin; in the Big seam ores it is about 8-mesh and in the ferruginous sandstones about 28-mesh.

DEGRADATION

If attention is not given to the history and method of handling the sample, screen analysis of the Big seam iron ores is likely to be mis-

leading. That is, degradation must be taken into account. By degradation is meant disintegration by wetting, drying, and rough handling, such as screening on a Ro-tap. Degradation is comminution by the mildest method of handling, after which come, in order of severity of treatment, self-grinding, and next, ball milling. Many tests have shown the rate of comminution in self-grinding and ball milling, but here the rate of degradation is considered.

For the examination a sample of Big seam ore was taken from the mill hopper; it was dry-screened on 6-mesh and the oversize rejected.

Under "Original" in table 9 is shown the usual screen analysis. To prepare the screened dry charge for the first cycle the minus 100-mesh product was rejected, and the remainder was soaked and washed in water for a few minutes, after which the accrued slime was decanted. The residue was given a quick drying for screening in "1st cycle". Data concerning the original, the first, and the sixth cycles are shown in table 9. The test was stopped after the sixth cycle was screened.

TABLE 9.—*Degradation of Big seam ore by screening, washing, decanting and drying, in six cycles*

Mesh	Original		1st cycle		6th cycle	
	Wt., gm.	Fe, percent	Wt., gm.	Fe, percent	Wt., gm.	Fe, percent
6 to 8	16.0	31.7	14.5		13.0	31.1
8 to 10	21.5	30.5	20.5		19.5	30.5
10 to 14	22.0	29.3	21.0		20.5	27.5
14 to 20	41.5	28.7	34.0		33.5	26.9
20 to 28	39.5	31.5	31.5		31.0	28.1
28 to 35	47.5	34.5	36.5		36.0	31.1
35 to 48	35.5	36.3	30.5		30.0	33.5
48 to 65	23.0	37.0	20.5		20.0	34.9
65 to 100	16.5	37.8	15.5		14.5	35.2
Minus 100	29.5	40.6	15.5		.5	40.4
	292.5	36.6	240.0		218.5	30.6

Most of the degradation was in the first cycle and was in the finer sizes; for example, the weight of the 28- to 35-mesh decreased from 47.5 in the original to 36.5 grams in the first cycle, and the grade of that size throughout all the cycles dropped from 34.5 to 31.1 percent iron.

Removal of the minus 100-mesh screen sizes and removal of the slimes by decantation reduced the iron content from 36.6 to 30.6 percent. Hence the material that was removed (74.0 grams from a total sample of 272.5 grams) must have been a graded-up product.

Taking into account the 46.6 grams of minus 100-mesh material that was removed by screening, 27.4 grams is left that was removed as slime formed by degradation in water. This decanted slime, most of which came off in the first steps, is of prime interest. It amounted to nearly 10.0 percent of the whole sample and had a tenor as follows:

Product	Fe, percent	CaO, percent	Insol., percent
Slime	49.2	5.2	15.4

A sample from another place in the Big seam gave analogous results. Hence, the indications are that if the run-of-mine ore is screened dry the screens will retain about 10 percent weight that would have yielded to degradation if the sample had been washed.

SINK-FLOAT

The statement by Crane *(38)* that "The average porosity of the ore is 3.48; the average true and apparent specific gravities are 3.61 and 3.48" should have been taken as a warning against leaning too heavily on sink-float data as a guide to commercial concentration. However, sink-float fractionation *(40, 41)* is valuable for appraising liberation, and recent sink-float data on the 8- to 35-mesh sizes of a typical Big seam sample are given in table 10.

TABLE 10.—*Sink-float of run-of-mine Big seam ore*

	Weight, percent	Fe, percent	CaO, percent	Insoluble, percent
8- to 10-mesh, wt., percent, 6.5:				
Float, 2.90 sp. gr	9. 8	8. 4	18. 5	53. 9
2.90–3.34	31. 9⎫64. 0	23. 4	17. 0	33. 6
3.34–3.70	32. 1⎰	35. 0	11. 8	25. 5
3.70–4.20	23. 6⎱26. 2	43. 5	9. 3	17. 4
Sink 4.20 sp. gr	2. 6⎰	54. 4	5. 0	9. 6
Composite	100. 0	31. 2	13. 1	28. 5
10- to 14-mesh, wt., percent, 13.8:				
Float 2.90 sp. gr	15. 3	7. 4	13. 6	66. 0
2.90–3.34	30. 7⎫56. 3	23. 3	16. 8	35. 6
3.34–3.70	25. 6⎰	35. 9	12. 2	25. 3
3.70–4.20	25. 3⎱28. 4	44. 1	8. 8	16. 6
Sink 4.20 sp. gr	3. 1⎰	54. 6	4. 6	9. 6
Composite	100	30. 3	12. 7	32. 0
14- to 20-mesh, wt., percent, 13.6:				
Float 2.90 sp. gr	27. 0	6. 3	14. 1	63. 6
2.90–3.39	29. 4⎫44. 1	20. 4	17. 9	34. 0
3.34–3.90	14. 7⎰	33. 8	12. 0	20. 8
3.70–4.20	23. 7⎱28. 9	46 0	5. 7	14. 2
Sink 4.20 sp. gr	5. 2⎰	57. 4	1. 9	6. 6
Composite	100. 0	26. 6	12. 3	33. 9
20- to 28-mesh, wt., percent, 16.0:				
Float 2.90 sp. gr	24. 0	6. 1	14. 6	62. 8
2.90–3.34	24. 3⎫37. 7	20. 7	17. 3	32. 2
3.34–3.70	13. 4⎰	34. 0	13. 0	18. 6
3.70–4.20	25. 8⎱38. 3	46. 1	5. 7	14. 0
Sink 4.20 sp. gr	12. 5⎰	57. 6	1. 9	7. 0
Composite	100. 0	30. 0	11. 2	29. 9
28- to 35-mesh, wt., percent, 13.2:				
Float on 2.90 sp. gr	22. 4	5. 8	16. 7	59. 3
2.90–3.34	21. 7⎫34. 1	20. 0	18. 5	30. 6
3.34–3.70	12. 4⎰	35. 0	13. 9	18. 8
3.70–4.20	23. 6⎱43. 5	45. 8	6. 9	14. 7
Sink 4.20 sp. gr	19. 9⎰	57. 7	1. 4	9. 8
Composite	100. 0	32. 3	11. 5	27. 7

Although the part that floated on 2.90 sp. gr. is only about 60 percent insoluble matter, it would be acceptable as tailing. The next fraction, 2.90 to 3.34 sp. gr., has to be reground.

In all of the sizes, in fractions with a density of 3.34 to 3.70 the "CaO" and "Insoluble" came so near neutralizing each other that the tenor of about 33.4 percent iron ranks these fractions as of a little better grade than marginal, as regarded by some operators; the excess acidity is least in the finest sizes.

The regression of excess acidity in 3.34 to 3.70 specific gravity is as follows:

Mesh:	Fe, percent	Excess acidity, percent
8 to 10	35.0	13.7
10 to 14	35.9	13.1
14 to 20	33.8	8.8
20 to 28	34.0	5.6
28 to 35	35.0	4.9

The iron content of each size is about the same; hence the regression makes the finest sizes the most desirable.

Some ores have shown an excess alkalinity in the fine sizes at this density. "Some" is used advisedly, because in elaborate sink-float of eight samples from the Big seam some show a reversed regression in the 3.34 to 3.70 fraction with the coarse sizes more alkaline than the fine sizes, and others show a high excess acidity throughout. Possibly this offers a clue for further identification of different parts of the seam. When these features are linked with the particle size of sea sand and petrographic characteristics, particularly a study of the fossils, the variety of seams will be better understood.

The material heavier than 3.70 specific gravity is of acceptable grade and is in the order of 35 percent weight.

Another approach to evaluation of the degree of liberation is presented in table 11, where the amount requiring regrinding is 72.8 percent at 8- to 10-mesh and 19.5 percent at 150- to 200-mesh.

TABLE 11.—*Weight percent, size by size, requiring regrinding*

[Specific gravities, between 2.90 and 3.70]

Sieve size, mesh	Weight, percent	Number of samples examined	Sieve size, mesh	Weight, percent	Number of samples examined
8 to 10	72.8	3	48 to 65	31.7	5
10 to 14	52.7	3	65 to 100	29.8	4
14 to 20	41.6	8	100 to 150	23.4	4
20 to 28	35.8	8	150 to 200	19.5	4
28 to 35	32.4	8	200 to 270	12.5	4
35 to 48	32.3	5	270 to 325	12.1	4

Truly the degree of liberation of minerals of different specific gravities can be delineated by sink-float, but there is another feature that sink-float does not recognize. It is particle shape, and shape may make the difference between success and failure in some gravity-concentration problems. Shape will be considered on page 88.

To learn more about the degree of dispersion of the limestone, which constitutes about 20 percent of the mill feed, heavy liquid dissections (table 12) were made at specific gravities of 2.72 and 3.14 to segregate the limestone between those densities. The products taken for the test were typical table concentrates from concentration tables 1, 2, and 3 (treating the three coarsest classifier spigot products), a spigot product from the concentration classifier treating a portion of middling from table 1, and a grab sample of table tailings. The intermediate-gravity fraction (2.72 to 3.14 sp. gr.) should contain all of the liberated limestone. The column "Distribution, CaO, percent" shows that, of the CaO present, the *maximum* that could be in a liberated condition ranged from 8.8 percent (spigot) to 43.3 percent (tailing). The preponderance of locked "Fe" and "Insol." beclouded the experiment and

demonstrated that reduction must be carried to flotation size for good liberation of limestone. Lee *(20)* showed that by flotation a minus 100-mesh limestone concentrate containing only slightly more insoluble matter than the best dolomite of the district could be obtained from magnetic log-washer tailings. Lee reported recoveries of about 80 to 85 percent, which points to good liberation at 100-mesh. The Birmingham red ores do not vary greatly, so it may be assumed that for the present ore a clean limestone is liberated at some size between 35- and 100-mesh.

TABLE 12.—*Fractionation of Big seam concentrates and tailings*

CONCENTRATES

Product	Specific gravity	Weight, percent	Assay, percent			On 35-mesh, weight, percent	Quartz sand, weight, percent	Distribution, CaO, percent
			Fe	CaO	Insol.			
Spigot from middling at table 1.	Float 2.72_____	2.4	4.0	3.6	87.3	97.0	_____	1.1
	2.72–3.14_____	5.7	13.9	12.2	55.4	96.9	_____	8.8
	Sink 3.14_____	91.9	46.1	7.7	14.9	98.6	_____	90.1
	Composite___	100.0	43.3	7.9	18.9	98.5	12.3	100.0
Table 1_____	Float 2.72_____	4.5	3.6	9.1	78.2	_____	_____	5.2
	2.72–3.14_____	8.6	12.7	17.9	46.5	_____	_____	19.6
	Sink 3.14_____	86.9	48.9	6.8	12.1	_____	_____	75.2
	Composite___	100.0	43.7	7.9	18.0	_____	11.3	100.0
Table 2_____	Float 2.72_____	2.2	3.8	12.3	72.2	15.4	_____	3.0
	2.72–3.14_____	9.6	12.1	22.9	39.6	15.4	_____	24.4
	Sink 3.14_____	88.2	48.0	7.4	11.3	29.6	_____	72.6
	Composite___	100.0	43.6	9.0	15.4	27.9	7.1	100.0
Table 3_____	Float 2.72_____	1.8	3.9	12.9	79.8	2.5	_____	2.6
	2.72–3.14_____	8.3	11.4	25.7	33.2	3.9	_____	23.4
	Sink 3.14_____	89.9	48.7	7.5	11.0	27.4	_____	74.0
	Composite___	100.0	44.8	9.1	14.1	25.0	8.1	100.0

DESLIMED TAILINGS

	Float 2.72_____	22.6	4.0	7.3	80.3	73.2	_____	12.3
	2.72–3.14_____	31.9	11.5	18.2	48.1	68.5	_____	43.3
	Sink 3.14_____	45.5	35.7	13.1	20.0	75.8	_____	44.4
	Composite___	100.0	20.8	13.4	42.6	72.9	_____	100.0

For further sink-float tests, a sample of ferruginous sandstone (table 13) was taken from the quarry above the Muscoda operations of the Tennessee Coal, Iron & Railroad Co. at Bessemer, Ala. It contains no lime and is a loosely consolidated sandstone consisting of sand grains cemented with iron oxide and clay. The sample contained 24.7 percent Fe, 2.9 percent Al_2O_3, 57.0 percent SiO_2, and 0.014 percent P. The absence of limestone accounts for the loose texture. It was ground in stages through 28-mesh in a ball mill and proved to be so very soft and crumbly that only 0.5 percent remained on 48-mesh. Although the iron was somewhat segregated in the finest size, the grade was very poor due to a segregation of clay; the alumina assay of the finest size was 4.5 percent, the equivalent of 11.4 percent kaolin. The phosphorus was segregated in the finest size.

TABLE 13.—*Screen analysis of ferruginous sandstone for sink-float fractionation*

Screen size, mesh	Weight, percent	Assay, percent			
		Fe	P	Al₂O₃	SiO₂
−28+48	0.5	23.9	0.004	2.8	59.1
−48+65	2.3	15.6	.009	1.4	73.0
−65+100	24.6	12.7	.007	1.4	78.1
−100+150	12.5	15.5	.006	2.3	72.7
−150+200	13.0	25.1	.010	2.5	57.9
−200+270	5.9	28.6	.008	2.8	51.0
−270+325	9.0	32.2	.007	2.6	47.1
−325	32.2	35.2	.029	4.5	37.0
Composite	100.0	24.7	.014	2.9	57.0

In table 14 the sink-float fractionation of the sizes from 48- to 325-mesh showed gradual increase in liberation of the iron with decrease in size. From the coarsest to the finest sizes, the assay of the floats on 2.95 specific gravity decreased from 5.5 to 2.7 percent iron. The grade

TABLE 14.—*Sink-float fractionation of ferruginous sandstone*

Screen size, mesh	Weight, percent	Assay, percent			
		Fe	P	Al₂O₃	SiO₂
Specific-gravity increments					
−48+65:					
Float on 2.95	59.4	5.5	0.009	0.8	89.3
2.95–3.3	21.2	19.9	.009	1.6	65.8
3.3–3.7	8.0	34.3	.016	2.4	44.3
3.7–4.2	6.9	43.2	.008	3.7	27.5
Sink in 4.2	4.5	54.2	.009	3.7	12.6
Composite	100.0	15.6	.0095	1.4	73.0
−65+100:					
Float on 2.95	71.0	5.1	0.006	1.0	90.3
2.95–3.3	15.4	20.0	.009	1.8	67.1
3.3–3.7	4.8	32.4	.006	3.1	45.9
3.7–4.2	3.8	43.0	.007	4.9	25.7
Sink in 4.2	5.0	56.0	.004	3.9	9.5
Composite	100.0	12.7	.0065	1.4	78.1
100+150:					
Float on 2.95	69.8	4.1	0.006	1.3	90.8
2.95–3.3	8.6	19.1	.010	3.8	65.1
3.3–3.7	2.6	31.4	.004	5.0	45.5
3.7–4.2	4.3	40.1	.012	7.2	27.7
Sink in 4.2	14.7	57.4	.004	4.0	8.8
Composite	100.0	15.5	.0062	2.3	72.7
−150+200:					
Float on 2.95	53.0	3.3	0.005	1.2	91.7
2.95–3.3	6.9	18.3	.015	3.9	64.2
3.3–3.7	2.5	28.9	.008	5.6	45.9
3.7–4.2	3.0	39.5	.018	7.0	28.6
Sink in 4.2	34.6	58.3	.017	3.6	8.2
Composite	100.0	25.1	.0103	2.5	57.9
−200+270:					
Float on 2.95	48.5	3.0	0.005	1.5	91.9
Sink in 2.95	51.5	53.7	.005	3.3	13.9
Composite	100.0	29.1	.005	2.4	51.7
−270+325:					
Float on 2.95	45.8	2.7	0.003	1.6	92.4
Sink in 2.95	54.2	55.3	.007	3.3	11.8
Composite	100.0	31.2	.0052	2.5	48.7

of the sink in 4.2 specific gravity was 58.3 percent Fe on the minus 150- plus 200-mesh size, whereas nearly this grade (53.7 percent Fe) was present in the sink 2.95 on the minus 200- plus 270-mesh product. The minus 200-mesh sizes were the only ones from which a "float" of tailing grade and a "sink" of concentrate grade were separated without an intermediate product. Thus sink-float has shown a difficult problem in liberation.

An examination from the lightest to the heaviest throughout all the respective sizes in table 14 shows a definite trend in the ratio of alumina to iron (Al_2O_3:Fe); the alumina content is relatively higher in the lightweight products. At first this was taken as information about the genesis of the ore. Seemingly the argillaceous material was the first to settle and coat the grains of gangue. This theory would hold if all the grains of gangue were quartz; but examination shows that the gangue has a few grains of feldspar, and the suspicion arises that it contributes enough alumina to explode the theory on genetic relationship. Then, too, discrete particles of clay might have been present.

Another orderly variation appears in table 14, but it can be explained. Reference is made to the "floats" on 2.95 specific gravity in the respective sizes. There they are progressively leaner in iron down through the finer sizes. This is brought about by better liberation, whereas the coarser material contains clusters of very fine sand cemented together with enough hematite to augment the iron content. In all the tests with both red ores and ferruginous sandstones, the coarser "floats" on 2.9 specific gravity had a higher iron content than the finer sizes. This must be recognized when milling for a low-grade tailing. As a reminder of the extreme fineness of the sand in the ferruginous sandstone, reference should be made back to the bottom of table 2.

In discussing table 10, depicting the calcareous red ores, attention was called to the grade of the particles with a density between 3.34 and 3.70. They were thought to be of a little better grade than marginal. Here in table 14 the same observation could be made about the iron content of material of that same density, but the condemning fact is that no friendly lime is present to replace a part of the silica.

PRECISION BATCH JIGGING

In our every-day life we have seen old houses torn down to make way for new ones. The old houses had given good service, but the new ones were more desirable because they incorporated facilities not known when the old houses were built. Similarly, sink-float must, for some purposes, give way to an improved method of laboratory batch jigging (29).

In the jigging of sized samples of coal this method has given results that were almost identical to sink-float; but when the shape of the particles is adverse, sink-float is a poor guide in predicting results in gravity concentration. This was brought out in testing one of the iron ores. The jigging method was developed only recently. That is why the "old house" has such a prominent part in this report.

Data from sink-float fractions and batch jigging of six sized products of an iron ore are given in table 15. The sink-float tests required several days. Bringing all the solutions to the desired specific gravity was in itself time-consuming, and the use of the thallium malonate-formate at a specific gravity of 4.20 was irksome, because the solution

had to be maintained at considerably above room temperature. Furthermore, the finest sizes were dilatory in reaching equilibrium on account of their slow settling (rising) rate. On the other hand, when the simple art of batch jigging has been mastered, the operation is all but child's play.

In making up the columns under "Batch jig," the "Fe, percent" was taken from cumulative curves based upon the grade and weight, percent, of the several cuts obtained from the jig. That is, the batch-jig data were set up in a form that would correlate with the sink-float data.

Numbers in parentheses appear in the top row of the table. These are independent of those taken from the cumulative curve. They represent the topmost layers, respectively, taken from the basket.

TABLE 15.—*Comparison of precision batch jigging with sink-float fractionation of iron ore*

Specific gravity	8- to 10-mesh			10-to 14-mesh		
	Weight, percent	Fe, percent		Weight, percent	Fe, percent	
		Sink-float	Batch jig		Sink-float	Batch jig
	(8.3)		(14.3)	(12.9)		(11.8)
Float 2.90	10.7	9.0	15.4	25.2	6.9	14.3
2.90–3.32	23.4	24.3	26.5	21.0	22.8	24.8
3.32–3.70	48.4	36.3	36.6	32.8	36.7	35.0
3.70–4.20	17.1	44.7	42.0	20.6	45.7⎱	43.3
Sink 4.20				.4	55.3⎰	
Composite	100.0	32.0	32.9	100.0	28.1	29.4
	14- to 20-mesh			20- to 28-mesh		
	(11.8)		(12.0)	(9.7)		(12.2)
Float 2.90	31.0	6.9	13.9	29.0	7.3	14.1
2.90–3.32	19.4	22.2	20.9	16.9	20.6	27.2
3.32–3.70	23.4	36.1	33.8	14.5	33.5	29.0
3.70–4.20	24.1	46.7⎱	44.3	27.1	46.3	42.3
Sink 4.20	2.1	55.3⎰		12.5	56.9	51.4
Composite	100.0	27.3	27.9	100.0	30.1	30.8
	28- to 35-mesh			35- to 48-mesh		
	(11.1)		(13.9)	(3.8)		(8.9)
Float 2.90	27.0	8.1	19.3	29.5	6.1	16.5
2.90–3.32	16.9	21.8	26.8	19.3	17.8	21.1
3.32–3.70	12.6	34.4	27.6	19.6	33.1	24.1
3.70–4.20	25.1	47.3	37.7	18.6	46.3	43.7
Sink 4.20	18.4	58.3	55.0	13.0	57.3	53.1
Composite	100.0	32.8	32.8	100.0	27.8	28.7

Table 15 is so easy to interpret that the reader is left to make his own deductions about leaning too much on sink-float when commercial concentration is anticipated. However, attention may be directed to the work on the 14- to 20-mesh sample. There it is shown that the float on 2.90 specific gravity amounts to 31.0 percent weight and has a tenor of 6.9 percent iron. Nothing could be more ruinous than to take this as a criterion in table concentration. The jig work showed a tenor of 13.9 percent iron (at 31.0 percent weight), and that was obtained by superlative laboratory work.

The numbers in the parentheses show that, when only 11.8 percent

weight was taken off for a tailing, the tenor was as high as 12.0 percent iron. Thoughtful examination and judicious sampling of well-performing tables justify the expressed admonition that in a study of gravity concentration sink-float fractionations may be fallacious because they fail to take particle shape or porosity into account.

The jigging described on page 84 is closely related to this work.

WET AND DRY CRUSHING, GRINDING, SCUFFING, AND ATTRITION SCRUBBING

The presence of clay makes wet crushing preferable to dry crushing. About one-tenth of the content of the Big seam ores is clay, and it is not visible megascopically and scarcely visible microscopically. As this clay is dispersed quite uniformly, the idea occurred that water would make the clay slake and soften the whole mass. This idea was put to test, and wet grinding was found to be much easier than dry grinding. The clayey nature increases the permeability and wettability of the particles. Pieces of Big seam ore of a size between ¾- and ½-inch were used, and one group of particles was saturated with water before testing while the other group was tested dry. The pieces in each group were broken in a testing machine and the crushing strengths recorded. The average crushing strength of 26 pieces dry was 208 pounds, while that of the other 26 pieces wet was 110 pounds. The ratio of average crushing strength of the two groups was 1.89. Another lot of particles of ½- to ⅓-inch size that was treated in the same way gave similar results. The average resistance of 20 particles crushed dry was 136 pounds, while that of 20 particles wet was only 72 pounds. Here the ratio again was 1.89. Variations were observed in the resistance of individual pieces; but on the average, the dry ore was nearly twice as resistant as the wet.

The variation in resistance of the ores with variation in moisture content can be detected when as little as 1 percent of water is added to a thoroughly dried sample. In practice, a flow sheet that incorporates wet grinding rather than dry will have an advantage with respect to power consumption, capacity, and type of grind.

The degradation mentioned on page 63 is a factor in wet crushing and grinding.

Grinding is a broad and general term to signify the comminution of material that has been reduced to about 8-mesh by crushing. This limit is so arbitrary that no two workers would agree upon it. Here it is set low in the scale of sizes because most grinding is done in drum (ball) mills, finishing at 100-mesh or finer, a size at which a higher reduction ratio is not conducive to the most efficient grinding. As many drum mills receive feeds coarser than 8-mesh, the conclusion would be that they are used as both crushers and grinders, which indeed they are, following a practice that may be justified on account of surrounding conditions.

The term "differential" is used in the study of particle-size reduction of a mixture of minerals in which some are reduced more readily than others. As most materials are heterogeneous, it follows that most grinding is differential, but we pay attention to it only when it can be used advantageously. In the reduction of the red iron ores the slime may be taken off as a concentrate if principles of differential grinding are followed closely. Simultaneously the quartz grains are shelled out

to make a tailing. The principles followed determine whether the work is called grinding, scuffing, self-grinding, or attrition scrubbing.

In scuffing (exaggerated differential grinding) a supplementary grinding medium is used that must be small enough to permit the hard particles to escape but large enough to polish all particles and reduce the soft ones. Self-grinding applies where no grinding medium is added and the hard particles are expected to abrade the soft. In this abrasion process the drum had best be called a tube mill, and the abraded material may be expected to be very fine.

Attrition scrubbing (37) and self-grinding accomplish the same end, but attrition scrubbing is much faster because it permits a greater power input. In self-grinding the material is tumbled in a drum, but in attrition scrubbing it is given a severe beating in a tank containing a rotor with rubber-covered impellers. Water is added in all methods to give fluidity. The energy input in self-grinding in a drum is limited by the weight of the charge and the centrifugalizing, which has a definite upper limit. In attrition scrubbing the energy input depends solely upon the facilities available. Attrition scrubbing and self-grinding have one advantage over other methods in that, except for the presence of water, every unit of occupied volume is filled with material that is the object of the work. But the use of "advantage" requires further thought. The method was not designed for iron-ore concentration. It was designed for polishing hard minerals, and no thought was given to the byproduct of polishing. It was wasted.

Wet self-grinding will yield a slime concentrate of satisfactory grade from any of the Red Mountain ores, but the power consumption in terms of concentrate (byproducts of polishing) is excessive beyond consideration. An example is the self-grinding of a mill tailing from Big seam ore. The tailing had a tenor of 20.2 percent iron. The recovery in slime was 28 percent of the iron of 52.0-percent grade. The net energy consumption in terms of material treated was excessive, being 19.2 hp.-hr. per ton; furthermore, when calculated in terms of concentrate, it was even worse. The input was as high as 186.8 hp.-hr. per ton. This limits self-grinding of the tailings to the laboratory. A further objection would be the excessive fineness of the slime. If there were a premium on fineness and the area of new surface were determined, the energy input per unit of new surface might fall within the range of that obtained by customary practice. But this is far removed from the interests of the mineral dressers of iron ore.

A presentation of methods for obtaining differential grinding follows but it is not intended to encroach upon the next chapter, which deals with concentration tests.

The nature of the Birmingham ores invited efforts to obtain liberation of the hematite without fracture of the quartz. This is why so much effort in the early work was made to obtain a satisfactory differential reduction. In that work much time was spent in comparing the results with mediums of different diameters. Actually, later developments removed some of the fright about fractured quartz, but the work recorded here still has a value in portraying the nature of the ore.

An examination of table 16 shows that in "ball milling" the use of ¼- by ⅛-inch punchings was the most satisfactory of the series because it yielded 27.6 percent weight through 100-mesh having a grade of 43.0 percent iron when calculated on a self-fluxing basis. The feed

TABLE 16.—*Differential grinding of red iron ore with various types of grinding mediums*

Size, mesh	Feed	Weight, percent							
		Discharge after grinding with—							
		1.5-in. rods	2-in. balls	Empty pipes	⁷⁄₁₆-in. balls	⁵⁄₁₆-in. balls	¼-in. balls	⅜-in. slugs	¼-x⅛-in. punchings
+8	5.2		0.2		2.7	1.7	2.6	2.6	2.1
10	8.9		.4		5.2	5.2	5.9	5.0	4.8
14	12.6	0.6	1.2		7.1	7.8	9.5	7.9	8.0
20	15.9	1.2	5.7	0.4	9.9	11.0	12.6	11.8	11.8
28	15.6	5.3	15.4	5.5	9.6	11.4	13.0	11.6	11.9
35	16.1	18.3	21.8	22.8	12.2	11.4	12.0	14.6	13.7
48	12.1	23.2	17.5	21.3	10.4	9.1	9.0	10.8	10.2
65	8.1	17.9	12.5	16.8	8.7	6.6	5.5	7.2	6.3
100	5.5	11.3	8.1	9.9	7.1	6.5	3.9	5.1	3.6
−100		22.2	17.2	23.3	27.1	30.3	26.0	23.4	27.6
	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
ANALYSES									
−100 {Fe		38.96	38.63	39.20	42.56	43.78	44.34	44.50	45.76
CaO		6.86	8.26	7.56	8.85	7.25	7.19	9.32	8.13
Insoluble		25.32	23.36	21.40	13.90	14.08	13.68	12.10	11.52
Self-fluxing Fe		29.0	30.4	31.4	39.0	39.4	40.0	42.2	43.0
+100 {Fe	31.52	29.14	30.05	29.18	27.39	26.18	27.01	27.55	26.09
CaO	9.23	9.86	9.43	9.74	9.36	10.08	9.95	9.20	9.65
Insoluble	32.10	33.86	33.91	35.34	38.86	39.92	38.57	38.21	39.94

sample contained 31.5 percent iron, 9.2 percent lime, and 31.9 percent insoluble, but if limestone were added to make it self-fluxing the tenor would be only 22.4 percent iron. Thus, on the slimed portion, the grading up in iron content was about 2:1. A further illustration of grinding (scuffing) the same material with the small punching, and for longer periods, is given in table 17. An 18-minute scuffing period gave

TABLE 17.—*Effect of prolonged scuffing of red iron ore using ¼- by ⅛-inch punchings*

Size, mesh	Feed	Weight, percent		
		Length of grind, minutes		
		3	9	18
Plus 8	5.2	2.4	1.8	1.7
10	8.9	4.8	4.8	4.5
14	12.6	8.0	7.9	7.2
20	15.9	12.6	11.2	10.3
28	15.6	13.1	10.7	8.5
35	16.1	15.9	11.4	10.1
48	12.1	13.0	7.5	5.3
65	8.1	8.7	3.8	2.5
100	5.5	5.0	2.2	1.5
Minus 100		16.5	38.7	48.4
	100.0	100.0	100.0	100.0
ANALYSES				
−100 {Fe		46.26	45.25	44.60
CaO		6.72	9.54	10.24
Insoluble		11.28	11.76	12.18
Self-fluxing Fe		42.7	43.5	43.2
+100 {Fe	31.52	28.61	22.85	19.19
CaO	9.23	9.72	9.04	8.27
Insoluble	32.10	36.21	44.94	50.77

48.4 percent weight of minus 100-mesh material that had a tenor of 43.2 percent iron self-fluxing. The "lime" yielded so readily that the excess acidity was only 1.94 (12.18-10.24) percent.

In the 18-minute test 68.5 percent of all the iron was removed, and the excess acidity was not increased by the longer period. The lime-stone seems to serve as a buffer, so that the process can be carried to great length before the quartz yields. This is exemplified by many tests in which the excess acidity decreases as the work progresses and is more marked when the ore contains soft clay that fouls the slime in the first step.

In attrition-scrubbing tests (table 18) the ore had been reduced to 8-mesh. The most severe treatment (test 4) yielded 56.0 percent weight as slime that had a tenor of 44.8 percent iron self-fluxing. In the different tests the disintegration of the limestone progressed with the "Insoluble," so that there was no increase in excess acidity.

TABLE 18.—*Attrition scrubbing of red ore, in percent*

Test	Removed as slime	Fe	CaO	Insol.	Self-fluxing Fe
1	35. 9	52. 2	5. 2	10. 2	47. 9
2	40. 6	51. 3	5. 8	11. 1	46. 9
3	51. 4	49. 5	7. 0	12. 0	45. 4
4	56. 0	48. 4	7. 6	12. 1	44. 8

TABLE 19.—*Scuffing Spaulding red ore, in percent*

	Weight	Fe	CaO	Insol.	P
−200 mesh, 1st scuff	18. 6	47. 0	8. 3	12. 3	0. 33
−200 mesh, 2d scuff	35. 8	46. 0	9. 1	12. 7	. 34
−200 mesh, 3d scuff	10. 7	44. 7	9. 5	14. 2	. 36
Sink 2.95	8. 6	36. 7	10. 9	24. 4	. 61
Composite concentrate	73. 7	45. 0	9. 2	14. 2	. 37
Tailing	26. 3	6. 2	9. 5	73. 7	. 12
Original ore	100. 0	34. 7	10. 3	29. 8	0. 35

In table 19 the scuffing of red ore may be taken as typical of differential grinding; the first output is usually richer than the succeeding slimed products. But to say that the hematite is ground differentially

TABLE 20.—*Scuffing ferruginous sandstone with punchings, in percent*

Mesh	Punchings 90 percent ¼- by ⅛-in., 10 percent ⅛- by ⅛-in.		
	Weight	Assay, Fe	Distribution of—Fe
−200	44. 6	42. 6	82. 3
+200	55. 4	7. 4	17. 7
Composite	100. 0	23. 1	100. 0
	Punchings 100 percent ⅛- by ⅛-in.		
−200	59. 5	41. 8	96. 4
+200	40. 5	2. 0	3. 6
Composite	100. 0	25. 8	100. 0

is not telling the whole story. Some hematite particles are hard and high grade, and in this test remained in "Sink 2.95" as hard, disseminated grains.

If no limestone is present and the quartz grains are very small, as in the ferruginous sandstone, steel punchings are pretty severe unless they are very small. Some results are given in table 20 for illustration. The recovery was very high, and the grading up was almost twofold.

The next sample (table 21) is hard to classify. The fineness of the sand would classify it as a ferruginous sandstone—the weight, percent, "peaks" at 65-mesh—but the lime content approaches that of the red ores. The dilemma that it introduces is a reminder that still more must be learned about the Red Mountain ores. However, the physical value in testing is in accord with expectations; the screen analysis is of

TABLE 21.—*Attrition scrubbing of ferruginous sandstone*

DRILL CORE (411 TO 437 FEET), NEAR SPAULDING MILL

Mesh	Screen analysis, in percent				
	Wt.	Fe	CaO	Insol.	P
10	18.8	22.1			
14	17.4	21.0			
20	12.2	20.6			
28	8.2	20.4			
35	9.0	22.1			
48	10.7	21.4			
65	10.8	21.0			
100	5.0	28.0			
150	2.4	33.3			
200	2.1	35.9			
−200	3.8	35.7			
Composite	100.0	22.7	6.9	51.7	0.096

SLIME REMOVED THROUGH 200-MESH SCREEN

Product	Periods, minutes	Weight, percent	Fe, percent	CaO, percent	Insol., percent	P, percent
No. 1 slime	4	20.6	45.8	3.9	22.2	0.099
No. 2 slime	4	12.2	42.6	5.0	24.7	.121
No. 3 slime	6	6.9	36.3	7.3	29.9	.145
No. 4 slime	10	4.6	29.2	11.4	33.1	.167
No. 5 slime	15	3.5	22.6	15.5	34.0	.176
No. 6 slime	20	4.5	20.6	15.8	37.2	.170
No. 7 slime	30	2.7	21.4	19.3	29.9	.172
No. 8 slime	45	3.6	17.1	19.6	35.5	.165
No. 9 slime	90	2.7	16.3	21.4	33.5	.165
Sand residue		38.7	2.7	3.2	89.9	.033
Composite		100.0	22.9	6.7	51.8	.095

SIZING, WEIGHT, PERCENT

Mesh	Sand residue (38.7 percent wt.), after attrition scrubbing	Quartz-sand (44.1 percent wt.), after acid treatment
14	Tr	3.0
20	2.5	3.8
28	7.3	6.3
35	9.8	9.8
48	23.2	21.7
65	39.0	31.2
100	12.2	9.3
150	3.7	2.2
200	2.3	1.8
−200	Tr	10.9
	100.0	100.0

the usual type, and the limestone, by persisting to the last, shows a reluctance to disintegrate so that the lime content from the ninth run was 21.4 percent, whereas it was only 3.9 percent in the first. Of course, what is said about the lime may be stated in the converse by saying that the sand shells out more readily than the limestone.

The phosphorus content was lowest in the first period, appearing particularly low when considered with respect to the iron. Most of it was ground out before the residue was reached. The deportment of the phosphorus suggests that pressing for a low-grade tailing will give a relative increase of phosphorus in the concentrate.

Although the phosphorus content of the "sand residue" is very low its ratio to the iron is about the same as in No. 9 slime. Positively the phosphorus was reluctant to yield to the grinding.

ECONOMIC UTILIZATION OF SLIME CONCENTRATE

Aside from the power costs involved in reducing the hematite to slime size, another major obstacle stands in the way of this method of concentration. This obstacle is the cost of thickening and filtering the slime in preparation for sintering. The Bureau of Mines has found that either electrophoresis or centrifuging will dewater a slime concentrate produced by attrition scrubbing to about 50 percent moisture, but these methods are, at present, too costly to have commercial interest.

DIRECT MAGNETIC CONCENTRATION

Before the turn of the century, Wilkens and Nitze (16) practiced magnetic concentration without roasting. Separations could be made but the process was not practical.

In the present work a laboratory Stearns magnetic separator was used to fractionate a table concentrate which had a tenor of 43.7 percent iron. The results were as follows, in percent:

Products	Weight	Fe	CaO	Insol.
Magnetic fraction No. 1	49.4	50.0	7.3	12.0
Magnetic fraction No. 2	16.3	46.5	9.3	13.7
Magnetic fraction No. 3	12.2	39.8	11.9	18.3
Tailing fraction No. 3	12.3	37.1	12.6	21.3
Tailing fraction No. 2	4.9	27.3	14.4	32.5
Tailing fraction No. 1	4.9	12.8	14.3	54.8
	100.0	43.7	9.5	17.3

Repeated passes were required. The roughing was at 110 volts and the cleaning at 60 volts. About one-half of the table concentrate was brought up to 50.0 percent iron. So much of the limestone was locked with the hematite that the excess acidity was only 4.7 percent in the cleanest concentrate.

IRON DETERMINATION BY SPECIFIC GRAVITY

For mill work a method of "quick iron" analysis requiring only 15 or 20 minutes has been developed. It is for quick results on a few special samples that would become engulfed for 24 hours in a mass of routine if sent to the chemical laboratory. The method has been described by Feld and associates (30).

Inasmuch as the predominant gangue minerals in the Big seam ores

have a specific gravity of about 2.6 it was thought that a definite relationship must exist between specific gravity and iron content. The first attempts to appraise the iron content by means of the usual specific-gravity measurement were discouraging because of an occasional "wild" result. This was attributed to incomplete wetting of the porous grains after they had been dried. The method, as now employed, is accurate to about +0.5 percent, and all trouble from air bubbles and incomplete wetting has been eliminated. The method consists of nearly filling a 100-ml. tared volumetric flask with the undried sample and filling to the mark with water. The weight of ore and water is recorded. The sample is then transferred to a pan, most of the water decanted, and the ore taken to dryness on the hot plate. The weight of the dry ore is subtracted from the weight of ore plus water. The remainder, subtracted from 100, gives the volume of the ore, and the weight of dried ore divided by this volume equals specific gravity. The iron assay corresponding to the determined specific gravity is found by reference to either a table (table 22) or a curve (fig. 15).
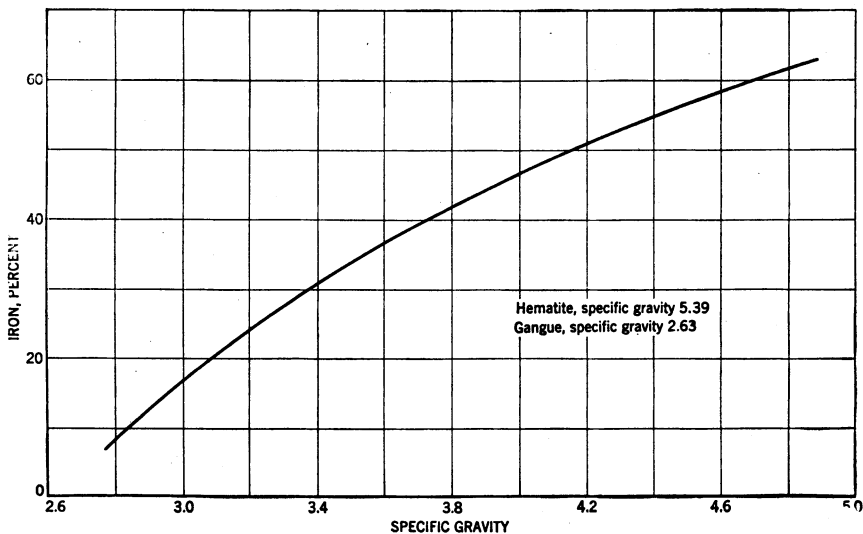

FIGURE 15.—Specific-gravity—iron relationships for Big seam ore.

The technique employed in filling and emptying the flask deserves mention. To avoid wet ore plugging the neck, a funnel is inserted in the flask and filled to overflowing with water. The ore is then placed in the funnel and water added at intervals to keep the funnel full. (There is always some leakage of water between the sides of the funnel and the mouth of the flask.) The ore sifts in with practically no tendency to plug. After the sample is in the flask the surplus water is poured off and the water level adjusted to the mark by adding water dropwise from a burette.

In emptying the flask, much time can be saved if it is filled to the rim with water and inverted in a pan nearly full of water. The ore particles will empty completely. Sluicing with a fine stream of water to dislodge the particles will be unnecessary.

TABLE 22.—*Specific gravity—iron relationship for Big seam ore*

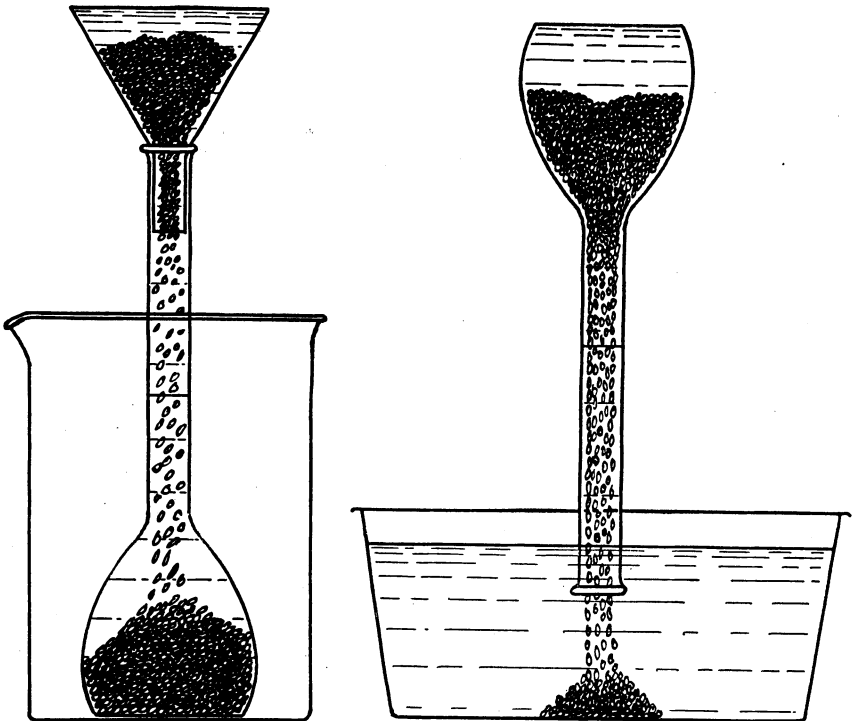| Specific gravity	Iron, percent	Specific gravity	Iron, percent	Specific gravity	Iron, percent
2.90	12.6	3.50	33.9	4.10	48.9
2.92	13.5	3.52	34.5	4.12	49.4
2.94	14.3	3.54	35.1	4.14	49.8
2.96	15.2	3.56	35.6	4.16	50.2
2.98	16.0	3.58	36.2	4.18	50.6
3.00	16.8	3.60	36.7	4.20	51.0
3.02	17.7	3.62	37.3	4.22	51.4
3.04	18.5	3.64	37.9	4.24	51.8
3.06	19.2	3.66	38.4	4.26	52.2
3.08	20.0	3.63	38.9	4.28	52.6
3.10	20.7	3.70	39.5	4.30	53.0
3.12	21.5	3.72	40.0	4.32	53.4
3.14	22.2	3.74	40.5	4.34	53.8
3.16	22.9	3.76	41.0	4.36	54.1
3.18	23.6	3.78	41.5	4.38	54.5
3.20	24.3	3.80	42.0	4.40	54.9
3.22	25.0	3.82	42.5	4.42	55.2
3.24	25.7	3.84	43.0	4.44	55.6
3.26	26.3	3.86	43.5	4.46	56.0
3.28	27.0	3.88	44.0	4.48	56.4
3.30	27.7	3.90	44.5	4.50	56.7
3.32	28.4	3.92	44.9	4.52	57.1
3.34	29.0	3.94	45.3	4.54	57.5
3.36	29.7	3.96	45.8	4.56	57.9
3.38	30.3	3.98	46.2	4.58	58.2
3.40	31.0	4.00	46.7	4.60	58.5
3.42	31.6	4.02	47.2	4.62	58.9
3.44	32.2	4.04	47.6	4.64	59.2
3.46	32.8	4.06	48.0	4.66	59.6
3.48	33.4	4.08	48.5	4.68	59.9

FIGURE 16.—Specific-gravity flask. Left, Method of filling flask with ore; right, method of emptying ore from flask.

The filling and emptying technique is illustrated in the accompanying sketch (fig. 16).

<div align="center">LIME IN ORE</div>

Some operators look upon lime (limestone) as a semieconomic material and feel that its rejection is undesirable. Others, with evidence not too well-founded, regard the lime in the ore bodies as inherently impure and, even at a loss of some of the iron, would prefer to reject it and get their flux from the stone quarries which yield a high grade of dolomite. In favor of those who would not reject the lime, it must be said that for a given tenor in iron content the material that has even a small amount of limestone is the more acceptable because the lime fluxes its equivalent weight of silica. Repeated beneficiation tests have shown that satisfactory concentrates can be made more easily when lime is inherent (disseminated) in the ore, because liberation of the hematite is never perfect and if lime is present some of it will remain locked with the hematite and occupy zones that otherwise would be acidic. In the fine sizes limestone is more tenacious than quartz in clinging to the hematite.

Those who would reject the limestone because they feel that it is inherently impure have available two kinds of noncalcareous ores—those supposedly laid down without the presence of limestone, like the Ida and the ferruginous sandstones, and those from which the lime has been leached. The former ores have the undesirable feature of having the silica cemented to the hematite without lime as a buffer, and the latter contain clay (which is thought by some to have originated through dissolution of the limestone but more likely is carried in by circulating waters).

In favor of rejection of lime one positive statement can be made. Its removal will reduce the tonnage to the sintering plant. Then there is a probability that a sinter made without lime is more acceptable. In the examination of Big seam samples, the limestone therein has been found to be of good grade. This observation warrants the statement that the district would be more fortunate if the hematite were as inherently clean as the limestone in the Big seam.

In the examination of limestone, the following test, corroborated by others, is indicative of its purity. A high-lime product was taken from a tailing in dry magnetic separation. The analysis was: 11.5 percent Fe, 32.0 percent CaO, 23.9 percent insol., and 0.129 percent P. It was roasted, ground to 100-mesh, and treated by magnetic separation; and the magnetic tailing was treated by flotation to take off limestone in the froth:

	Percent				
	Wt.	Fe	CaO	Insol.	P
Magnetic iron concentrate	22.4	41.6	11.9	15.8	0.194
Flotation lime concentrate	49.6	2.0	53.0	1.4	.073
Flotation tailing	28.0	5.7	12.1	66.8	.142
	100.0	11.9	32.4	22.9	.142

A lime product with a tenor of 53.0 percent CaO (56.0 percent CaO would have indicated pure limestone) and 1.4 percent insoluble was

obtained. Phosphorus also was dropped out of the limestone froth. The limestone froth contained 81.3 percent of all of the lime. It was 53/56 part pure. A hematite product of equal purity is difficult to make.

CALCULATIONS FOR SELF-FLUXING GRADE

Burchard *(1)* makes the following statement, which cannot be passed as casually as a first reading might indicate. He says: "Although the soft ore carries more iron, some of the hard ore has the advantage of containing enough lime to flux the silica that it contains." He gives a table, which for convenience has been condensed and rearranged and to which has been added a column for the iron content on a self-fluxing basis.

Analyses of bedded hematite, showing gradation from hard to soft ore, in percent

Ore No.	Fe	CaO	Insoluble	Self-fluxing Fe
1	37. 0	19. 2	11. 0	
2	45. 7	8. 7	17. 5	39. 0
3	50. 4	4. 6	18. 1	40. 5
4	54. 7	. 5	19. 3	41. 0

Ore 1 has excess alkalinity, and the furnace man will have to use his discretion in blending it with his other stock piles, but the others may be subjected to rigid appraisal. That appraisal is in the last column, where it appears that on a self-fluxing basis the iron content of each of the three would be about the same. A calculation for self-fluxing is always necessary if limy ores are in the group undergoing scrutiny.

When only the iron content (self-fluxing) is required the answer can be obtained by two settings of the slide rule after subtracting lime from insoluble for excess acid. The calculation will be made step-by-step, so that the simplicity of slide-rule work will be apparent.

Assume that the analysis from the chemical laboratory is as shown in column A below. In B, CaO has been converted to $CaCO_3$ by

	A	B	C
Fe	34. 7	34. 7	34. 7
CaO	10. 3		
$CaCO_3$		18. 4	18. 4
Limestone			33. 2
Insol.	28. 9	28. 9	28. 9
		82. 0	115. 2
		18. 0	18. 0
		100. 0	133. 2

assuming that the carbonate is 56 percent CaO, and in C the required units of limestone have been added. B totals 82 units, and since the calculation must be made on the basis of 100 units, 18 has to be added. C adds up to 115.2, and the 18 units added in B must also be added in C. Then the iron content of the self-fluxed material is

$$\frac{100}{133.2} \times 34.7 = 26.0 \text{ percent iron. A general formula is:}$$

$$\frac{100 \times \text{Fe, percent}}{100 + 1.78} \text{(excess insoluble).}$$

The short calculation is: Divide excess acidity by 0.56, put "1" in front of the quotient, and find how many times the new number is contained in the iron content of the sample. The quotient will be the iron content of the material after enough limestone is added to make it self-fluxing.

Sometimes the CaO and insoluble content are required. The steps are enumerated so that if the results differ from those that would be obtained by the operator, he can see wherein the difference lies.

Calculation to a self-fluxing basis is expressed by the formula:

$$\text{Fe, percent self-fluxing} = \frac{\text{Fe assay}}{100 + \dfrac{\text{insoluble, percent} - \text{CaO, percent}}{0.56}}.$$

The derivation of the formula may be illustrated by the following example. As before, the ore has the assay:

	Percent			Percent
Fe	34.7	$\approx$	Fe_2O_3	49.6
CaO	10.3	$\approx$	$CaCO_3$	18.4
Insol.	28.9	$=$	Insol.	28.9
				96.9

100.0 percent —96.9 percent = 3.1 percent of unspecified constituents.

(For example, the red ores of the Birmingham district are known to contain minute quantities of MgO, Na_2O, K_2O, P_2O_5, Mn_3O_4, TiO_2, and H_2O.) It is assumed that 1 unit of CaO is required to flux 1 unit of insoluble. In the ore there is 18.6 percent (insoluble, percent — CaO, percent) of excess insoluble, and to flux this there must be added 18.6 percent of CaO in the form of limestone. 18.6÷0.56=33.2 percent of C,CO_3 required. After the limestone is added, the resulting ore-limestone mixture will have the composition:

Fe_2O_3	49.6	
$CaCO_3$	51.6	(18.4 plus 33.2).
Insol.	28.9	
Unspecified	3.1	
	133.2	

Dividing each of these values by 133.2 to convert back to a 100.0-percent basis gives

	Percent			Percent	
Fe_2O_3	37.2	$\approx$	Fe	26.0	self-fluxing
$CaCO_3$	38.8	$\approx$	CaO	21.7	self-fluxing
Insol.	21.7	$=$	Insol.	21.7	self-fluxing
Unspecified	2.3				
	100.0				

Table 22 contains the result of 34 concentration tests in which the self-fluxing grade is shown.

APPRAISAL OF ORES FOR CONCENTRATION

MINIATURE-SCALE CONCENTRATION TESTS

In the early part of the work, small-scale tests were made on many samples to get the first glimpse of their nature—to find out if physical separations were at all possible. The methods were a combination of ball milling, scuffing, self-grinding, screening, and sink-float. The results are shown in table 23 in the geologic order of the seams—Upper

TABLE 23.—*Miniature-scale concentration tests*

UPPER FERRUGINOUS SANDSTONE

Sample No.	Location	Products		Percent					
			Wt.	Assay iron	Distribution iron	Assay lime	Assay insol.	Self-fluxing basis, Fe	Increase in iron from furnace due to concentration
03–A	Sparks Gap	Crude ore	100.0	29.3	100.0	0.06	55.6	14.7	
		Concentrate	51.0	47.4	82.4	.09	28.8	31.3	113
		Tailing	49.0	10.5	17.6	.04	83.6		
02–A	Raimund No. 1	Crude ore	100.0	26.9	100.0	.25	59.2	13.1	
		Concentrate	47.5	48.3	85.3	.38	27.2	32.7	150
		Tailing	52.5	7.5	14.7	.14	88.2		
01–1–A	Muscoda No. 4	Crude ore	100.0	29.0	100.0	1.2	58.0	14.4	
		Concentrate	45.7	54.3	85.6	1.5	26.5	37.6	161
		Tailing	54.3	7.7	14.4	.9	84.5		
Q(1025E)	Muscoda No. 5	Crude ore	100.0	31.8	100.0	.16	51.4	16.6	
		Concentrate	54.9	52.8	91.1	.20	20.0	39.0	135
		Tailing	45.1	6.2	8.9	0.11	89.6		
Q(1025F)	Muscoda No. 5	Crude ore	100.0	27.0	100.0	.11	55.7	13.5	
		Concentrate	49.3	48.6	88.7	.13	26.5	33.0	144
		Tailing	50.7	6.0	11.3	.09	84.1		
1/2–A	Sloss No. 1	Crude ore	100.0	26.7	100.0	.08	57.9	13.1	
		Concentrate	46.1	48.2	83.2	.08	24.8	33.5	156
		Tailing	53.9	8.3	16.8	.08	86.3		
2–A	Wenonah No. 9	Crude ore	100.0	26.0	100.0	.06	59.1	12.7	
		Concentrate	45.3	47.9	83.4	.11	24.7	33.3	162
		Tailing	54.7	7.9	16.6	.02	87.6		
2¼–A	Wenonah No. 10	Crude ore	100.0	22.9	100.0	2.1	59.4	11.3	
		Concentrate	41.8	45.2	82.4	2.2	25.6	31.9	182
		Tailing	58.2	6.9	17.6	2.1	83.6		
8–D	Old Hammond	Crude ore	100.0	27.2	100.0	.13	51.1	14.2	
		Concentrate	50.6	45.4	84.5	.06	19.9	33.5	136
		Tailing	49.4	8.5	15.5	.20	83.1		

IDA SEAM

Sample No.	Location	Products	Wt.	Assay iron	Distribution iron	Assay lime	Assay insol.	Self-fluxing basis, Fe	Increase in iron from furnace due to concentration
02¼–A	Raimund No. 3	Crude ore	100.0	37.0	100.0	0.0	45.0	20.5	
		Concentrate	64.0	55.3	95.5	.0	18.1	41.8	104
		Tailing	36.0	4.6	4.5	.0	93.0		
5–D	Walker Gap	Crude ore	100.0	44.0	100.0	.18	34.1	27.4	
		Concentrate	76.5	56.2	97.6	.20	16.1	43.7	60
		Tailing	23.5	4.5	2.4	.14	92.9		
9½–C	Ruffner No. 1	Crude ore	100.0	32.6	100.0	.37	50.1	17.3	
		Concentrate	59.7	52.1	95.3	.48	20.4	33.4	122
		Tailing	40.3	3.8	4.7	.22	94.0		

MIDDLE FERRUGINOUS SANDSTONE

Sample No.	Location	Products	Wt.	Assay iron	Distribution iron	Assay lime	Assay insol.	Self-fluxing basis, Fe	Increase in iron from furnace due to concentration
5½–A	Valley View No. 2	Crude ore	100.0	20.0	100.0	0.14	68.8	9.0	
		Concentrate	38.3	46.8	89.5	.11	27.5	31.5	250
		Tailing	61.7	3.4	10.6	.16	94.7		
5¾–B	Valley View No. 1	Crude ore	100.0	19.9	100.0	.23	69.4	8.9	
		Concentrate	38.7	46.7	90.8	.19	28.5	31.0	248
		Tailing	61.3	3.0	9.2	.25	95.5		
9–C	Ruffner No. 3	Crude ore	100.0	34.2	100.0	1.1	45.7	19.0	
		Concentrate	60.4	51.3	90.6	1.1	19.6	33.6	103
		Tailing	39.6	8.1	9.4	1.0	85.6		
9½–D	Ruffner No. 1	Crude ore	100.0	28.9	100.0	.31	55.9	14.5	
		Concentrate	51.0	51.3	90.7	.42	21.5	37.3	157
		Tailing	49.0	5.5	9.3	.20	9.16		

BIG SEAM

Sample No.	Location	Products	Percent						
			Wt.	As-say iron	Dis-tri-bu-tion iron	As-say lime	As-say in-sol.	Self-flux-ing basis, Fe	In-crease in iron from furnace due to concen-tration
02½-A	Potter No. 1	Crude ore	100.0	51.8	100.0	0.03	21.2	37.6	
		Concentrate	91.5	55.8	98.5	.03	15.1	44.1	17
		Tailing	8.5	8.9	1.5	.06	86.4		
02-C	Raimund No. 1	Crude ore	100.0	32.0	100.0	18.4	15.3		
		Concentrate	73.1	40.2	91.8	15.0	11.8		
		Tailing	26.9	9.7	8.2	27.6	24.8		
¾-02-A	Sloss No. 2	Crude ore	100.0	35.1	100.0	19.0	10.8		
		Concentrate	81.0	41.0	94.5	15.6	7.6		
		Tailing	19.0	10.0	5.5	33.5	24.4		
3-02-B	Ishkooda No. 11	Crude ore	100.0	35.6	100.0	10.5	25.5	28.1	
		Concentrate	71.8	45.5	91.8	9.5	11.7	43.7	56
		Tailing	28.2	10.4	8.2	13.0	60.6		
4-02-C, 4-02-D	Ishkooda No. 13	Crude ore	100.0	32.0	100.0	9.7	32.2	22.8	
		Concentrate	63.5	45.0	89.3	9.0	13.4	41.7	83
		Tailing	36.5	9.4	10.7	10.9	64.9		
4¼-A	Ishkooda No. 14	Crude ore	100.0	42.2	100.0	.86	33.9	26.6	
		Concentrate	73.6	54.8	95.5	.88	14.5	44.2	66
		Tailing	26.4	7.1	4.5	.79	87.8		
4¼-01-C, 4¼-01-D	do	Crude ore	100.0	33.3	100.0	8.4	32.8	23.2	
		Concentrate	62.9	46.9	88.5	7.0	13.9	41.8	80
		Tailing	37.1	10.3	11.5	10.7	64.8		
4¼-01-E	do	Crude ore	100.0	30.6	100.0	10.8	32.3	22.1	
		Concentrate	62.6	43.6	89.3	9.4	15.5	39.3	78
		Tailing	37.4	8.7	10.7	13.0	60.3		
4¼-02-D, 4¼-02-E	do	Crude ore	100.0	33.5	100.0	9.4	30.7	24.3	
		Concentrate	66.3	45.6	90.3	8.8	13.0	42.4	74
		Tailing	33.7	9.7	9.7	10.6	65.5		
1095B	Spaulding No. 2	Crude ore	100.0	33.6	100.0	9.6	27.6	26.9	
		Concentrate	71.4	46.5	93.3	8.9	12.1	44.0	64
		Tailing	28.6	8.4	6.7	11.4	66.4		
4½-A	do	Crude ore	100.0	35.9	100.0	10.2	26.5	27.8	
		Concentrate	67.1	48.8	91.2	8.2	10.8	46.6	68
		Tailing	32.9	9.6	8.8	14.3	58.5		
4½-02-A	do	Crude ore	100.0	38.4	100.0	14.7	15.6	37.8	
		Concentrate	71.9	50.4	94.3	8.6	8.9	50.0	32
		Tailing	28.1	7.7	5.7	30.3	32.7		
5½-C	Valley View No. 2	Crude ore	100.0	30.3	100.0	10.1	34.9	21.0	
		Concentrate	58.7	45.3	87.9	8.7	14.7	41.0	95
		Tailing	41.3	8.9	12.1	12.1	63.7		
8-B	Old Hammond	Crude ore	100.0	34.0	100.0	.24	46.9	18.6	
		Concentrate	59.8	52.8	93.0	.25	17.4	40.4	117
		Tailing	40.2	5.9	7.0	.22	90.8		
¾-01-C	Sloss No. 2	Crude ore	100.0	23.7	100.0	15.3	32.5	18.1	
		Concentrate	62.5	32.3	85.3	13.1	22.9	27.5	52
		Tailing	37.5	9.3	14.7	19.0	48.3		
3-02-C	Muskoda No. 11	Crude ore	100.0	24.2	100.0	18.7	26.8	21.1	
		Concentrate	57.9	34.0	81.4	16.2	15.8	34.2	62
		Tailing	42.1	10.7	18.6	22.2	41.9		
4½-B	Spaulding No. 2	Crude ore	100.0	27.8	100.0	14.1	27.9	21.8	
		Concentrate	63.4	37.9	86.4	13.0	15.8	36.1	65
		Tailing	36.6	10.3	13.6	16.1	53.7		
10-B	Ruffner No. 2	Crude ore	100.0	36.5	100.0	11.3	23.3	30.0	
		Concentrate	70.6	47.8	92.7	8.3	12.2	44.7	49
		Tailing	29.4	9.0	7.3	18.6	50.1		

Ferruginous, Ida, Middle Ferruginous, Big seam, and Irondale. The arrangement in each group is from south to north.

When these tests were made (1941) it was not expected that plant operations could reach the efficiency shown thereby; but, although the grade of the tailings is a little lower than could be expected in commercial work, it appeared that if the operators were willing to put up with much of the concentrate as slime the results could be used as a basis when contemplating investments in concentrators.

One feature of the table may seem to be too bold because the estimates deal with the performance of the blast furnace about which the authors do not claim knowledge gained by experience. At the right in the table are shown both crude ore and concentrate calculated to a self-fluxing basis, and the estimated increased output of the furnace is based on the hypothesis that the output varies directly as the grade of the feed. If this is not correct the operator, at any rate, is supplied with the figures to which he can apply factors in accord with his experience. Two of the Big seam samples had excess alkalinity, so that the calculations were not feasible.

No attempt was made to control the course of the lime in these tests. Unless lime is in great excess and no siliceous ores are available to flux it, probably it should not be removed. If it is removed it must be regarded as a tailing and as a tailing it must be charged with the loss of some entrained iron. Because of its economic value as a flux, lime should be regarded as a semieconomic mineral; in the opinion of some of the operators, this waste should be avoided.

The samples that do not contain lime may be consistently compared with each other without giving thought to fluxing; but if they are to be compared with those having lime, or limy samples are to be compared with each other, the calculations for self-fluxing grade must be made.

In the milling of a limy ore the assay for iron in the feed and concentrates does not give a true picture. In fact, it might be misleading. Example from milling results, in percent:

	Fe	CaO	Insoluble	Self-fluxing Fe
Feed	33.4	10.4	29.7	24.8
Concentrate	46.7	6.8	16.0	40.0

A study of analyses of "Feed" and "Concentrate" shows a grading up of only 13.3 percent in iron, whereas on a self-fluxing basis the grading up is 15.2 percent iron. The latter figure is the better guide.

Where enough lime is present to be of consequence, the example just given is representative of all the tests with respect to lime; the lime content of the concentrate is less than that of the feed. Possibly the loss of lime is of no consequence and is not a waste, but the deportment should be recognized. To a large degree the course of the lime can be controlled by flotation.

A swap of some of the "insoluble" for "CaO" in the concentrate would be attractive, but nature has not been kind in the scrambling and we have not been artful in the unscrambling. The inherent "insoluble" in the hematite must take part of the blame.

In studying table 23 close attention must be given to "self-fluxing."

JIGGING

At the outset in the consideration of jigging, the failure of all of the

red ores to show significant segregation of the iron in particles coarser than 10-mesh and the failure of the ferruginous sandstone to yield segregation in particles coarser than 28-mesh must be emphasized as adverse to coarse concentration. The corollary is that the jigging of sizes coarser than those mentioned above would be useless. Arbitrarily the ferruginous sandstone may be eliminated from discussion because grinding for liberation would give a feed that is irrefragably too fine for commercial jigging, but the red ores may be considered.

In considering the red ores, one must not lose sight of the fact that worth-while concentration can be attained only by removal of the sea sand. The fineness and quantity of the sand in the red ores (Big seam) have been fully delineated in table 3. Only a trace of sand is coarser than 10-mesh; the mean size is at about 42-mesh, being a little coarser than this in the north end of the range. But that is not the whole criterion required because sea-sand analysis in table 3 includes all the sand after both physical and chemical methods of liberation have been used. The physical separation is the important one. Data on the amount of sand physically liberated may be obtained by referring to the sink-float work in table 10. Only half of its float on 2.9 specific gravity is "insoluble."

The poor opportunity for rejection of waste may be illustrated by examining table 11, which shows that 72.8 percent of the 8- to 10-mesh size is unliberated middling. The quantity of true middling decreases with size but even at the 65- to 100-mesh size (which is too fine for jigging) nearly 30 percent consists of locked particles. It must be granted that, by a struggle, a jig could be made to yield some tailings and also some concentrates, but a large amount of middlings would accrue for the regrind department. The data do not encourage expectations for a satisfactory removal of sea sand without further grinding, and the adverse feature about jigging is that the material, being already too fine for jigging, must suffer further grinding for more liberation.

Jigging has not been condemned by analysis of data sheets. It has been given a thorough trial in the laboratory and in continuous runs with current plant products. After hard daily struggles the jig was abandoned.

Many series of jig tests were run on natural ore through 10-mesh and on enriched table middling cuts. The natural ore contained too much of the intermediate specific gravity, as shown by table 7, but the middling (p. 86) was more amenable.

In jigging the natural ore a 13- by 16-inch jig with a 3-mm. punched jig cloth was used. Usually it was loaded with 40 pounds of ¼- by ⅛-inch steel punchings for an artificial bed. The results of hutching with different amounts of feed rates ranging from 650 to 3,150 pounds per hour are given:

Hutch (feed, 33.0 percent Fe)

Test	Weight, percent	Pounds per hour	Fe, percent
1	0.8	5.2	53.7
2	4.5	97.0	49.4
3	4.9	44.0	51.6
4	5.5	36.0	51.0
5	7.0	147.0	48.5
6	10.6	220.0	46.1
7	16.4	520.0	42.8
8	28.4	425.0	41.6

They are arranged in the order of percentage of feed removed by the hutch. When the adjustment of the jig permitted only a small amount to pass the artificial bed and enter the hutch, a very fine mesh rich product (53.7 percent Fe) was obtained, but when capacity was sought the grade dropped to 41.6 percent iron.

A struggle was made to withdraw part of the coarsest size of satis-factory grade, but when the bed was opened enough to permit hutching of the coarse sizes the period of retention was so brief that stratification was poor; too much of the fines passed the artificial bed. A sizing analysis of the hutch product in test 7 shows this:

Mesh: *Weight, percent*
   10 to 14.. 7.0
   14 to 20.. 15.7
   20 to 28.. 19.0
   28 to 35.. 24.8
   35 to 48.. 19.3
   Minus 48.. 14.2
                                                                        100.0

The sizes finer than 35-mesh amounted to 33.5 percent weight.

Table middlings taken from a narrow fan next to the concentrate zone acted more favorably. The reason is brought out by the sizing analysis of sink-float products:

Middling

Mesh	Float, 2.9 sp. gr.; weight, percent, 14.3	Float, 2.9 to 3.7 sp. gr.; weight, percent, 33.9	Sink, 3.7 sp. gr.; weight, percent, 51.8
On 10			0.3
On 14			2.3
On 20	5.6		20.2
On 28	38.3		62.5
On 35	83.2		90.6
On 48	96.6		97.7
On 65	99.0		99.6
Minus 65	1.0		.4

The material is more suitable for jigging because the heaviest is the coarsest; in the float on 2.9 specific gravity only 38.3 is on 28-mesh but 62.5 percent of the same size is sink in 3.7 specific gravity. Unfortu-nately about one-third is of intermediate grade and cannot be satis-factorily beneficiated until it is reground. However the table in the plant had done a precision job in preparing a feed for a jig or a con-centration classifier. One of many jigging tests is shown:

	Hutch	Cup draw	Overflow
Weight _____percent___	53.4	21.7	24.9
Distribution of iron _____do____	61.2	20.2	18.6
Grade, Fe _____do____	45.7	37.2	29.8

The cup draw, set high, was used in all tests to draw off overly coarse material that would accumulate and tend to reduce the hutching rate. From a feed with a tenor of 39.2 percent iron, 53.4 percent weight was taken off containing 45.7 percent iron.

In plant work the jig had an insurmountable shortcoming. It could not adapt itself to changing conditions of particle size and feed rate. The amount and particle size of artificial bed, length of stroke, and
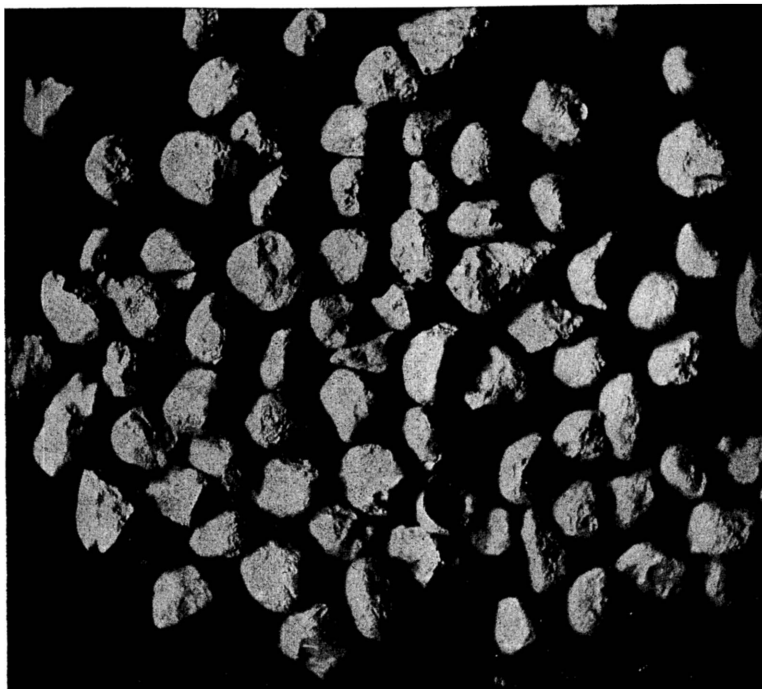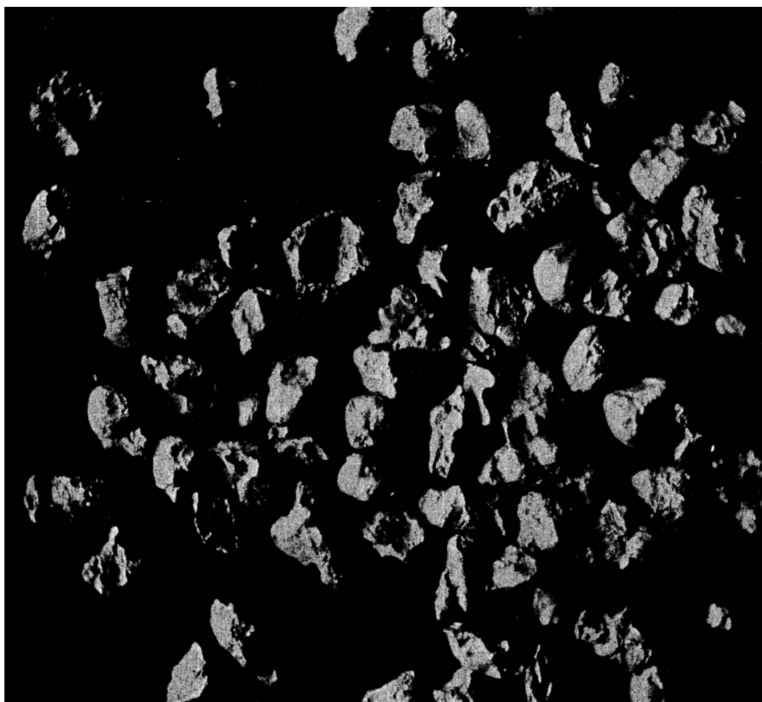
FIGURE 17.—Spherical quartz and irregular particles of ore in same jig stratum. Sink-float, 14- to 20-mesh. Stratum cut from precision batch jig, 11.5 percent weight, 15.7 percent Fe. Left, float, 2.90 sp. gr., 64 percent weight, 7.8 percent Fe. Right, sink, 2.90 sp. gr., 36.0 percent weight, 29.8 percent Fe.

water had to be set for a feed that did not vary. When a variation occurred the jig fizzled.

The adverse effect of irregular shapes is shown in figure 17, taken from a recent paper on precision jigging *(29)*. To obtain these, one of the upper strata taken from the jig was fractionated on a 2.90 specific-gravity solution. The "float" was veneered quartz that may be described as spherical; but the "sink," although it contained 29.8 percent iron, had come to equilibrium in the same stratum because of irregular shape with reentrant surfaces. Then, too, some of the particles were known to be porous. Thus jigging may be dismissed.

MILLING

General. Milling of red iron ores is not commonly practiced in the Birmingham district. Outside of the district the brown ores receive a cleaning in log washers (jigs are used at some places) to remove the sand and clay, but within the district there is (1945) only the gravity concentrator of the Republic Steel Co. at Spaulding and the roasting-and-magnetic separation plant of Sloss-Sheffield Steel & Iron Co. at Ruffner.

Gravity concentration and flotation. The Republic concentrator has been described by Lee *(27)*. In the original plans of the company there was to be a 1,000-ton-per-day pilot plant consisting of rod mill, classifiers, 24 tables, a thickener and 2 filters, and all the accessories required for gravity concentration. During the planning and construction period the "pilot-plant" idea was uppermost.

However, by the time the pilot plant was ready to run industry had a new impulse. The war was on! Production had to supersede experiments, and because the converted pilot plant could not supply the demand a much larger plant was built following the general plans of the first. A description of this plant or any other is not in the scope of the present writing, which deals with mineral-dressing characteristics.

Suffice it to say that the results confirmed those in the laboratory, which showed that much slime of the micron sizes would be made if comminution were carried far enough for satisfactory liberation.

The beneficiation of this slime for the blast furnace after it has been purged of fine quartz, thickened, and filtered is the unsolved part of the problem. It is a problem that, from the outset, will require large, expensive equipment.

The steps in concentration, including removal of the fine sand by flotation, are worked out and fully described by Clemmer and associates *(39)*. The method for flotation of the fine sand has proved its reliability during continuous test runs in the plant as well as the laboratory. The results of a laboratory pilot-plant run are given in table 24. The concentrate had a tenor of 47.7 percent iron and a very moderate excess of "insoluble."

The part recovered on the tables was only 31.0 percent of the total concentrate, and it was very fine; it was 97.0 percent through 20-mesh and as much as 20 percent through 200-mesh. This has to be accepted because as was shown frequently throughout the text of this bulletin fine grinding is required for liberation.

The filter cake was 58.0 percent minus 20 microns.

The table concentrate deserves more comment. Whereas numerous

TABLE 24.—*Pilot-plant test on calcareous red ore in which recirculated water was employed*

Product	Lb. per min.	Weight, per-cent	Assay, percent			Distribution, percent		
			Fe	CaO	Insoluble	Fe	CaO	Insoluble
Feed	3.32	----------	36.1	11.7	22.9	--------	--------	----------
Concentrates:								
Table	0.72	21.7	53.0	5.3	10.6	31.3	10.2	9.6
Flotation	.63	19.0	40.6	15.0	11.1	21.0	25.3	8.8
Slime	.99	29.8	48.3	8.8	10.9	39.2	23.3	13.6
Composite concentrates	2.34	70.5	47.7	9.4	10.9	91.5	58.8	32.0
Tailings:								
Table	.79	23.8	11.2	14.7	55.8	7.3	31.1	55.7
Flotation	.19	5.7	7.1	19.8	51.1	1.2	10.1	12.3
Composite tailings	.98	29.5	10.4	15.7	54.9	8.5	41.2	68.0
Composite feed	3.32	100.0	36.7	11.3	23.9	100.0	100.0	100.0

mentions have been made about the readiness with which the ore yields slime, this must be set right by saying that the table concentrate (that which survived the grinding) consisted of "fairly hard hematite."

Roasting and magnetic concentration. The new (1945) work being started to practice reduction roasting and magnetic separation on an operating scale is the sequel of the work of the Bureau published in 1927 *(19)*.

The plant is designed for soft ore, a large amount of which remains in the northern extremity of the district. On account of the absence of lime, the tenor of the concentrate is higher in iron than it would be if natural leaching of the lime had not occurred.

SUMMARY

Extending over a period of many years, the Bureau of Mines made intermittent examinations of the Red Mountain iron ores of Birmingham, Ala. The object was to devise methods of concentration. This work was followed by an organized effort, begun in 1941, to study the mineral-dressing characteristics of the ores of the five main seams. These seams, in the order of geologic sequence from top to bottom, are: Upper Ferruginous, Ida, Middle Ferruginous, Big, and Irondale. More than 100 samples were taken in key positions. Supplementing these samples, there were many hundred others and fractions thereof, all of which were examined.

The Big seam is the most important, the amount of the main constituents being about 34 percent Fe, 10 percent CaO, and 28 percent insoluble. All of these ores are characterized by the presence of particles of sea sand, which are cemented by hematite and clay and by limestone, if lime is present. They all crumble with a degree of readiness depending on characteristics. In doing so, they release hematite slime and veneered-sand grains. This veneer, after moderate scrubbing, may contribute as much as 5 percent of the iron content of the segregated grains of sand. Then, too, all hematite is of low grade. Particles with a tenor of more than 60 percent Fe are difficult to find; hence, an heroic effort is required to make high-grade concentrates and low-grade tailing. In those areas examined for the grade of limestone, the limestone was found to be of better grade than the hematite.

Fortunate it would be if nature had reversed this condition. Since enrichment by concentration depends wholly upon the removal of sea sand, much attention was given to its nature. The particle-size distribution by sieve analysis was about the same as would be found in a crushed homogeneous rock. A disconcerting factor in these analyses was the cracks in the sand grains permitting some of the particles to fall to pieces during the severe chemical treatment required to dissolve the hematite from interstices and crevices.

The sand in the Big seam has a top size at about 10-mesh, the coarseness increasing toward the north end of the range. The Ida seam has sand at about the same particle size as the Big seam, and the Upper Ferruginous is the finest, having a top size at about 28-mesh. The veneer on the sand in those seams with fine sand is less-persistent than on those with coarse sand.

In addition to the substances mentioned, the ores contain MgO, Na_2O, K_2O, TiO_2, and P_2O_5. The ignition loss on those samples thought not to contain limonite may be more than 1 percent. Accordingly, the usual assay for iron, lime, and insoluble may fall more than 5 percent short of a complete analysis.

The ores crumble readily in water, so that degradation may yield as much as 20 percent as a slime of superior grade, the grade depending upon the amount of liberated clay. The ore is porous. A dry sample may yield air bubbles for a long while after it has been submerged in water. This condition gives an apparent density adverse to gravity concentration, and another adverse condition is the pitted surface remaining after the sand grains have shelled out.

Although milling has been practiced at sizes corresponding to an 8- or 10-mesh screen, evidence is abundant that 50 percent of all the material between that size and 20-mesh is too low-grade for a concentrate and too high-grade for a tailing. Furthermore, at 35- to 48-mesh, one-third is middling.

Much effort was made to develop a plan for obtaining liberation without fracturing the quartz sand. Finally this was given up, and severe ball milling was resorted to, the subsequent steps being table concentration of the granular hematite and flotation of the fractured quartz from the slime. If the ore did not contain free clay and all of the quartz entering the slime had at least one face amenable to flotation, a slime of satisfactory grade could be made. A combined table and flotation concentrate from a representative Big seam sample was 47.7 percent iron, 9.4 percent CaO, and 10.9 percent insoluble. Plant operation is recommended.

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APPENDIX. SUITE OF PHOTOMICROGRAPHS OF SPECIMENS OF RED MOUNTAIN ORE

Throughout the course of several years a great many samples were examined in the petrographic laboratory. Some were thin sections, and others were granular material. Usually the records were preserved by means of photographs. Some of them are shown here in figures 18 to 27. Those from figures 18 to 23 inclusive are of the Big seam and figures 24 to 27, inclusive, are from a drill core described in table 21 (p. 75). The work was by Ernest E. Fairbanks, associate mineralogist.
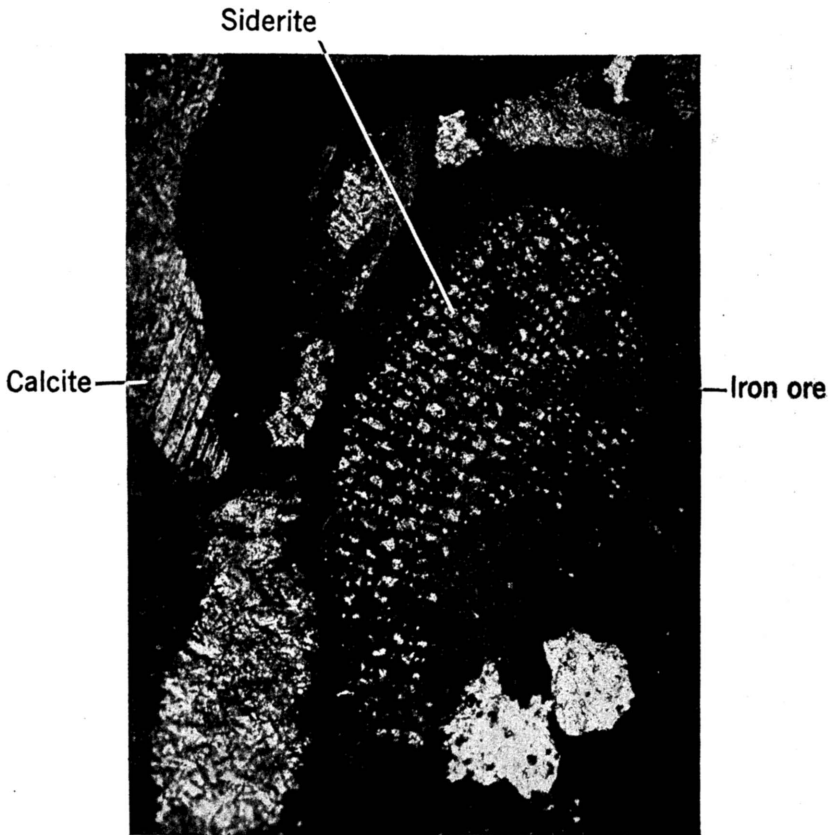
FIGURE 18.—Photomicrograph of thin section of Spaulding ore, showing siderite, rarely found in this ore, which has replaced and preserved fossil structure. 1 inch = 48-mesh; X 118; polarized light.
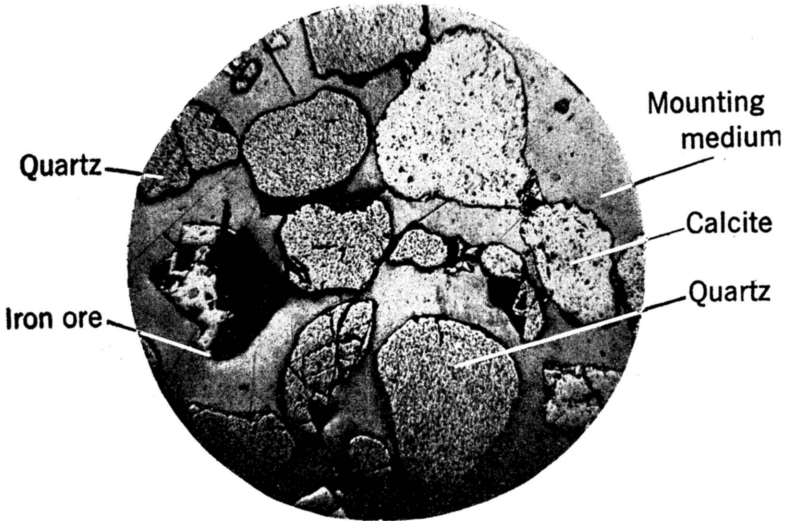
FIGURE 19.—Photomicrograph of Spaulding mill feed, showing a minor amount of ore locked with gangue materials. Quartz and calcite of this fraction are generally found unlocked from each other. The iron was found to consist chiefly of hematite, with some lepidocrocite and limonite intimately intermixed. 5/16 inch = 48-mesh; X 40; reflected light. Minus 28- plus 48-mesh; float, 2.9.
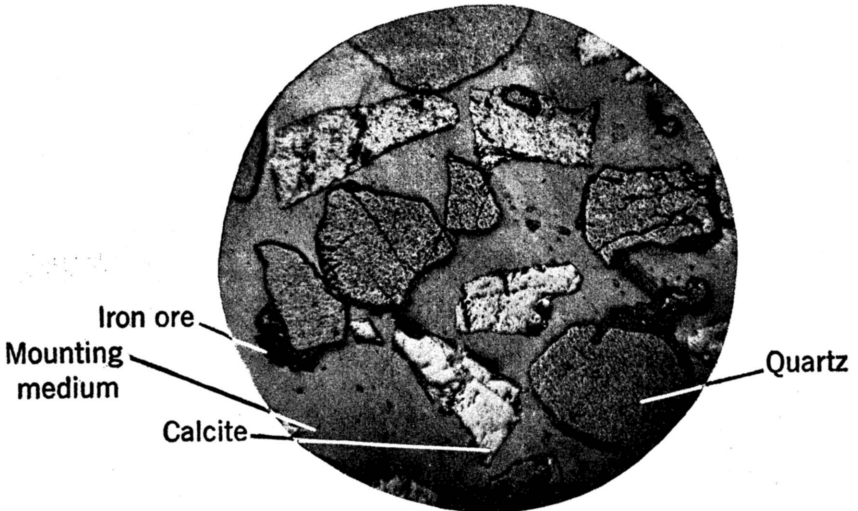
FIGURE 20.—Additional photomicrograph of Spaulding mill feed, showing extent of locking of iron ore with quartz and calcite. 5/16 inch = 48-mesh; X 40; reflected light. Minus 28- plus 48-mesh; float, 2.9.
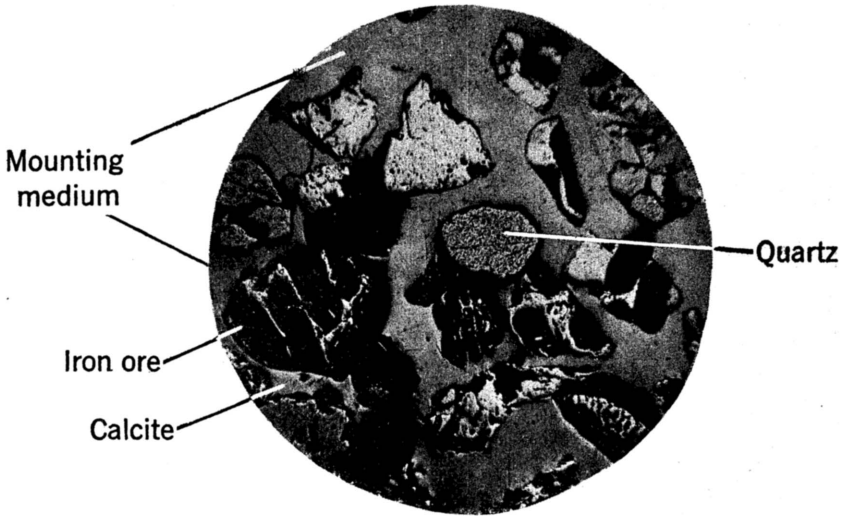
FIGURE 21.—Photomicrograph of Spaulding mill feed, showing interlocking of iron ore and gangue. 5/16 inch = 48-mesh; X 40; reflected light. Minus 28- plus 48-mesh; sink, 2.9; float, 3.7.

FIGURE 22.—Photomicrograph of Spaulding mill feed, showing locking of iron ore in the form of oölites (essentially hematite) with calcite. 5/16 inch = 48-mesh; X 40; reflected light. Minus 28- plus 48-mesh; sink, 2.9; float, 3.7.
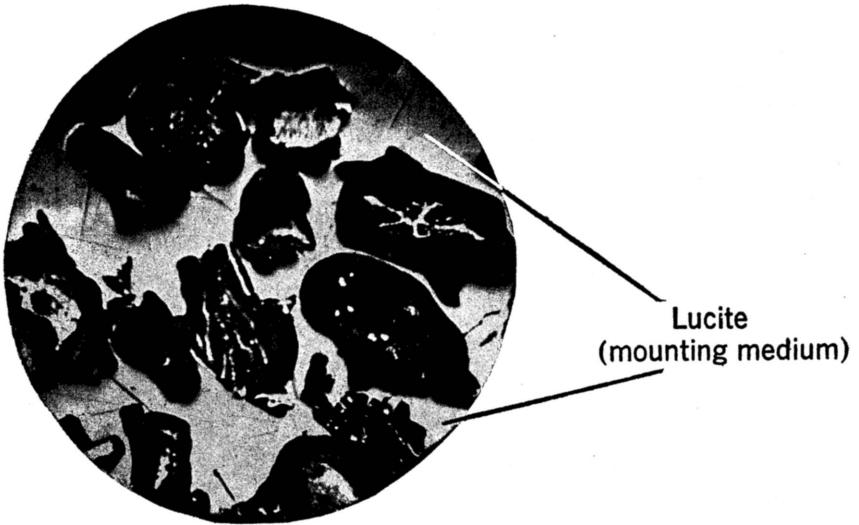
FIGURE 23.—Photomicrograph of Spaulding mill feed, showing gangue minerals extensively locked with iron ore. Note concentric banding of oölites, which have two types of nuclei. The first type consists of quartz-sand grains. The second consists of calcite, or more rarely siderite, pseudomorphs after fossils and usually preserving the organic structure. 5/16 inch = 48-mesh; X 40; reflected light. Minus 28- plus 48-mesh; sink, 3.7.
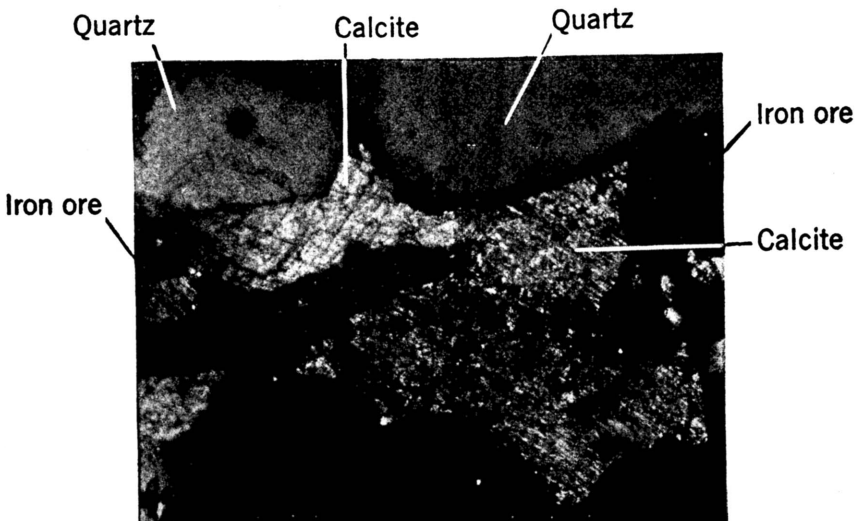

FIGURE 24.—Photomicrograph of thin section prepared from drill core, showing association of iron ore with calcite. 13/16 inch = 48-mesh; X 97; polarized light.
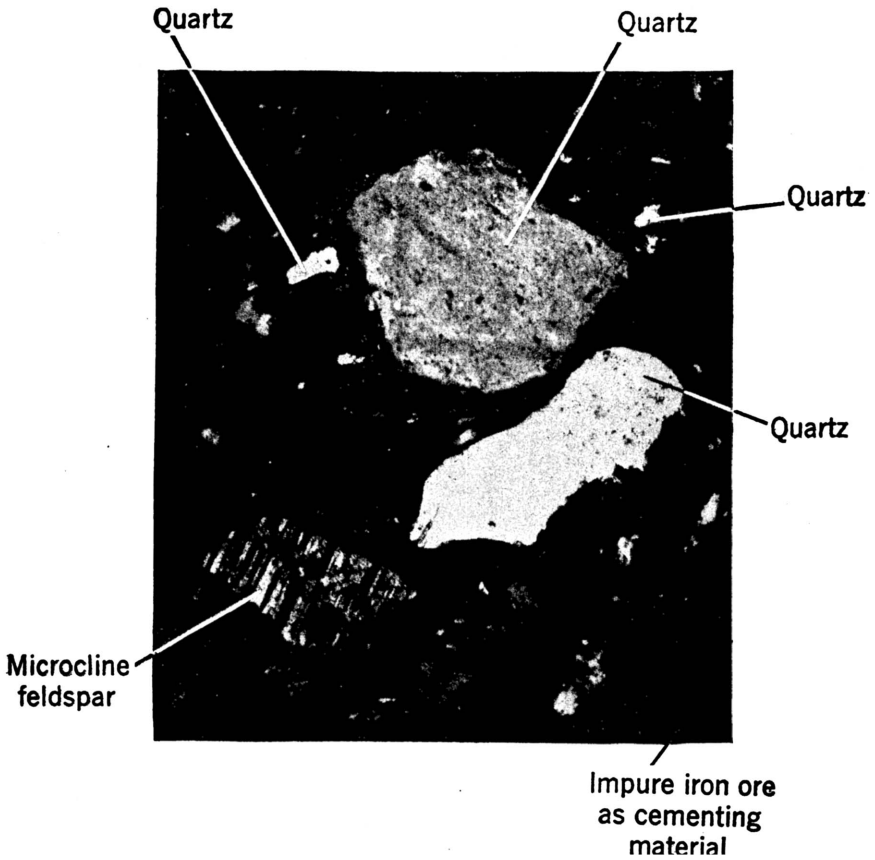
FIGURE 25.—Photomicrograph of thin section prepared from drill core, showing relatively large quartz and feldspar grains in a fine-grained groundmass consisting essentially of quartz and iron ore. The grains of the original sediment are cemented by iron oxide with possibly some secondary silice. 13/16 inch = 18-mesh; X 97; polarized light.
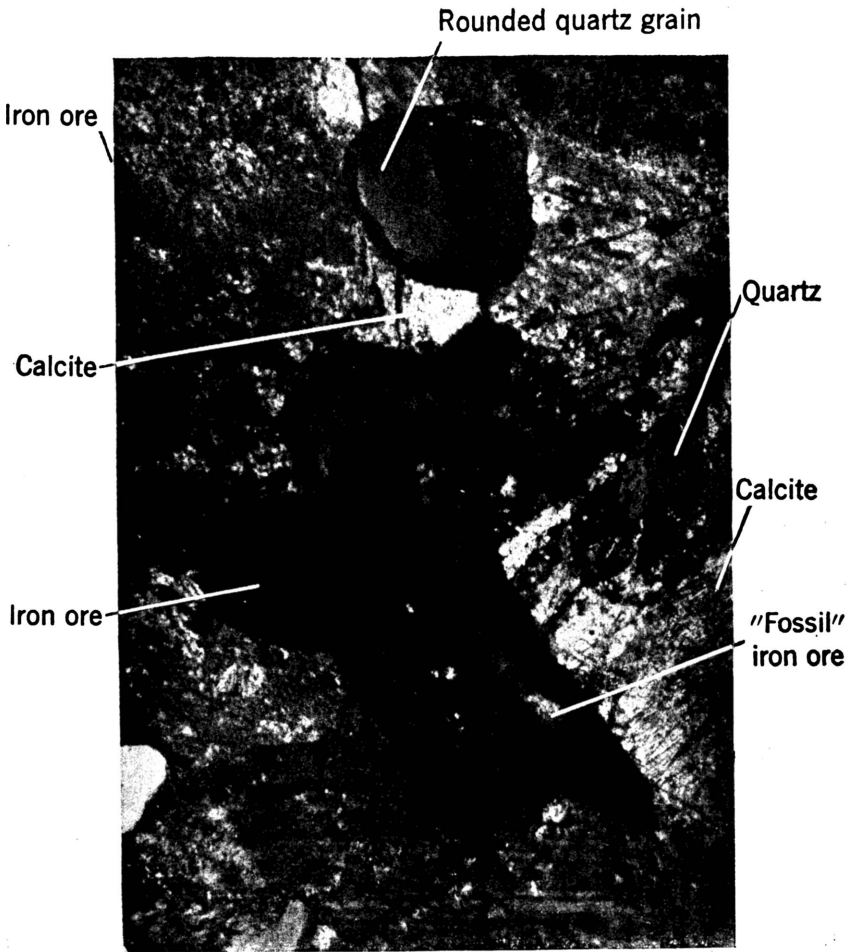
FIGURE 26.—Photomicrograph of thin section prepared from drill core, showing relations of two types of iron ore with calcite and quartz. Note fracturing of quartz. Most of the fractures have been recemented by iron ore. 3/4 inch = 48-mesh ; X 88 ; polarized light.
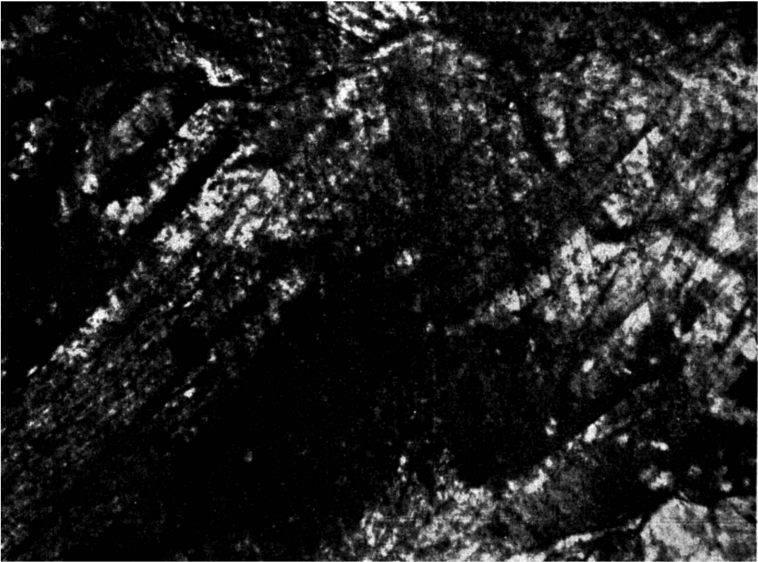
FIGURE 27.—Photomicrograph of thin section prepared from drill core, showing particle essentially calcite. Use of polarized light brings out characteristic twinning of calcite group of minerals. Calcite indicated either a stringer of interbedded limestone or a more extensive limestone formation. 13/16 inch = 48-mesh; X 92; polarized light.