The major physical problems involved and the present status of the methods of winning metals as applied to iron ores are shown in Fig. 1. The impurities that affect the value of the iron formations are silica, sulphur, and phosphorus. Moisture, volatile matter, and adverse physical structure can be removed by known processes whenever it becomes economically advisable to do so.

The applicability of the various types of metallurgical processes for the removal of the impurities is indicated in the chart under three headings. The commercial processes are in use under special conditions as to kind and location of the ores. The possible application is limited as yet to certain types of ores and to locations having special economic conditions favorable to low costs or high value of ore. Those marked impossible cannot be considered under the present state of the art of iron-ore benefication because of inherent physical or chemical difficulties. In some cases the application of a process is limited to certain minerals and to their association in the ore; such types of processes are marked as dependent on the ore.

In this tabulation the relative costs of the different processes are not shown. Under present conditions, as far as the United States is concerned, electro-metallurgical processes are too expensive for the general production of iron. Hydrometallurgy is also of very limited application because of this factor.

All these possible methods of winning iron from low-grade ores are being investigated by various agencies and individuals throughout the world. Local factors and the nature of the ore formations in most cases determine the process selected for investigation.

The work, at the Michigan College of Mining and Technology has been largely confined to gravity concentration and flotations, because these two types of processes are the cheapest in operation and also have the widest application in concentration of other ores. Since any product from a concentrating process will be in direct competition with direct shipping ores for some time to come the research into possible beneficiation processes has been limited to those that are low in cost or to those that are adapted to materials that can be secured at reduced price.

In order to get an idea of the various structural and other features of interest from the ore-dressing standpoint, a systematic study of the Gogebic Range was undertaken about four years ago after a preliminary survey of the iron-bearing formations had been made. Dr. T. M. Broderick with the cooperation of the mining officials secured samples from various available locations.* These represent several cross sections of the Ironwood formation and include a number of others taken from members that presented favorable possibilities of concentration.

From the laboratory tests on these samples much data have been secured. This data will be of value even if new processes of ore dressing are developed or changing economic conditions alter their practical application. In this report will be considered the data applicable to gravity concentration of the various members of the Ironwood formation. Originally these members may have been uniform for long stretches along the strike, but now there is quite a variation due to alteration by oxidation and leaching. Although some samples from a member showed that concentration was feasible, other samples from the same member at other locations showed lower iron content and also much less liberation with the same degree of crushing. There were also large variation in the members in going upward from the footwall. Hence the results as here given cannot be interpreted as applying to the complete members throughout the range.

*Paper by T. M. Broderick on Sampling of Ironwood Iron Formation.

Statement of the Problem Involved

The chief impurity in the Ironwood formation is silica. Any process of beneficiation applicable to the concentration of the iron-bearing members will involve the removal of the silicious or cherty portions from the iron-rich particles. The laboratory work involved the determination of the degree of liberation that could be secured by crushing to various practical sizes and the efficiency of the several methods of separating the particles by means of their difference in specific gravity. Because of the large number of samples that will ultimately have to be examined, efforts were made to find a test that would give the required information without the necessity of making a large number of routine tests. This involved the study of the various tests as to efficiency and also as to the degree to which the results could be duplicated in actual practice.

To facilitate the calculation and comparison of the results a system for this purpose was adapted from that used by the German Institute of Mining and Metallurgy [7, 9]. An explanation of this method with some representative examples has been published [12]. In Table I is given a summary of the important equations that were used.

The tests on these samples as well as on those examined previously showed that in order to make an efficient separation modifications of the general ore-dressing methods or else new processes were necessary because of various conditions such as the small actual differences in specific gravity and structural features such as porosity of the richer portion of the ore.[1] The most feasible way of overcoming this difficulty seemed to be the substitution of a heavier medium than water in the ore-dressing machine. In the laboratory heavy liquids, such as acetylene-tetrabromide and thallous formatethallous malonate solution, can be used for the purpose, but their high cost prevents their use in commercial application. Hence other means of securing a medium of high specific gravity was necessary. An investigation of the various suspensions of solids in water was made, and for iron-ore concentration the fines from the ore itself were deemed to be the most suitable.

SUMMARY OF FORMULAE.

Symbols:

a = Assay of Feed.
c = Assay of Concentrate.
b = Assay of Tailing
r = Assay of Pure Ore Mineral.

a_r = Percent of Ore Mineral in Feed.
c_r = Percent of Ore Mineral in Concentrate.
b_r = Percent of Ore Mineral in Tailings.
v = Percent Weight Recovery as Concentrate (Actual).
v_{opt} = Ideal Weight Recovery or Percent of Pure Mineral in Feed.
n = Concentration Ratio.
m = Percent Metal Recovery.
w = Percent Total Gangue Present in the Concentrate.
E = Percent Efficiency of the Separation.
q_a = Weight of Feed.
q_c = Weight of Concentrate.
q_b = Weight of Tailing.

Formulae:

$$v = \frac{q_c}{q_a} \cdot 100 = \frac{a-b}{c-b} \cdot 100.$$

$$n = \frac{q_a}{q_c} = \frac{c-b}{a-b} = \frac{100}{v}.$$

$$m = \frac{cq_c}{aq_a} \cdot 100 = \frac{c(a-b)}{a(c-b)} \cdot 100 = \frac{cv}{a}.$$

$$w = \frac{(a-b)(r-c)}{(c-b)(r-a)} \cdot 100 = \frac{v(r-c)}{r-a} = \frac{(100-c_r)v}{100-a_r}$$

$$E = m - w = m - \frac{v(r-c)}{r-a}.$$

$$E = \frac{m-v}{100-v_{opt}} \cdot 100.$$

$$E = \frac{(a-b)(c-a)r}{(c-b)(r-a)a} \cdot 100.$$

Table 1

1 This problem has been treated in detail in a previous publication (2,11).

History of High Density Media Processes

The sink-and-float method of laboratory examination of coal was introduced by Berard about 80 years ago, and in 1858 Sir Henry Bessemer proposed to use this method in cleaning coal. As a heavy medium he advocated the use of solutions of the various chlorides, specifying calcium chloride as the most suitable. The separation was to take place in a tank having a skimmer on top to remove the floating coal, while a worm-screw or a set of buckets on a chain removed the sinks from the bottom. The solution was to be drained from the products which were afterwards washed by water. The salts from the wash water were recovered by evaporation.

A similar process was tested extensively in Germany in a full-sized plant but was finally abandoned. Brown tried the same process in America in 1884. At the present time Lessing is investigating a similar process in England.

Nagelvoort has patented a process in which acetylene tetrabromide is used as the medium. In order that this process can become of commercial value a very drastic reduction in the cost of this chemical will be necessary.

All these processes in which chemicals are used as media have the disadvantage of high cost of the original media and also of the cost of its recovery from the washing liquid by evaporation or distillation.

Suspensions of solids in water have been investigated and also used to some extent in ore-dressing at various times. In 1917 T. M. Chance patented the first practical process in which suspensions of sea sand were used in the cleaning of coal [3]. In his method the same, which is between 40 and 100 mesh, is held in suspension by rising currents of water aided by mechanical agitation.

In the Conklin process, an account of which was published a little later, finely ground magnetite was to be used as the solid in the suspension. [1] The separation was to be a sink-and-float process carried on in much the same manner as originally proposed by Sir Henry Bessemer.

Wuensch proposed the use of galena as the suspensoid in a conical separator with an adjustable sorting-column-discharge as part of the machine. The actual separation was to be by a static sink-and-float process.

In Europe at present experiments are being conducted with the DeVooy's process—also a static sink-and-float separation in which a suspension of barite mixed with clay is used for the cleaning of coal [5]. The costs are reported to be low and the separation very efficient.

IRON ORE BENEFICIATION

IMPURITY	GRAVITY CONCEN-TRATION	FLOTATION	MAGNETIC SEPARATION	HYDRO-MET.	PYRO-MET. SINTERING	ELECTRO MET.	DIRECT REDUCTION
SILICA	C P	P	C P	P	I	P	P
SULPHUR	D	P	D	P	C P	P	P-D
PHOS-PHORUS	D	P	D	P TO I	I	P	I
MOISTURE	D				C	P	P
VOLATILE MATTER CO.	D	D	D	D	C	P	P
PHYSICAL STRUCTURE					C	P	P

C - COMMERCIAL PROCESSES I - IMPOSSIBLE
P - POSSIBLE APPLICATION D - DEPENDS ON ORE

Fig. 1—Chart of Iron Ore Beneficiation Methods

In January, 1932, a laboratory model of the Wuensch machine was tested in this laboratory. After several trials on iron ore it was decided that the difficulties in handling the suspension of galena and in washing it from the products outweighed the advantages of state separation over a method in which iron-ore suspension of lower specific gravity was used, much in the same manner as water Is used in ore dressing.

All the subsequent work has been on this type of separation. Even if the difficulties of washing' and handling the suspensions in a static sink-and-float separation could be overcome, there is still the fact that no suspension as yet proposed has the necessary specific gravity for all types of iron-ore separation. A specific gravity of over 4, which is required to make a

suitable concentrate from some types of iron ores, cannot be obtained with the common minerals when mixed with the amount of water necessary to make a workable suspension. Where only a certain amount of silicious material must be removed from the ore to make it commercial, suspensions of galena or special iron alloys can be used.

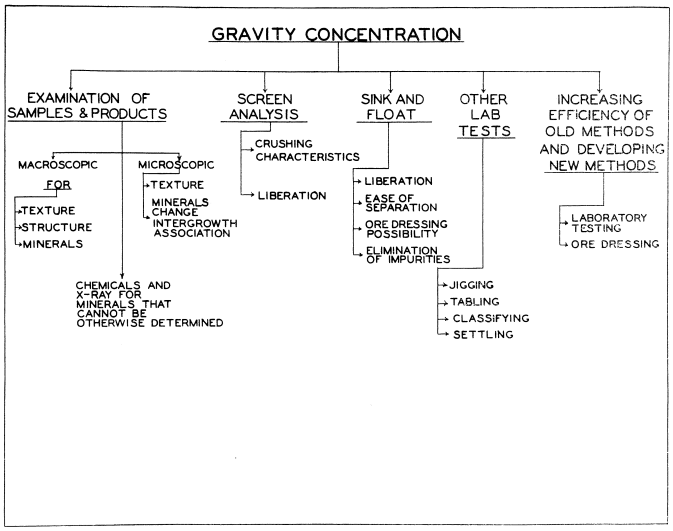

Fig. 2—Tests for Gravity Concentration

THE TESTS

The general outline of laboratory investigation is shown in Fig. 2. Distinct tests for each subdivision were not attempted as more than one of these points was involved in every test. The efficiency with which the tests were conducted increased as the work proceeded.

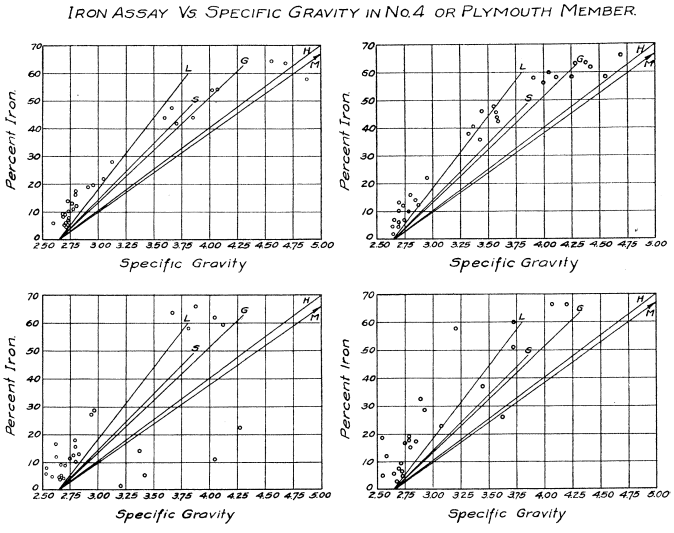

Fig. 3—Iron Assay vs. Specific Gravity in No. 4 or Plymouth Member

There is a general distinctive structure characteristic of each of the members of the Ironwood formation as shown by the samples, although there are gradations in character even in the individual members.

As a general rule the Plymouth showed bands of iron rich enough to be ore with distinct band of silica assaying under 10 per cent iron. In the Anvil member the richer bands, were much thinner and the silicious band graded in richness from 10 to over 40 per cent in iron. In most cases very little rich concentrate could be secured by crushing through ¼-inch. The Norrie was intermediate in structure between the two. The richer bands, though wide enough for coarse concentration, were lower in assay than were those of the Plymouth. The silicious bands also varied so much in iron assay that there is a large amount of material ranging from 20 to 45 per cent in iron.

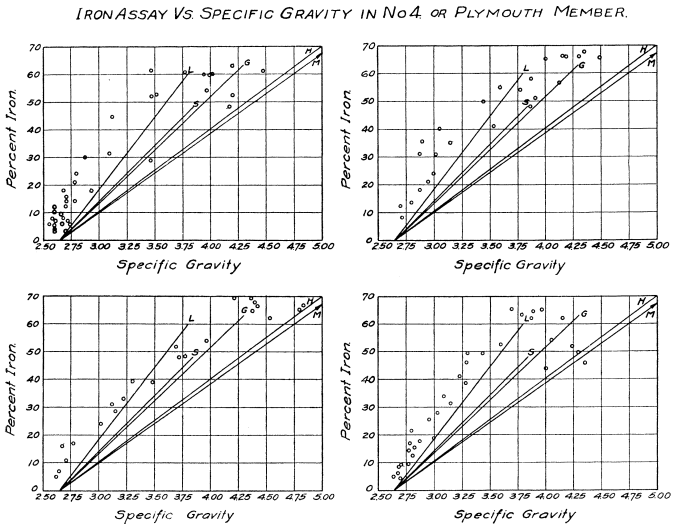

Fig. 4—Iron Assay vs. Specific Gravity in No. 4 or Plymouth Member

In each of the members there is a different relationship between the specific gravity of the particles and their iron content. This is shown clearly in Fig. 3, 4, 5, 6 and 7. Without regard to the actual abundance of each kind, particles about ½-inch in size were selected from the samples to represent the different combinations of iron ore and gangue that were present at this size. The specific gravity and the iron assay determination of each particle were made and plotted.

Lines for the various iron minerals are drawn in each figure. These lines indicate the relationship to be expected if the sample were composed of silica and the various common iron-ore minerals. The limonite line is marked L, siderite line S, Geothite line G, hematite line H, and magnetite line M. In most of the calculations the iron minerals were assumed to be hematite, although some of the samples by actual tests showed a fairly large percentage of hydrated iron minerals. In most cases the deviation from the straight line relationship in the figures is due to porosity and other textural factors.

Samples from various locations in the Plymouth member (Fig. 3 and 4) contained iron-rich bands which varied in porosity and thus had quite a range in specific gravity for particles rich enough to be in the concentrate. In some cases the silicious part was also rather porous. The

third graph in Fig. 3 shows some particles low in iron but with a high specific gravity due to manganese minerals.

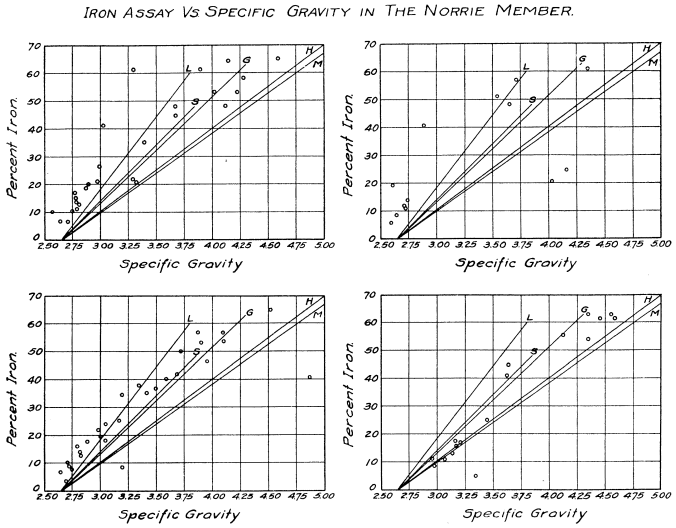

Fig. 5—Iron Assay vs. Specific Gravity in the Norrie Member

Of the samples from the Norrie member (Fig. 5) two contained considerable amounts of geothite, and the other two were rather porous. In the Anvil member samples (Fig. 6) the iron assay rises much faster than the specific gravity, especially in the range from 10 to 40 per cent. This makes it difficult to get low tailings in samples of this ore. The amount of material over 50 per cent iron in some of these samples was very limited.

The samples taken from the various slate members (Fig. 7) were finely banded, and at ½-inch the various particles showed very little differentiation in iron assay.

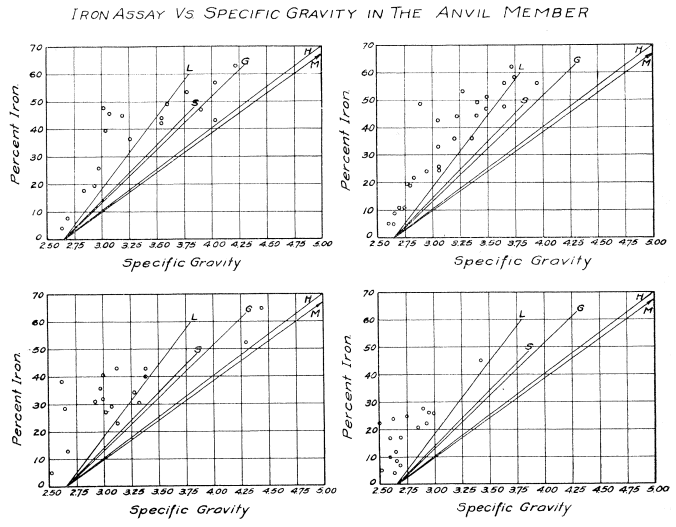

Fig. 6—Iron Assay vs. Specific Gravity in the Anvil Member

The screen analysis shows the relative resistance to crushing of the ore and gangue minerals and also the approximate amounts of each sizes which will have to be treated if the laboratory crushing is done under conditions similar to those in the actual plant. In the iron ores there has been more or less oxidation and leaching of the original formation. The screen analysis with a sink-and-float test on each fraction will usually indicate how far this process has taken place.

Unless the original formation is structurally segregated into relatively large units of rich ore and gangue, the concentration by gravity methods is not feasible. If, however, leaching or oxidation has enriched certain portions and rendered them more easy to crush, then screen analysis and the sink-and-float test will reveal the possibility.

In most of the iron-bearing formations of Michigan the ore portions have been enriched by these agencies. At a result, the screen analysis shows that the fines are richer in iron than the coarser sizes. This test in conjunction with the macroscopic examination of the sample is sufficient to indicate the size at which the more precise testing must begin.

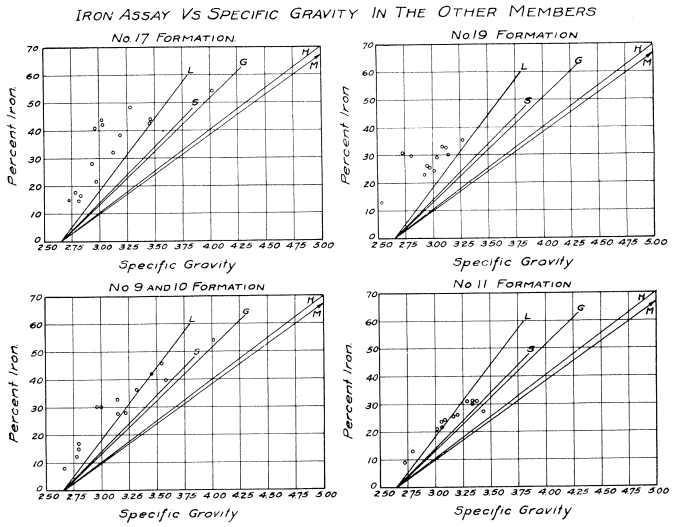

Fig. 7—Iron Assay vs. Specific Gravity in the Other Members

Table II gives a screen analysis typical of the Plymouth member. The very coarse sizes are rather low in iron, while the fines are much richer. Those portions of this member that were not oxidized and leached showed less variation and also gave a much poorer liberation and separation in the other tests.

Table III gives a typical screen analysis of a Norrie member. For the altered portions the screen analyses are somewhat similar to those from the Plymouth except that the richer portions are of somewhat lower grade while the cherty portions are somewhat higher in iron assay.

Table IV shows a screen analysis that is typical of the Anvil member. In this the poorer bands are decidedly richer in iron than is the case for the Plymouth, while the iron rich bands are lower in grade.

Table V shows a typical screen analysis of the Slate members. These are structurally too fine to show much liberation when crushed through ¼-inch.

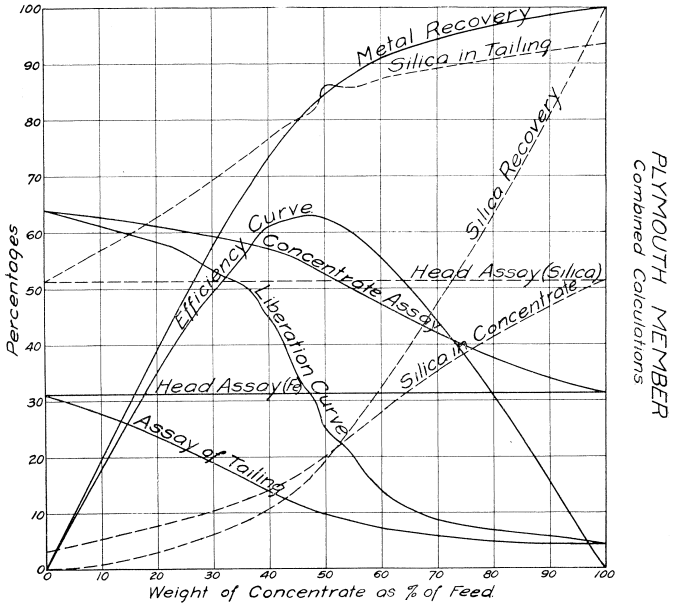

Fig. 8—Combined Curves for Plymouth Member

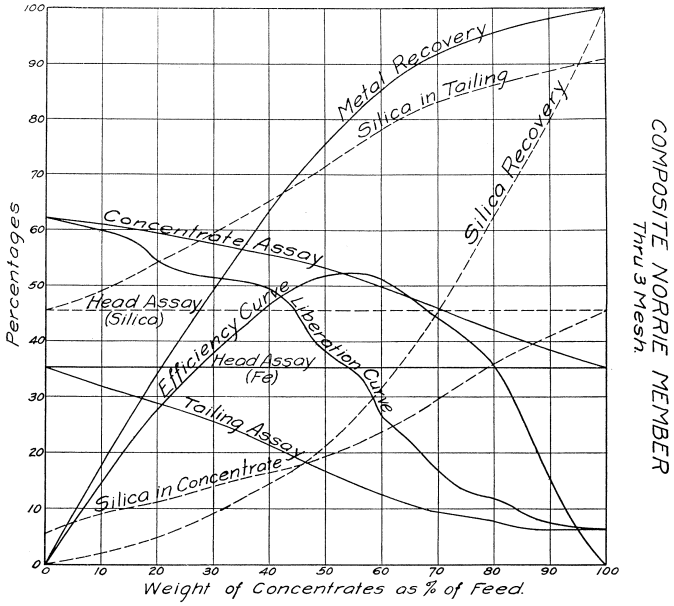

Fig. 9—Composite of the Samples from the Norrie Member

The success of any ore-dressing operation depends on the degress of liberation of the constituent minerals made by the crushing operations. This liberation is measured by the amount of purity of the various groups into which the crushed sample can be separated. Of all the laboratory methods of making the separation on the basis of constituent minerals, the sink-and-float method is the most accurate, provided there is a difference in specific gravity between the minerals to be separated.

In these tests acetylene-tetrabromide was used in the screen analysis tests, while thallous-formate-thallous-malonate solution was used for obtaining liberation curves and other data for the graphs. With the latter solution it is possible to fractionate a sample into six or more parts, each of which differs from the others by any desired difference in specific gravity. Porosity in some of the minerals affects the results; but that factor affects the ore-dressing operations in the same manner, and so the value of the test is not decreased thereby. The method of making these tests has been fully described in previous publications. [12, 13]

All of the samples were tested by this method when crushed through 3 mesh or ¼-inch for the purpose of comparing the different formations. Composites of the three major members were calculated from these tests. As has been mentioned before, there is a variation in the members going from the footwall upward and also away from the commercial ore-bodies. Hence the results as given cannot be interpreted to apply absolutely to these members as a whole.

The calculations for the composite of the Plymouth members are given in Tables VI and VII. The results are also shown graphically in Fig. 8.

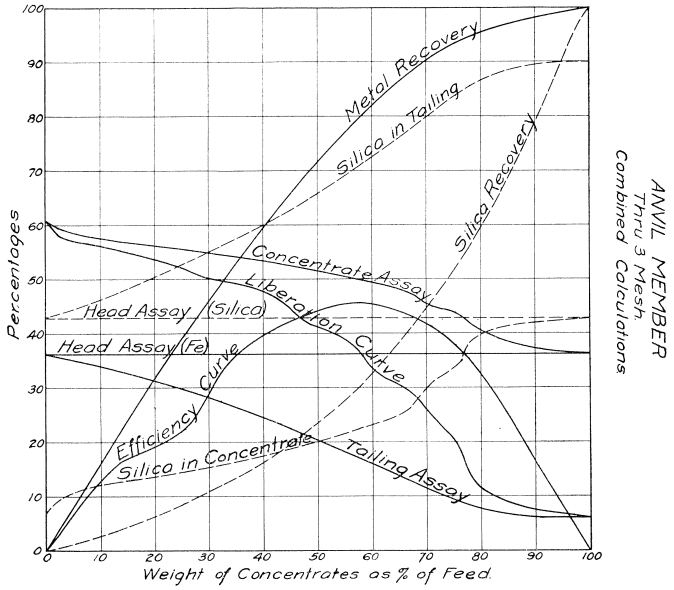

Fig. 10—Composite of the Samples from the Anvil Member

Table VIII shows the variation in a complete cross section of the Plymouth member. In the first part of the table are given the data for the iron and silica assay of the concentrate and tailing, the metal recovery, and efficiency for a concentration ratio of about 3 to 1 or 35 per cent weight recovery as concentrate. In the second part are given the same data for the point of maximum efficiency as defined by the point at which the poorest particle in the concentrate has the same assay as the heads. In this case the grade of concentrate varies with the head analysis and also with the degree of liberation. In the third part the data refers to a concentrate having 10 per cent silica. The top 50 feet of the member was omitted in this tabulation, because the amount of concentrate of this grade to be obtained from it was negligible.

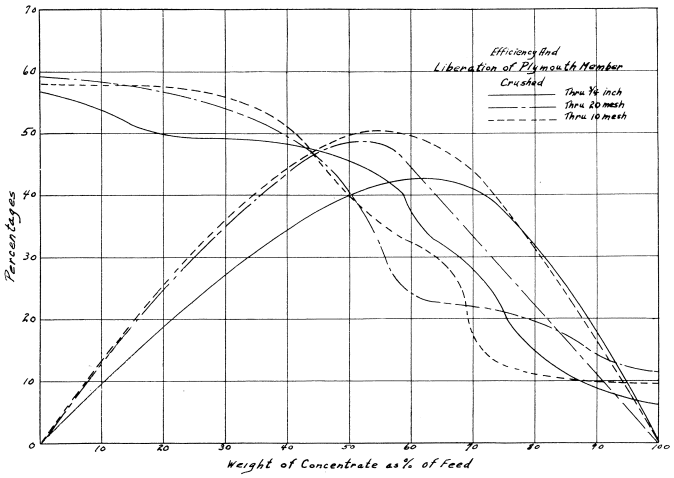

Fig. 11—Liberation and Efficiency of Separation for the Plymouth Member at Various Sizes

The composite of the Norrie member samples is given in Tables IX and X and also graphically in Fig. 9. In this member there is also a large variation in structure, analysis, and degree of liberation that can be achieved on crushing at the various points sampled. In Table XI are given the data for a complete section of this member. The top here also is too silicious to make an acceptable concentrate.

The composite of the Anvil member samples is given in Tables XII and XIII and also graphically in Fig 10. As can be seen from the tables and the graph, this member as a whole has more silica in the richer bands than the others. In Table XIV are given the data for a complete cross-section of this member. Here also the top is more silicious than the lower portion.

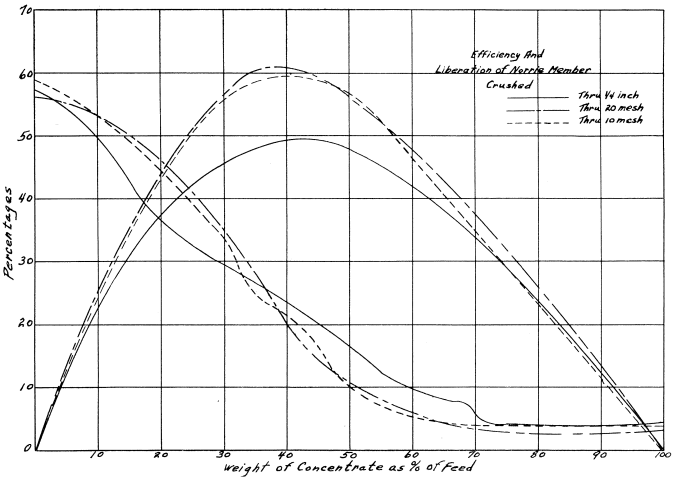

Fig. 12—Liberation and Efficiency of Separation for the Norrie Member at Various Sizes

The composites of the three members are compared in Table XV. In the first section the concentrates and tailings are compared when 35 per cent of the sample is recovered as concentrate. The silica in the concentrate rises from 12 per cent for the Plymouth to 16 and 17 per cent for the Norrie and Anvil respectively. At the same time the iron assay of the tailings goes up from 16.5 to 24 and 27 per cent respectively.

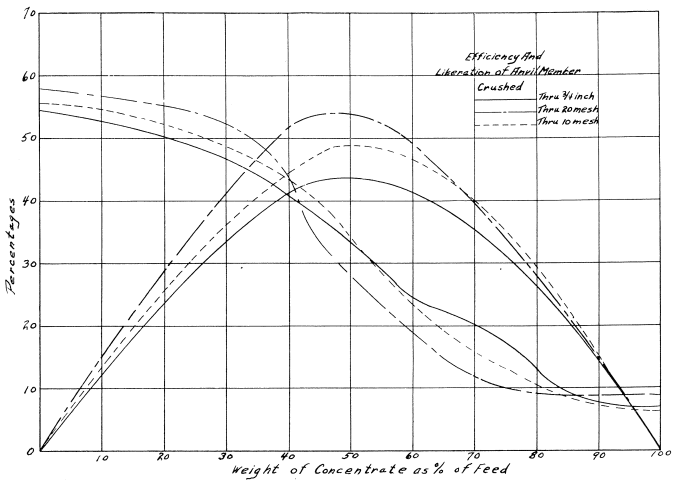

Fig. 13—Liberation and Efficiency of Separation for the Anvil Member at Various Sizes

In the second section the data are given for the point of maximum efficiency, which is 62.5 per cent for the Plymouth and 52.5 and 45.5 for the Norrie and Anvil respectively. The grade of concentrate decreases in, the same manner although the head assays of the composites are 31.18, 35.13 and 36.20 per cent iron for the Plymouth, Norrie and Anvil respectively. In the last section the data are given for a 10 per cent silica concentrate. A weight recovery of 29 per cent is indicated for the Plymouth, while for the Norrie and Anvil it is 13 and 4 per cent. The metal recovery drops from 48.5 to 22 and 5 per cent respectively. The slate members were not included in this comparison, as there is no possibility of their concentration at sizes above 1/32 of an inch, the size which was the lower limit for most of the tests treated in this report.

In every ore-dressing investigation it is necessary to determine the economic limit to which crushing and grinding can be carried. Liberation is usually increased by finer grinding; but, after a certain limit is reached, the separation becomes more difficult and inefficient. The handling and also reduction processes are also adversely affected. Hence a balance between the increased metal recovery and the increased costs must be made.

An inspection of some of the samples of iron formations shows a fairly coarse banded structure. In some cases these samples cleave at the band boundaries so that fairly coarse crushing will affect a marked liberation of the various bands. On the other hand these same samples will show very little increase in liberation on fine grinding, as the texture of the bands themselves is very fine. [(2)] In finely banded material and in cases where the bands vary in thickness increased liberation is shown by fine crushing.

Crushing tests were made on several of the samples, especially on those that did not yield very good results at the coarser sizes. The samples for this test were cut into five portions which were then crushed through ¾, ¼, 1/16, 1/32 and 1/64 of an inch square opening. These portions were then fractionated by means of thallous-

formate-thallous-malonate solution, and the results calculated and plotted in the usual manner.

In Fig. 11 are plotted the liberation and efficiency curves for a rather difficult sample from the Plymouth member. In order to avoid confusion in the graph only three of the five tests are shown. In this sample the fines were very much richer in iron than were the coarser particles; further, since these high-grade fines made a rather stable suspension, the separation of 1/32 of an inch was actually less efficient than at 1/16 of an inch.

In Fig. 12 are plotted the same curves for a sample from the Norrie. This sample was rather finely banded and had a relatively low head assay. It shows some increase in liberation on fine grinding, but other tests showed that after a separation at fairly coarse sizes had been made, the middling products did not yield a suitable concentrate product on further grinding.

In Fig. 13 are shown the same curves for a sample from the Anvil member. Here there was a decided increase in liberation on further crushing because the richer bands were rather thin. But even at 1/32 of an inch no great amount of rich concentrate could be obtained from the sample. The almost uniform slope of the liberation curve shows that still finer grinding is necessary for concentration of this sample.

In gravity concentration jigs are generally used for separating the coarser sizes. There are a large number of different types of these machines, and each kind must be adapted to the ore being separated. The variables under the control of the operators of these machines are speed, length and kind of stroke, rate and size of feed, the amount and location of the inlet for addition of medium and also the kind of medium used for the separation.

In the work on jigging, parallel tests were made using as media water, iron ore suspensions and galena suspensions. In other cases a sink-and-float test for actual liberation was made instead of the galena suspension test.

Some of the results have been published before (11, 12, 13, 14). These show conclusively that the heavier media have a decided advantage over water in separating ores in which the difference in specific gravity is slight.

In Table XVI are given the results of comparative jig tests on the various sizes from a sample of the Plymouth member. No separation could be made on the plus 1-inch size by using water as a medium within the range of speed and stroke that could be obtained in the 17" x 24" Woodbury jig used in these tests. The efficiencies given are for the amounts of concentrate and tailing produced and are not directly comparable except where the weight by per cent of these products are approximately the same. However, the efficiencies for the suspensions are much higher than is the case for water, and the difference increases with the difficulty of separation.

In Table XVII are given the results of these tests on a sample of the Norrie. In the sizes above ½-inch no acceptable concentrate could be made, although a considerable amount of tailings could be discarded. The liberation and hence also the absolute efficiency of separation in this sample were much lower than in the sample of the Plymouth cited above.

In Table XVIII are given the results of the tests on a sam-of the Anvil. In this case the tailings at all sizes were much richer than in the preceding samples, while the concentrates were of a lower grade. In other words there was less difference in specific gravity between the products, and the amount of middlings was large. The absolute efficiencies also show much larger variation than in the previous cases. The separation by water was only about one-half as good as that by heavy solutions and suspensions.

There are several practical problems connected with the use of suspensions that had to be solved. For this purpose there was set up a complete flowsheet of the process, consisting of a jig, washing trommels, tanks, thickener and the necessary pumps [(14)]. A large number of tests were made with continuous operation for periods up to 8 hours each. These tests showed that there would be no difficulties inherent in the use of iron ore suspensions. The washing of the products could be carried to any desired degree of completeness, and the suspensions could be easily recovered by thickening. The thickener overflow could be recirculated as wash water; so that very little additional water was necessary. The products could be kept at any desired analysis if the feed was uniform in assay and amount. No trouble was experienced in starting up again after a shut-down of a day or two. The only precaution that was taken was to flush out the suspension from the pumps on stopping.

Most of the work as mentioned before was done on the sizes above 1/16-inch. Tabling tests were also made on the minus 1/16-inch. These tests showed that though the separation on a table is not as efficient as in a jig, still concentration was possible provided there had been sufficient liberation of the minerals on crushing to this size. Where the liberation was good, as shown by the sink-and-float test, the tabling results were excellent; but where a large amount of midding was indicated by the graphs the results were poorer.

The work on flotation is still in the stage of finding the fundamental principles. Although the results are encouraging, still it is advisable to confine the work to a few standard samples.

SUMMARY

The various subdivisions of the Ironwood formation differ from one another and also within themselves in those properties that are of interest to the ore dresser. For gravity concentration to be possible, the ore must consist of separable structural units that are large enough for efficient separation by this method. The grade of concentrate obtainable from a given ore is governed by the richness of the structural units containing the valuable minerals.

The method of fractionating a sample by means of a solution of thallous formate-thallous malonate and graphically illustrating the results show how far the ore minerals can be separated from the gangue at any degree of crushing. From the curves can be determined the amount of any given grade of concentrate and the amount of discard that can be obtained from any ore. It is also possible directly to compare different samples as to amenability to ore dressing operations.

The difference in specific gravity between the chert and the iron-oxide minerals is not sufficiently great for best gravity concentration, especially where large amounts of middlings are involved. In order to improve the separation a method was developed in which suspension of finely ground iron ore is used instead of water as a medium in a jig.

The Plymouth member showed the largest number of samples that offered possibilities of concentration, the Norrie and Anvil ranking next. The slate members cannot be considered as amenable to gravity concentration. But even in the individual members there are portions that are either too low in original assay to be worth mining or else are too fine or too uniformly low-grade structurally to be concentrated.

The criteria for obtaining an acceptable concentrate lies in the presence in the sample of bands or other coarse aggregates rich enough in iron to make a commercial product and in such amounts as to make it a profitable venture.

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A Typical Screen Analysis of a Sample from the Plymouth Member

Size Mesh—	Samples			Sinks			Floats		
	Weight Per Cent	Fe Per Cent	Total Fe Per Cent	Weight Per Cent	Fe Per Cent	Total Fe Per Cent	Weight Per Cent	Fe Per Cent	Total Fe Per Cent
-2″ + 1″	14.21	18.65	8.26	5.65	26.45	4.66	8.56	13.50	3.60
-1″ + 2 Mesh	32.21	23.30	23.39	9.72	50.60	15.33	22.49	11.50	8.06
-2 + 3	14.70	35.65	16.33	8.30	53.20	12.76	6.40	12.90	2.57
-3 + 4	6.70	38.90	8.12	4.39	53.85	7.37	2.31	10.55	0.75
-4 + 6	8.71	41.20	11.19	6.15	54.30	10.41	2.56	9.80	0.78
-6 + 10	7.86	43.10	10.56	5.86	54.55	9.97	2.00	9.40	0.59
-10 + 14	3.34	44.60	4.64	2.53	55.50	4.37	0.81	10.70	0.27
-14 + 20	2.05	45.00	2.88	1.56	55.60	2.70	0.49	11.65	0.18
-20 + 28	2.17	44.55	3.01	1.61	56.05	2.82	0.56	11.10	0.19
-28 + 35	4.56	46.00	6.54	4.16	49.40	6.41	0.40	10.80	0.13
-35 + 65	2.21	46.40	3.20	2.11	47.90	3.15	0.10	14.85	0.05
-65 + 100	0.18	46.55	0.26	0.17	47.40	0.26	0.01	11.20	0.00
-100 + 200	0.46	47.20	0.68	0.46	47.55	0.68	0.00	7.00	0.00
-200	0.64	47.20	0.94	0.62	47.35	0.91	0.02	43.50	0.03
	100.00	32.08	100.00	53.29	49.85	82.80	46.71	11.82	17.20

TABLE III

Typical Screen Analysis of a Sample from the Norrie Member

Size Mesh—	Samples			Sinks			Floats		
	Weight Per Cent	Fe Per Cent	Total Fe Per Cent	Weight Per Cent	Fe Per Cent	Total Fe Per Cent	Weight Per Cent	Fe Per Cent	Total Fe Per Cent
+ 1	19.47	24.20	12.93	9.32	36.65	9.37	10.15	12.80	3.56
-1 + 2	28.53	30.50	23.86	20.29	37.95	21.11	8.24	12.15	2.75
-2 + 3	10.35	40.57	11.51	7.77	49.50	10.54	2.58	13.70	0.97
-3 + 6	13.68	44.13	16.55	10.77	52.50	15.50	2.91	13.15	1.05
-6 + 10	11.32	46.39	14.40	9.49	53.50	13.90	1.83	10.00	0.50
-10 + 14	1.40	45.60	1.75	1.17	52.75	1.69	0.23	9.10	0.06
-14 + 20	2.68	45.30	3.33	2.17	52.00	3.09	0.51	16.90	0.24
-20 + 28	2.68	45.20	3.32	2.14	52.60	3.09	0.54	15.70	0.23
-28 + 35	2.74	45.70	3.43	2.19	53.50	3.21	0.55	14.50	0.22
-35 + 48	1.63	45.50	2.04	1.36	52.25	1.95	0.27	11.70	0.09
-48 + 100	2.61	45.60	3.26	2.15	52.20	3.08	0.46	14.80	0.18
-100 + 200	0.57	45.60	0.71	0.48	49.90	0.66	0.09	23.30	0.05
-200	2.34	45.40	2.91	1.67	45.80	2.10	0.67	44.40	0.81
	100.00	36.47	100.00	70.97	45.88	89.29	29.03	13.47	10.71

TABLE IV

A Typical Screen Analysis of a Sample from the Anvil Member

Size Mesh—	Samples			Sinks			Floats		
	Weight Per Cent	Fe Per Cent	Total Fe Per Cent	Weight Per Cent	Fe Per Cent	Total Fe Per Cent	Weight Per Cent	Fe Per Cent	Total Fe Per Cent
-2″ + 1″	21.61	29.20	18.65	12.84	37.25	14.14	8.77	17.40	4.51
-1″ + 2 Mesh	38.85	29.45	33.82	22.98	38.60	26.22	15.87	16.20	7.60
-2 + 2½	4.25	34.60	4.35	2.77	47.00	3.85	1.48	11.40	0.50
-2½ + 3	3.49	35.75	3.69	2.32	47.30	3.24	1.17	12.90	0.45
-3 + 4	8.58	37.40	9.48	5.98	48.45	8.56	2.60	11.95	0.92
-4 + 6	4.67	40.25	5.56	3.44	50.60	5.15	1.23	11.20	0.41
-6 + 8	3.49	42.80	4.42	2.65	52.50	4.11	0.84	12.15	0.31
-8 + 10	2.07	43.70	2.68	1.57	53.80	2.50	0.50	12.00	0.18
-10 + 14	3.00	44.80	3.97	2.37	53.50	3.75	0.63	11.90	0.22
-14 + 20	1.86	44.70	2.46	1.38	56.00	2.29	0.48	12.35	0.17
-20 + 28	1.59	45.40	2.13	1.25	54.15	2.00	0.34	13.15	0.13
-28 + 35	1.50	45.50	2.02	1.19	54.40	1.91	0.31	11.60	0.11
-35 + 48	1.21	46.20	1.65	0.84	56.85	1.41	0.37	22.40	0.24
-48 + 65	1.42	45.20	1.90	0.96	55.70	1.58	0.46	23.15	0.32
-65 + 100	0.42	44.60	0.56	0.28	55.40	0.46	0.14	23.70	0.10
-100 + 150	0.41	44.50	0.54	0.36	47.30	0.50	0.05	26.30	0.04
-150 + 200	0.71	45.20	0.95	0.61	45.65	0.82	0.10	42.65	0.13
-200	0.87	45.60	1.17	0.64	46.20	0.87	0.23	43.90	0.30
	100.00	33.83	100.00	64.43	43.77	83.36	35.57	15.81	16.64

TABLE V

A Typical Screen Analysis of a Sample from the Slate Members

Size Mesh—	Weight Per Cent	Fe Per Cent	Total Fe Per Cent	Weight Per Cent	Fe Per Cent	Total Fe Per Cent	Weight Per Cent	Fe Per Cent	Total Fe Per Cent
-3 + 4	11.77	23.40	11.90	6.71	28.00	8.12	5.06	17.30	3.78
-4 + 6	26.16	23.20	26.21	16.79	27.00	19.58	9.37	16.40	6.63
-6 + 10	28.78	22.80	28.34	17.99	27.00	20.98	10.79	15.80	7.36
-10 + 14	7.36	23.20	7.38	4.01	29.20	5.06	3.35	16.00	2.32
-14 + 20	5.52	22.70	5.41	2.76	29.40	3.50	2.76	16.00	1.91
-20 + 28	4.70	22.80	4.63	2.51	28.90	3.13	2.19	15.80	1.50
-28 + 35	3.73	22.80	3.67	1.99	29.10	2.50	1.74	15.60	1.17
-35 + 48	2.62	22.90	2.59	1.57	28.90	1.96	1.05	13.90	0.63
-48 + 100	3.88	23.10	3.87	1.97	30.55	2.60	1.91	15.40	1.27
-100 + 200	2.33	23.60	2.37	1.13	33.00	1.61	1.20	14.70	0.76
-200	3.15	26.70	3.63	0.93	31.70	1.27	2.22	24.60	2.36
	100.00	23.15	100.00	58.36	27.90	70.31	41.64	16.50	29.69

TABLE VI

Composite of Plymouth Member
Iron Calculations

Percent Weight	Percent Iron	Weight of Iron per 1,000 Grams of ore	Cumulative Percent Weight	Cumulative Weight of iron In concentrate	Assay of Concentrate	100-Cumulative Percent Weight	Weight of Iron in Tailings	Assay of Tailings	Metal Recovery	Metal recovery Minus cumulative Percent weight	Efficiency
7.904	62.80	49.64	7.904	49.64	62.80	92.10	262.16	28.46	15.92	8.02	14.49
10.102	60.72	60.73	18.006	110.37	61.30	81.99	201.43	24.56	35.40	17.39	31.42
9.786	57.31	56.08	27.792	166.45	59.86	72.21	145.35	20.12	53.38	25.59	46.24
8.823	52.00	45.88	36.615	212.33	57.99	63.39	99.47	15.69	68.09	31.48	56.88
2.747	47.67	13.09	39.362	225.42	57.27	60.64	86.38	14.24	72.29	33.93	61.31
3.489	43.36	15.13	42.851	240.55	56.14	57.15	71.25	12.47	77.14	34.29	61.96
2.175	36.81	8.01	45.026	248.56	55.20	54.97	63.24	11.50	79.71	34.68	62.67
2.511	32.00	8.04	47.537	256.60	53.97	52.46	55.20	10.52	82.29	34.75	62.79
2.025	29.02	5.88	49.562	262.48	52.96	50.44	49.32	9.71	84.18	34.62	62.56
6.162	22.36	13.78	55.724	276.26	49.58	44.28	35.54	8.03	88.60	32.88	59.41
5.396	15.54	8.39	61.12	284.65	46.57	38.88	27.15	6.98	91.29	30.17	54.51
8.936	10.30	9.20	70.056	293.85	41.94	29.94	17.95	5.99	94.24	24.18	43.69
23.75	6.51	15.46	93.806	309.31	32.97	6.19	2.49	4.02	99.20	5.39	9.74
6.194	4.02	2.49	100.00	311.80	31.18	0.00	0.00	0.00	100.00	0.00	0.00
		.7)311.80									
		44.68									

100.00 — 44.68 = 55.32

TABLE VII

Composite of Plymouth Member
Silica Calculations

Percent Weight	Percent Silica	Weight of silica Per 1,000 grams Of ore	Cumulative Percent Weight	Cumulative Weight of silica In concentrate	Assay of Concentrate	100-Cumulative Percent Weight	Weight of Silica in Tailings	Assay of Tailings	Silica Recovery
7.90	5.00	3.95	7.90	3.95	5.00	92.10	510.57	55.43	0.768
10.10	8.90	8.99	18.00	12.94	7.18	82.00	501.58	61.16	2.52
9.79	14.00	13.71	27.79	26.65	9.58	72.21	487.87	67.56	5.18
8.82	21.50	18.96	36.61	45.61	12.45	63.39	468.91	73.97	8.87
2.75	27.64	7.60	39.36	53.21	13.51	60.64	461.31	76.07	10.35
3.49	32.90	11.48	42.85	64.69	15.10	57.15	449.83	78.71	12.58
2.18	42.50	9.27	45.03	73.96	16.42	54.97	440.56	80.14	14.38
2.51	49.50	12.42	47.54	86.38	18.17	52.46	428.14	81.61	16.80
2.02	53.90	10.89	49.56	97.27	19.63	50.54	417.25	85.55	18.92
6.16	61.90	38.13	55.72	135.40	24.30	44.28	379.12	85.61	26.34
5.40	72.50	39.15	61.12	174.55	28.56	38.88	339.97	87.44	33.95
8.94	82.46	73.72	70.06	248.27	35.43	29.94	266.25	88.92	48.29
23.75	87.92	208.81	93.81	456.48	48.66	6.19	57.44	92.79	88.78
6.19	92.80	57.44	100.00	513.92	51.39	0.00	0.00	0.00	100.00
		513.92							

TABLE VIII

Liberation at 3 Mesh
Cross-Section of Plymouth Member. Each Sample 25 Feet
Head Analyses—25.83, 30.45, 30.01, 30.65, 30.06, 30.08, and 22.82

	Concentrate					Tailings		
	Weight Recovery	% Fe	% SiO_2	Metal Recovery	Efficiency	% Fe	% SiO_2	% Total SiO_2
1	35	54.0	19.0	73.0	60.5	10.5	84	88.0
2	35	6.05	8.5	69.5	60.5	14.5	79.5	94.5
3	35	61.5	9.0	71.5	65.0	13.0	79.5	94.5
4	35	61.0	9.0	69.5	61.5	14.0	78.0	94.5
5	35	59.5	11.5	70.0	61.5	14.0	76.0	92.5
6	35	47.5	15.5	70.0	51.5	11.5	82.0	84.5
7	35	48.4	28.0	75.0	58.5	8.5	86.5	85.0
Average	35	56.07	14.35	71.21	59.86	12.28	80.79	77.64
Maximum Efficiency								
1	34.0	55.0	17.5	72.0	60.5	11.0	83.5	90.5
2	40.0	57.0	13.5	75.5	63.0	12.5	82.5	90.0
3	38.0	60.0	10.0	76.0	67.0	11.5	81.5	93.0
4	37.5	61.0	12.0	72.0	61.5	13.5	79.0	92.0
5	41.0	56.5	15.0	77.0	63.5	11.5	79.0	87.5
6	44.0	45.0	23.0	82.0	57.5	8.0	87.0	77.0
7	40.0	45.5	32.0	80.0	59.5	7.5	88.5	80.5
Average	39.21	54.28	17.51	76.36	61.78	10.78	83.0	87.14
10% Silica in Concentrate, Omitting 6 and 7								
1	27.5	60.0	10.0	68.5	57.0	13.0	80.5	95.5
2	37.0	59.5	10.0	72.0	62.0	13.5	81.0	93.0
3	38.0	60.0	10.0	76.0	67.0	11.5	81.5	93.0
4	36.0	60.0	10.0	70.0	61.7	14.0	77.5	93.5
5	32.0	60.5	10.0	65.0	57.5	15.5	73.5	94.0
Average	34.10	60.0	10.0	70.30	61.04	13.50	78.80	93.80

TABLE IX

Composite of the Norrie Member
Iron Calculations

Percent Weight	Percent Iron	Weight of Iron per 1,000 Grams of ore	Cumulative Percent Weight	Cumulative Weight of iron In concentrate	Assay of Concentrate	100-Cumulative Percent Weight	Weight of Iron in Tailings	Assay of Tailings	Metal Recovery	Metal recovery Minus cumulative Percent weight	Efficiency
11.01	60.76	66.90	11.01	66.90	60.76	89.99	284.42	31.61	19.04	8.03	16.12
8.20	58.08	47.62	19.21	114.52	59.61	80.79	236.80	29.31	32.59	13.38	26.85
16.87	51.91	87.57	36.08	202.09	56.01	63.92	149.23	23.35	57.51	21.43	43.01
8.08	49.35	39.87	44.16	241.96	54.79	55.84	109.36	19.58	68.86	24.70	49.57
4.22	42.66	18.00	48.38	259.96	53.73	51.62	91.36	17.70	73.98	25.60	51.38
8.54	36.69	31.33	56.92	291.29	51.17	43.08	60.03	13.93	82.90	25.98	52.15
3.33	29.50	9.82	60.25	301.11	49.97	39.75	50.21	12.63	85.70	25.45	51.07
8.36	22.74	19.01	68.61	320.12	46.66	31.39	31.20	9.94	91.11	22.50	45.16
11.04	13.89	15.33	79.65	335.45	42.12	20.35	15.87	7.80	95.47	15.82	31.75
7.67	10.27	7.88	87.32	343.33	39.31	12.68	7.99	6.30	97.71	10.39	20.85
12.68	6.30	7.99	100.00	351.22	35.13	0.00	0.00	0.00	100.00	0.00	0.00
		7)351.32									
		50.19									

100.00 — 50.19 = 49.81

TABLE X

Composite of the Norrie Member
Silica Calculations

Percent Weight	Percent Silica	Weight of silica per 1,000 grams Of ore	Cumulative Percent Weight	Cumulative Weight of silica In concentrate	Assay of Silica	100-Cumulative Percent Weight	Weight of Silica in Tailings	Assay of Tailings	Silica Recovery
11.01	9.20	10.13	11.01	10.13	9.20	89.99	444.94	49.44	2.22
8.20	13.50	11.07	19.21	21.20	11.03	80.79	433.87	53.70	4.66
16.87	21.00	35.43	36.08	56.63	15.70	63.92	398.44	62.33	12.44
8.08	24.20	19.55	44.16	76.18	17.25	55.84	378.89	67.85	16.73
4.22	34.00	14.35	48.38	90.53	18.71	51.62	364.54	70.61	19.89
8.54	41.70	35.61	56.92	126.14	22.16	43.08	328.93	76.35	27.71
3.33	52.50	17.48	60.25	143.62	23.84	39.75	311.45	78.35	31.55
8.36	62.14	51.95	68.61	195.57	28.50	31.39	259.50	82.66	42.97
11.04	77.50	85.56	79.65	281.13	35.30	20.35	173.94	85.47	61.76
7.67	81.30	62.36	87.32	343.49	39.42	12.68	111.58	88.00	75.46
12.68	88.00	111.58	100.00	455.07	45.51	0.00	0.00	0.00	100.00
		455.07							

TABLE XI

Liberation at 3 Mesh in a Cross-Section of Norrie Member
Heads—36.01, 30.86, 35.21, 37.18, and 34.29

	Concentrate				Tailings			
	Weight Recovery	% Fe	% SiO_2	Metal Recovery	Efficiency	% Fe	% SiO_2	% Total SiO_2
1	35.0	60.0	10.0	58.5	48.5	23.0	64.0	91.5
2	35.0	56.5	12.0	61.0	46.0	15.0	76.0	90.5
3	35.0	60.0	9.0	60.0	48.5	21.0	63.5	92.5
4	35.0	57.5	12.5	52.5	38.0	26.0	56.0	88.5
5	35.0	54.5	17.0	55.0	39.5	23.5	63.5	87.0
Average	35.0	57.0	12.10	57.40	44.10	21.70	64.60	90.0
Maximum Efficiency								
1	54.5	55.0	12.5	88.0	60.0	12.5	78.0	80.0
2	49.0	53.5	20.0	89.0	64.5	9.0	85.5	81.5
3	50.0	56.5	14.5	80.0	60.5	14.0	75.0	83.0
4	55.0	54.0	17.5	80.0	53.5	16.5	69.0	75.5
5	52.5	50.0	22.5	77.5	48.0	16.5	74.0	75.5
Average	52.20	53.80	17.40	82.90	57.30	13.70	76.30	79.10
Concentrate With 10% Silica, Omitting 5								
1	35.0	60.0	10.0	58.5	48.5	23.0	64.0	91.5
2	30.0	57.0	10.0	52.0	40.0	17.5	73.0	92.5
3	38.0	59.5	10.0	64.0	53.0	20.0	66.0	91.5
4	26.0	58.5	10.0	38.0	28.0	29.0	51.5	92.0
Average	32.25	58.75	10.0	53.12	42.37	22.37	63.62	91.87

TABLE XII

Composite of Samples from Anvil Member
Iron Calculations

Percent Weight	Percent Iron	Weight of Iron per 1,000 Grams of ore	Cumulative Percent Weight	Cumulative Weight of Iron In concentrate	Assay of Concentrate	100-Cumulative Percent weight	Weight of Iron in Tailings	Assay of Tailings	Metal Recovery	Metal recovery Minus cumulative Percent weight	Efficiency
2.40	58.44	14.03	2.40	14.03	58.44	97.60	347.96	35.65	3.88	1.48	3.07
11.25	56.60	63.68	13.65	77.71	56.73	86.35	284.28	32.92	21.46	7.81	16.17
12.05	53.86	64.90	25.70	142.61	55.49	74.30	219.38	29.52	39.39	13.69	22.35
6.46	50.32	32.51	32.16	175.12	54.45	67.84	186.87	27.54	48.37	16.21	33.57
10.90	48.70	53.08	43.06	228.20	52.99	56.94	133.79	23.50	63.03	19.97	41.36
5.56	43.80	24.35	48.62	252.55	51.94	51.38	109.44	21.30	69.75	21.13	43.76
9.83	39.56	38.89	58.45	291.44	49.86	41.55	70.55	16.98	80.49	22.04	45.64
8.46	31.98	27.05	66.91	318.49	47.60	33.09	53.50	13.14	87.96	21.05	43.59
4.18	27.25	11.39	71.09	329.88	46.40	28.91	32.11	11.10	91.11	20.02	41.46
4.38	22.05	9.66	75.47	339.54	44.99	24.53	22.45	9.15	93.78	18.31	37.92
2.28	17.30	3.94	77.75	343.48	44.18	22.25	18.51	8.32	94.87	17.12	35.46
5.63	11.08	6.24	83.38	349.72	41.94	16.62	12.27	7.38	96.59	13.21	27.36
7.56	8.57	6.48	94.94	356.20	39.17	9.06	5.79	6.39	98.38	7.44	15.41
9.06	6.39	5.79	100.00	361.99	36.20	0.00	0.00	0.00	100.00	0.00	0.00
100.00		7)361.99									
		51.71		100.00 — 51.71 — = 48.29							

TABLE XIII

Composite of Samples from Anvil Member
Silica Calculations

Percent Weight	Percent Silica	Weight of silica Per 1,000 grams Of ore	Cumulative Percent Weight	Cumulative Weight of silica In concentrate	Assay of Concentrate	100-Cumulative Percent of Weight	Weight of Silica in Tailings	Assay of Tailings	Silica Recovery
2.40	9.50	2.28	2.40	2.28	9.50	97.60	426.88	43.73	0.53
11.25	13.20	14.85	13.65	17.13	12.55	86.35	412.03	47.71	3.99
12.05	16.50	19.88	25.70	37.01	14.40	74.30	392.15	52.77	8.62
6.46	21.20	13.70	32.16	50.71	15.77	67.84	378.45	55.78	11.82
10.90	25.60	27.90	43.06	78.61	18.26	56.94	350.55	61.56	18.31
5.56	31.40	17.46	48.62	96.07	19.76	51.38	333.09	64.82	22.38
9.83	36.90	36.27	58.45	132.34	22.64	41.55	296.82	71.43	30.84
8.46	48.45	40.99	66.91	173.33	25.90	33.09	255.83	77.31	40.39
4.18	54.50	22.78	71.09	196.11	27.59	28.91	233.05	80.61	45.69
4.38	62.10	27.20	75.47	223.31	29.59	24.53	205.85	83.91	52.19
2.28	68.70	15.66	77.75	238.97	30.74	22.25	190.19	85.47	55.68
5.63	79.20	44.59	83.38	283.56	34.01	16.62	145.60	87.60	66.07
7.56	84.50	66.23	90.94	349.79	38.46	9.06	79.37	87.60	81.50
9.06	87.60	79.37	100.00	429.16	42.92	0.00	0.00	0.00	100.00
		429.16							

TABLE XIV

Liberation at 3 Mesh in Cross-Section of Anvil Member
Heads—35.86, 29.76, 26.13, 29.61, 26.85, and 27.36

	Concentrate				Tailings			
	Weight Recovery	% Fe	% SiO_2	Metal Recovery	Efficiency	% Fe	% SiO_2	% Total SiO_2
1	35	55.5	8.5	54.5	39.5	25.0	55.0	92.0
2	35	56.5	13.0	67.5	55.5	15.0	76.0	91.0
3	35	53.0	17.0	71.0	57.5	11.5	81.0	90.0
4	35	56.0	12.0	67.0	53.5	15.0	75.0	92.0
5	35	49.5	21.0	65.0	48.5	14.0	69.0	86.0
6	35	52.0	20.0	67.0	52.5	13.5	76.0	67.5
Average	35	53.75	15.25	65.33	51.17	15.67	72.00	89.75
Maximum Efficiency								
1	55.5	53.0	11.5	82.5	56.0	13.5	82.5	83.5
2	45.0	52.5	19.0	80.0	61.0	10.5	80.0	84.0
3	39.0	50.0	20.0	75.5	57.5	10.5	82.5	79.0
4	45.0	52.0	17.5	79.5	59.5	11.0	81.0	84.5
5	47.5	45.0	25.0	77.5	53.0	10.0	76.0	87.5
6	47.5	47.5	26.0	82.5	58.0	8.5	83.5	77.5
Average	46.58	50.00	19.83	79.58	57.50	10.67	80.92	82.67
10% Silica in Concentrate								
1	46.0	54.5	10.0	71.0	51.0	19.5	64.0	88.0
2	27.0	58.5	10.0	54.0	45.5	19.0	70.0	95.0
3	25.0	57.5	10.0	56.0	48.0	15.5	75.0	95.5
4	28.0	57.0	10.0	54.0	45.0	19.0	69.0	94.5
5	9.0	56.0	10.0	18.5	16.0	24.0	56.0	98.0
6	12.0	55.0	10.0	24.5	19.0	23.5	62.0	97.0
Average	24.50	56.41	10.0	46.33	37.41	20.08	66.00	94.66

TABLE XV

Comparison of Liberation at 3 Mesh in Samples from
Plymouth, Norrie and Anvil Members

	Concentrate					Tailings		
Member—	Weight Recovery	% Fe	% SiO_2	Metal Recovery	Efficiency	% Fe	% SiO_2	Total % SiO_2
Plymouth	35	57.5	12.0	66.0	55.0	16.5	72.5	92.0
Norrie	35	57.0	16.0	57.0	41.0	24.0	63.0	88.0
Anvil	35	54.5	17.0	54.0	37.0	27.0	57.0	86.5
At maximum efficiency								
Plymouth	47.5	54.5	16.0	80.0	62.5	11.0	81.0	84.5
Norrie	54.5	52.0	21.5	80.0	52.5	15.0	75.0	75.0
Anvil	58.0	49.5	22.0	80.0	45.5	16.5	70.5	70.0
10 per cent Silica in concentrate								
Plymouth	29.0	59.5	10.0	56.0	48.5	19.0	68.5	93.0
Norrie	13.0	60.0	10.0	25.0	22.0	31.0	51.0	97.0
Anvil	4.0	59.5	10.0	6.0	5.0	35.0	45.0	98.5

TABLE XVI

Comparative Jig Tests of a Sample from the Plymouth Member

Size—	Medium	Sp. Gr.	Product	% Wt.	% Fe	% Total Fe	Absolute Efficiency
+ 1 inch	Galena	2.3	Conc.	19.14	47.40	36.94	
			Midds	28.52	29.35	34.07	48.77
			Tils	52.34	13.60	28.99	
			Head	100.00	24.55	100.00	
	Iron Ore	1.9	Conc.	30.84	42.50	53.32	42.17
			Midds	17.69	22.30	16.04	
			Tails	51.47	14.60	30.64	
	Water	1.00	No action				
- 1″ + 2 Mesh	Galena	2.3	Conc.	35.19	52.90	60.99	63.40
			Midds	29.04	26.35	25.08	
			Tails	35.77	11.90	13.93	
			Heads	100.00	30.52	100.00	
	Iron Ore	1.8	Conc.	45.07	51.60	76.19	63.88
			Midds	15.60	19.80	10.12	
			Tails	39.33	10.60	13.69	
	Water	1.0	Conc.	30.57	51.50	51.58	40.54
			Midds	25.80	21.60	18.25	
			Tails	43.63	21.10	30.17	
- 2 + 3 Mesh	Galena	2.3	Cons.	49.29	59.55	72.24	76.14
			Midds	21.38	36.40	19.15	
			Tails	29.38	12.00	8.61	
			Heads	100.00	40.65	100.00	
	Iron Ore	1.65	Conc.	52.72	58.50	75.81	67.20
			Midds	21.09	24.80	12.98	
			Tails	26.19	17.40	11.21	
	Water	1.0	Conc.	38.30	54.50	51.36	42.33
			Midds	34.92	34.40	29.57	
			Tails	26.78	28.95	19.07	
- 3 + 10 Mesh	Galena	2.3	Conc.	53.94	63.60	77.58	79.88
			Midds	12.34	29.60	8.26	
			Tails	33.72	18.55	14.16	
			Heads	100.00	44.22	100.00	
	Iron Ore	1.6	Conc.	60.16	60.55	82.33	73.08
			Midds	10.10	26.00	5.97	
			Tails	29.74	17.40	11.70	
	Water	1.0	Conc.	52.67	55.70	66.34	48.39
			Midds	24.78	34.40	19.28	
			Tails	22.55	28.20	14.38	
			Summary of Jigging (Total Sample)				
+ 1″ to + 10 Mesh	Galena	2.3	Conc.	36.87	56.12	62.59	66.49
			Midds	25.20	29.30	22.33	
			Tails	37.93	13.15	15.08	
			Heads	100.00	33.05	100.00	
	Iron Ore		Conc.	45.26	53.40	73.06	
			Midds	16.72	22.32	11.32	61.93
			Tails	38.02	13.58	15.62	
	Water		Conc.	27.76	53.40	44.83	
			Midds	19.85	25.76	15.46	34.37
			Tails	28.91	25.47	22.27	
			Not separated	23.48	24.55	17.44	

TABLE XVII
Comparative Jig Tests of a Sample from the Norrie Member

Size—	Medium	Sp. Gr.	Product	% Wt.	% Fe	% Total Fe	Absolute Efficiency
+ 1 inch	Galena	3.3 to 2.7	Conc.	15.24	44.00	27.67	
			Midds	43.58	30.20	54.33	51.32
			Tails	41.18	10.60	18.00	
			Heads	100.00	24.33	100.00	
- 1 + 2 Mesh	Iron Ore	1.7	Conc.	32.77	46.80	50.26	
			Midds	38.35	30.30	38.16	50.61
			Tails	28.88	12.33	11.58	
			Heads	100.00	30.51	100.00	
- 2 + 3″	T. F. T. M.	4.8 to 2.7	Conc.	53.11	53.00	69.40	
			Midds	26.16	39.10	25.19	52.56
			Tails	20.73	10.59	5.41	
			Heads	100.00	40.57	100.00	
	Iron Ore	1.6	Conc.	51.09	53.77	67.89	
			Midds	32.97	33.75	27.22	53.77
			Tails	15.94	12.40	4.89	
	Water	1.0	Conc.	50.20	52.71	65.38	
			Midds	37.45	32.53	29.86	48.86
			Tails	12.35	15.60	4.76	
			Conc.	43.28	57.09	56.00	
- 3 + 6″	T. F. T. M.	4.8 to 2.7	Midds	35.47	46.70	37.69	57.47
			Tails	21.25	13.11	6.31	
			Heads	100.00	44.13	100.00	
	Iron Ore	1.6	Conc.	50.99	55.47	64.02	
			Midds	20.15	47.11	21.61	47.14
			Tails	28.86	22.00	14.37	
	Water	1.0	Conc.	49.44	55.69	62.32	
			Midds	19.11	47.91	20.85	46.46
			Tails	31,45	23.64	16.83	
- 6 + 10	T. F. T. M.	4.8 to 2.7	Conc.	60.13	57.55	74.60	
			Midds	16.20	47.40	16.56	53.67
			Tails	23.67	17.32	8.84	
			Heads	100.00	46.38	100.00	
	Iron Ore	1.8	Conc.	56.93	56.26	69.16	
			Midds	13.32	48.70	14.01	43.65
			Tails	29.75	26.20	16.83	
	Water	1.0	Conc.	62.19	55.41	74.41	
			Midds	21.80	36.50	17.19	42.15
			Tails	16.01	24.30	8.46	

TABLE XVIII
Comparative Jig Tests of a Sample from the Anvil Member

Size—	Medium	Sp. Gr.	Product	% Wt.	% Fe	% Total Fe	Absolute Efficiency
+ 1 inch	Galena	2.3	Conc.	23.14	49.00	38.82	
			Midds	16.75	32.00	18.34	41.40
			Tails	60.11	20.00	42.84	
			Heads	100.00	29.20	100.00	
	Iron Ore	1.9	Conc.	45.67	42.50	66.49	
			Midds	19.76	20.90	14.16	40.91
			Tails	34.57	16.40	19.35	
	Water	1.0	No action				
- 1″ + 2 Mesh	Galena	2.3	Conc.	35.13	43.25	51.60	
			Midds	24.03	31.80	25.93	42.34
			Tails	40.84	16.20	22.47	
			Heads	100.00	29.45	100.00	
	Iron Ore	1.9	Conc.	41.40	45.20	62.92	
			Midds	16.78	26.30	14.99	46.50
			Tails	42.21	15.40	22.09	
	Water	1.0	Conc.	29.29	39.20	39.00	
			Midds	21.49	32.30	23.57	25.64
			Tails	49.22	22.40	37.43	
- 2 + 3	Galena	2.3	Conc.	41.34	52.05	61.30	
			Midds	26.74	34.80	26.50	59.53
			Tails	31.92	13.50	12.20	
			Heads	100.00	35.15	100.00	
	Iron Ore	1.65	Conc.	41.34	53.30	62.66	
			Midds	15.38	26.35	11.58	50.07
			Tails	43.28	20.90	25.76	
	Water	1.0	Conc.	32.56	44.65	41.42	
			Midds	36.44	31.00	32.09	20.95
			Tails	31.00	30.00	26.49	
- 3 + 10	Galena	2.3	Conc.	51.41	56.50	72.94	
			Midds	11.59	26.60	7.77	58.64
			Tails	37.00	20.75	19.29	
			Heads	100.00	39.80	100.00	
	Iron Ore	1.65	Conc.	51.20	55.70	71.68	
			Midds	24.40	26.00	15.94	57.85
			Tails	24.40	20.20	12.38	
	Water	1.00	Conc.	35.35	52.65	46.80	
			Midds	31.58	33.00	26.20	30.68
			Tails	33.07	32.50	27.00	
			Summary of Jigging				
+ 1 to + 10	Galena	2.3	Conc.	36.33	49.12	55.38	
			Midds	19.77	31.54	19.41	44.78
			Tails	44.00	18.41	25.21	
			Heads	100.00	32.13	100.00	
	Iron Ore	1.9 to 1.65	Conc.	44.40	47.80	66.05	
			Midds	19.04	24.83	14.72	45.57
			Tails	36.56	16.90	19.23	
	Water	1.0	Conc.	23.62	44.23	32.52	
			Midds	19.66	32.33	19.78	25.69
			Tails	31.88	25.33	25.13	
			Unseparated	24.84	29.20	22.57	

ACKNOWLEDGMENTS

The research work here reported has been financed mainly by research grants by the State of Michigan. The federal government also has aided the projects by grants under the F. E. R. A. program, most of the work on the flowsheet part of the program being done under that arrangement.

A number of people have been employed on the project as assistants in the making of tests and analyses and also in calculating the results and plotting the data. Dr. O. W. Hothckiss, as President of the College, and Professor A. T. Sweet, as head of the department of metallurgy during this period, took an active interest in the progress of the work. Members of the faculty and the staffs of the metallurgy and other departments of the College have cooperated to make the tests successful. F. G. Pardee of the Conservation Department has also given all possible aid.

Special thanks are due to the officials of the various mining companies on the Gogebic Range for their assistance in securing samples and for the information about the formations that they have given

[Biographical]

THOMAS F. COLE

Thomas F. Cole, pioneer copper and iron mining executive and charter member of the Institute, was found dead in his garden near his home in Pasadena, California, June 3, 1939. His age was 75.

Mr. Cole was born at Cliff Mine, Keweenaw County, Michigan, and went to work shortly after his father was killed in a mine accident. At an early age he was employed in the railroad department of the Calumet & Hecla Company. He left the copper district to engage in mining in Michigan and at various times worked on all of the three Ranges. Before he was 21 he was clerk at the Chapin Mine, Iron Mountain, for the Schlesinger interests. At one time he was in charge of the Queen Group of mines, Negaunee, Michigan, and still later operated on the Gogebic Range.

When the Oliver Iron Mining Company was formed in 1901, Mr. Cole was chosen vice-president and a year later elevated to the presidency. He resigned this position in 1909 to devote all of his time to copper mining and had interests in Montana, Arizona, Michigan, and Nevada. He played a prominent part in building up what later became the Calumet & Arizona Mining Company, now a part of the Phelps Dodge organization. During the last years of his life he devoted most all of his time to his private gold mining interests in Nevada. This work is now being carried forward by his son, Fred, who, with a daughter, survive him.

Funeral services were held in Pasadena and burial took place there.

GEORGE HOWARD CROSBY, JR.

Goerge Howard Crosby, Jr., whose father is interested in mining on the Cuyuna Range, died November 3, 1939, at Garrison, Minnesota, at the age of 47.

Mr. Crosby was chief engineer in charge of his father's iron ore interests. He was educated at the MacKenzie Preparatory School at Dobbs Ferry, New York, and at Sheffield Scientific School at Yale University. He was well known throughout the Minnesota Ranges.

Surviving are his wife, three daughters, his father and a sister.

Burial took place November 18.

MARK ELLIOTT, SR.

Mark Elliott, Sr., at the time of death was manager of mines on the Marquette and Minnesota Ranges for the Jones & Laughlin Ore Co., and the Interstate Iron Company. He died October 31, 1939, in his home city, Virginia, Minnesota. His age was 74.

Mr. Elliott was born in Galena, Illinois, and went to Negaunee, Michigan, with his parents, and there he attended school. He started out in life as a telegraph messenger and graduated to operator, working at Cascade Junction, a short distance from Negaunee on the Chicago & Northwestern line. He left there to attend Lake Forest Academy in Illinois and later studied law for three years in Negaunee. He thought mining would be to his liking, and in this he proved to be correct. He accepted a position as clerk in the mine office of the Pittsburgh & Lake Angeline Mining Co., at Ishpeming, Michigan, later becoming chief clerk for the Volunteer Mining Company at Palmer, Michigan. He served for a time as clerk of Marquette County, Michigan, and next he went to Minnesota for the Jones & Laughlin interests, advancing to the position that he held when he died.

Burial took place in Marquette, Michigan, with a large group of mining men present to pay their last respects.

WALTER FITCH

Walter Fitch was born in England, January 20, 1854, and died in Eureka, Utah, April 23, 1939.

He came to this country at the age of 20 and began his mining career in Michigan in 1881, being connected with the Champion Iron Company, at Beacon, Michigan, for many years. He later went to the Calumet & Hecla Consolidated Co., Calumet, Michigan. He served as a member of the Board of Control of Michigan College of Mining and Technology from 1899 to 1904. He was reappointed to the board in 1907 but resigned the following year to engage in mining work in Utah. He was a charter member of the Institute and its president in 1902.

He was the organizer of the Chief Consolidated Mining Company at Eureka. His three sons, Cecil, Howard and Walter Jr., are engaged in mining work in Utah.

WILLIAM H. GALLAGHER, JR.

William H. Gallagher, Jr., chief mechanical and electrical engineer for Pickands, Mather & Co., died July 1, 1939, in Saginaw, Michigan, while on a vacation trip with his family. His home was in Duluth. His age was 57.

Mr. Gallagher was born in Adrian, Michigan, September 1, 1881, the son of an Episcopalian minister. After taking a course at Cornell University, he joined the staff of the Cleveland-Cliffs Iron Company, at Ishpeming, Michigan, leaving there in 1916 to spend a year in the Michigan Copper Country. From there he went to Duluth to enter the employ of the company in whose service he was when he died.

Mr. Gallagher was survived by Mrs. Gallagher and three daughters.

Burial took place in Duluth.

HENRY W. GRAMS

Henry W. Grams was born in LaCrosse, Wisconsin, June 23, 1884. At the age of six his family moved to Winona, Minnesota, where he attended the public schools. At the age of 12 he became a messenger boy employed by the Postal Telegraph Company at Winona. He was an operator in Winona at the age of 13. It was his proficiency with the telegraph key that brought Mr. Grams into connection and service with Paine-Webber & Co., stock and grain brokers. He joined the organization at Marquette, Michigan, in 1904, where he remained for only a month when he was transferred to Houghton, Michigan, and he became manager of that office in 1916. From there he was promoted to manager of the Duluth, Minnesota, Office of that Company in 1926, where he remained until 1937 when he retired because of ill health. He died on April 29, 1939, at Duluth, Minnesota,

He left his wife, two sons and two daughters.

HARRY C. HAMPTON

Harry C. Hampton was born February 1, 1867, and was employed as salesman by John A. Roebling's Sons Company for 31 years.

He traveled the Mid-Western States and later, for the greater portion of his term of service with the Company, he resided in Duluth and traveled the Iron Ranges and the Michigan Copper Country.

Mr. Hampton, upon reaching the age of 70, received a pension February 1, 1937. He passed away March 17, 1939, and was buried in St. Petersburg, Florida.

His wife survives him.

ARTHUR C. HANSEN

Arthur C. Hansen, whose age was 49 at the time of death on November 12, 1939, was assistant manager of the Witherbee Sherman iron ore properties at Mineville, New York. This Company is a subsidiary of the Republic Steel Corporation, with which he was connected for a number of years.

Mr. Hansen was born in Bessemer, Michigan, and was a, graduate mining engineer. He started work for the Republic Corporation at Bessemer and later transferred to the Cambria mine, Negaunee, Michigan, as assistant superintendent. When the late John E. Nelson went to Duluth, Minnesota, to manage all of the mines of the Republic Corporation, Mr. Hansen was named superintendent. He remained in this capacity until going to Mineville. His residence was at Port Edwards, New York.

His wife survived him. Burial took place in Bessemer.

HENRY R. HARRIS

Henry Robins Harris died at his residence in Marquette, Michigan, June 5, 1939, and was buried in the same city.

Mr. Harris was born in Beloit, Wisconsin, July 31, 1861, the son of a railroad man. At the age of 15, after a few years of common schooling, Mr. Harris began his long railroad career. He started as a clerk and operator on the Galena, Illinois, division of the Chicago & Northwestern Railroad and served in that capacity until 1880. From 1880 until 1889 he was clerk to the general superintendent of the Detroit, Lansing & Northern Railroad, which later became a part of the Pere Marquette. From 1889 ot 1890 he was rodman with a construction crew employed by the Chicago & West Michigan Road, which later also became a part of the Pere Marquette. He also served in various capacities with the Central Railway of Georgia, the Chicago & Northwestern, the Atchison, Topeka & Santa Fe Lines, and the Pennsylvania systems. In 1892 and 1893 he was superintendent of the Pullman Company and from the latter year until 1897 master of transportation for the Grand Rapids & Indiana Road.

It was in 1897, the year of the organization of the Lake Superior & Ishpeming Railway, when Mr. Harris went to Marquette to take charge of this line. He remained in that capacity until 1917, when he became vice-president and general manager, the position that he held at the time of death. The Lake Superior & Ishpeming is a subsidiary of the Cleveland Cliffs Iron Company.

Mr. Harris left his wife, two daughters and one son.

WILLIAM KELLY

William Kelly, a charter member of the Institute, died October 9, 1937, at the age of 83, at his home in Vulcan, Michigan.

He was born in the East and attended Columbia University, going to Iron Mountain, Michigan, after graduation. He was one of the few technical men in the district when he engaged in mining work, stepped up rapidly in his chosen profession and became head of the mining interests of the Penn Iron Mining Company on the Menominee Range. The Company was owned by the Cambria Steel Company until 1923, when the Bethlehem Steel Corporation took it over and from that date was operated by Pickands, Mather & Co.

In 1924 Mr. Kelly served as president of the American Institute of Mining & Metallurgical Engineers. This was a year after his retirement from active work. In 1898 he was honored with the presidency of the Lake Superior Mining Institute. For many years he was a member of the Board of Control of Michigan College of Mining and Technology and chairman of the board the greater part of that time.

His wife survived him. Burial took place in Vulcan.

DAVID HARTWELL LADD

David Hartwell Ladd was born in Milford, Oakland County, Michigan, in 1881, and was graduated from the Milford High School in 1899. The following year he attended the High School in Ann Arbor, Michigan, and in 1903 was graduated from the Michigan College of Mining and Technology with the degree of Bachelor of Science and Engineer of Mines.

From 1903 to 1909 Mr. Ladd was employed by the Michigan Smelting Company, Houghton, Michigan, serving as Assistant Engineer, Chief Engineer and Assistant Manager. From 1910 to 1912 he was Manager for the Wallaroo & Moonta Mining & Smelting Co., at Adelaide, Australia.

In 1913 Mr. Ladd went to Europe as Technical Sales Representative and later conducted exhaustive tests in the United States on Wallaroo copper and its adaptability for certain uses. From 1916 to 1918 he was Manager of the Sandusky Foundry & Machinery Co., Sandusky, Ohio, and was also associated with the Calorizing Corporation, at Detroit, Michigan.

In 1919 Mr. Ladd went to the Oklahoma and Kansas oil fields, remaining there until 1922. From 1923 to 1934 Mr. Ladd made his home in Birmingham, Michigan, where he was interested in real estate investments, appraisals and investigations. He continued his interest in mining and oil investigations and reports and was a Director and Consultant to the Development Mines Corporation, at Detroit.

Mr. Ladd died February 15, 1938.

DR. E. M. D. LIBBY

Dr. E. M. D. Libby, of Iron River, Michigan, died November 10, 1939, following injuries received in an automobile accident a few days previous. His age was 62.

Dr. Libby arrived in Iron Mountain in 1898, shortly after completing his medical studies. He was born in Chicago and spent his boyhood there. He attended Rush Medical College in that city.

He first went to Atkinson, then a boom community on the Paint River, and there he remained for two years. In 1900 he moved to Iron River and entered private practice. The Mercy Hospital was then the community's medical center. Later he converted his home into a hospital and it was the only one in the City of Iron River.

Music and wood working were Dr. Libby's hobbies and he maintained a complete woodworking plant in the basement of his home. Making and repairing violins kept him busy during his spare moments. He held membership in a number of medical societies.

He left his wife and five children.

Burial was at Iron River.

ALEXANDER F. MAITLAND

Alexander F. Maitland died July 2, 1939, at his home in Negaunee, Michigan. His parents were Alexander Maitland and Carrie V. (Sterling) Maitland. The elder Maitland was a mining man for many years and also served Michigan as lieutenant-governor.

Mr. Maitland was born in Negaunee and received his early education there. He later attended school in Racine, Wisconsin, and still later received a degree from the University of Michigan. He was head of the Maitland Mining Company for many years, this a family organization, owning an open pit mine in the Cascade district, Marquette Range. He was vice-president of the First National Bank, Negaunee.

His wife and two children survived him.

Burial took place in Negaunee.

CHARLES H. MUNGER

Charles H. Munger, of Duluth, Minnesota, retired manager of mines for Pickands, Mather & Co., died July 13, 1939, at the age of 83.

He was born in Akron, Ohio, April 10, 1856, and with his parents moved to Cleveland in 1871 and it was there that he received his education. When 18 years of age he entered the employ of the Atlantic & Great Western Railway Company and entered the iron business a few years later when he started to work for Tuttle, Masters & Co., in Cleveland.

After three years with the iron firm he went to the Marquette Range, where he was employed at the Tital and Beaufort mines. From this latter position he went to work for the Metropolitan Iron & Land Co., at Metropolitan, as clerk.

In 1904 Mr. Munger went to the Mesabi Range to open the Sellers mine, at Hibbing. This mine has since become a part of the great Hull-Rust-Mahoning open pit. Mr. Munger was the first superintendent of the Sellers. Later he went to Ironwood, Michigan, to become superintendent of the Norrie mine and still later took charge of the Sparta mine, at Gilbert, on the Mesabi Range.

In 1901 Mr. Munger became manager of mines for Pickands, Mather & Co., and served in that capacity until 1911, when he was succeded by William P. Chinn, who in turn was succeeded by A. D. Chisholm.

Mrs. Munger survived her husband. He also left a son and a daughter.

JOHN E. NELSON

John E. Nelson, manager of mines for the Republic Steel Corporation in the Lake Superior district, died March 29, 1939, at his home in Duluth. He was ill but a few hours.

Mr. Nelson was a native of Sweden and the date of his birth was September 25, 1878, making his age 60 at the time of death. He came to this country at the age of 11 years and received his schooling at Negaunee, Michigan. He started work at an early age in the Cambria Mine, Negaunee, and later was promoted to a place on the steam shovel crew. When there was an opening in the office he was made timekeeper. Not many years later he was chief accountant and the next step was his appointment as cashier.

In 1912; when Captain John Deacon died, Mr. Nelson was made superintendent of the Cambria, and he served in that capacity until 1923, when he was named superintendent of the Michigan mines of the Republic Company. In 1934, following the death of Francis Webb, he was promoted to the position of manager of all of the corporation's mines in the Lake Superior district.

Mr. Nelson served as alderman, supervisor and member of the Marquette County Road Commission.

Burial took place in Negaunee.

JOHN THORLEY REEDER

John Thorley Reeder, 80, died December 21, 1937, at his home on College Avenue, Houghton, following a long illness.

Mr. Reeder was born in Detroit on August 18, 1857, and was educated in the Detroit High School and Mayhew Business College in Detroit. Following his graduation he entered the office of the Detroit and Lake Superior Copper Works with his father who was a chief clerk for the concern for 40 years. When the Detroit Works closed in 1897 Mr. Reeder went to the Michigan Copper Country as chief clerk of the Osceola mine office at Osceola under the late Captain John Daniel. Later he was transferred to the Tamarack mine office and when the Calumet & Hecla Mining Co. gained control of the Bigelow interests, he was transferred to the Calumet & Hecla office as chief clerk and purchasing agent of the Tamarack, Osceola, Ahmeek and Isle Royale mines. He held this position from August, 1906 to May 31, 1919.

Mr. Reeder was a resident of Houghton for 30 years. He was active in the Keweenaw Historical Society of which he was a charter member. He later became president of the society and in 1913 was made a life member. He was a trustee of the First Congregational Church of Hancock, and was also a member of the Portage Township School Board on which he served from 1913 to July, 1922, when he resigned. He served as president of the board from 1914 up to the time of his retirement.

Mr. Reeder was a great lover of outdoor life and owned a famous collection of outdoor photos which included wild life, plants and flowers. He also acquired a fine collection of stamps, coins, Indian relics and minerals.

Surviving are his wife and two daughters and one son.

JAMES H. ROUGH

James H. Rough, of Negaunee, died suddenly May 20, 1937, in his home.

Mr. Rough was born in Ontonagon, Michigan, December 27, 1859, and the death of his father in a mine accident compelled him to seek employment at an early age. He was engaged in mining work all of his life. During the greater part of his active life he was in the employ of the Cleveland-Cliffs Iron Company and received a pension from this Company. He was at one time mining captain at the Cliffs Shaft Mine, Ishpeming, and was promoted from there to superintendent of underground operations, covering all of the Company's properties. He served for a few years as inspector of mines for Marquette County.

In 1928 Mr. and Mrs. Rough celebrated their golden wedding anniversary. A son, James, Jr., a mining engineer on the Mesabi Range, survived him.

Burial took place in Negaunee.

ARTHUR E. SEAMAN

Arthur E. Seaman, professor emeritus of mineralogy and geology, at Michigan College of Mining and Technology, died July 10, 1937, at the home of a daughter in Columbus, Ohio.

He was born at Casnovia, Newaygo County, Michigan, December 29, 1858, making his age 79 at the time of death. He entered the Michigan Mining School in 1888 and received a Bachelor of Science degree after teaching four years. In 1916 he received the degree of E. M., retiring in 1928. He was then made curator of the mineralogy and geology museum at the college. In June, 1932, the museum was given the name "A. E. Seaman Museum."

Professor Seaman, an authority on pre-Cambrian geology, conducted much research along these lines. His knowledge of the geology of the Lake Superior district was thorough.

He left a daughter and a son, Wyllys, associated professor of mineralogy and geology at the same school his father served so long.

Burial took place in Marquette, Michigan, where Professor Seaman spent a few years prior to going to Houghton, Michigan.

RAYMOND W. SEELYE

Raymond W. Seelye, who was a native of Minnesota and where he was engaged in mining work, died February 17, 1939, in Seattle, Wash., where he had lived for 20 years. He was a mining engineer.

He left his wife, and a son and two daughters. One of the daughters, Mrs. P. E. Durham, resides in Sault Ste. Marie, Ontario.

Burial was made in Seattle.

ROBERT H. SHIELDS

Robert H. Shields was born in Franklin Township May 22, 1861. He died in Houghton April 4, 1937. His funeral was held in Houghton April 7, 1937, with burial in Forest Hill Cemetery. His mining career, briefly, was as follows: Clerk Centennial Mining Company, 1889-1892. Clerk of Arcadian Copper Company, 1898-1899. Superintendent Arcadian Copper Company, 1899-1900. With reorganization as Arcadian Consolidated Mining Company in 1900 he became president and served until dissolution of Company early in 1937.

WILLIS LAW TINKER

Willis Law Tinker, secretary of the Lake Superior Iron Ore Association, died of a heart attack at his home in Hudson, Ohio, June 5, at the age of 53. He had been in ill health for over a year.

Mr. Tinker was born at Gates Mills, near Cleveland, and attended schools in that vicinity. He first became connected with the Iron Ore Association in 1906, which was six years after its organization, and was elected its secretary in 1911, holding that position continuously thereafter until his death. He was well known throughout the iron and steel industry as the statistical authority on the iron ore trade of the Great Lakes.

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